Proceeding of the 26th International Symposium on Mine Planning and Equipment Selection

Luleå, Sweden, 29-31 August 2017

Mine Planning and Equipment Selection (MPES 2017)

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Preface

The 26th International Symposium on Mine Planning & Equipment Selection (MPES2017) is to be held in Luleå, Sweden, during August 29-31, 2017. The MPES international Symposium has established, evolved itself and developed to provide an excellent platform where industry, academic and consultants can share knowledge, ideas and solutions relevant for a modern day mining industry.

As in the past, MPES 2017 provides a forum for the review and exchange of information on new and emerging technologies in mining engineering and equipment selection. This symposium addresses most aspects of surface and underground mining covering issues and challenges related to mining methods, equipment selection, operation & maintenance of mining equipment’s and mine environment, etc. A broad and distinguished international committee together with Division of Operation and Maintenance Engineering and Division of Mining and Rock Mechanics of Luleå University of Technology has identified the themes for this symposium.

These proceedings contain peer-reviewed papers written by experts from industry and academia from 22 countries. The papers cover emerging research and problem solving. The MPES2017 contains fourteen sessions; the sessions cover the following topics:

- Economic and Technical Feasibility Studies, Reserve Estimation, Mine Development Case Studies
- Design, Planning and Optimization of Surface and Underground Mines including Transition from surface to underground mining
- Drilling, Blasting, Tunneling and Excavation Engineering
- Mining Equipment Selection
- Mine Automation and Information Technology, Industrial Internet
- Reliability and Maintenance of Mining Machinery
- Mine Production Management
- Mining: Health, Safety and Environment
- Rock Mechanics and Geotechnical Applications
- Environmental Issues in Surface and Underground Mining of Metalliferous, Coal, and Industrial Minerals.
- Reliability of Waste Containment Structures, Tailings Treatment, Recycle, Disposal, and Decommissioning

The sessions for the different themes are organized to maintain unity within a specific area of mining technology. This is to make it easier for the participants to take part in the symposium activities of special interest to them.

The Division of Operation and Maintenance Engineering, Luleå University of Technology, Luleå is the host of this Symposium and we are thankful to faculty and administrative staff for their support. We would like to thank the authors for their valuable contribution and the reviewers and the editorial committee for their reviewing, support and editorial work for the proceedings. We would also like to thank the staff of Luleå University of Technology Press for meeting the time limit and doing an excellent job.

Behzad Ghodrati
Uday Kumar
Håkan Schunnesson

Editors

Luleå, Sweden, August 29, 2017
Foreword

The International Symposium on Mine Planning and Equipment Selection (MPES) was started more than twenty five years ago. Since then it has been held regularly becoming an internationally recognized event committed to technology transfer. It has been held in Turkey, Greece, Canada, Kazakhstan, Australia, Czech Republic, Brazil, India, Australia, China, Ukraine, Poland, Germany, Italy, and South Africa. MPES is being held in Sweden for the first time.

The basic aim of MPES 2017 is to contribute to the development of highly productive methods and technologies for the various segments of the mining and mineral processing industries. Theme of the 2017 symposium is “Innovations in Mining”. Major topics to be covered at MPES 2017 are: Data Collection and Modeling; State of the Art Practices; Mineral Resource and Mineral Reserve Estimation and Reporting; Economic and Technical Feasibility Studies; Mine Development Case Studies; Design, Planning and Optimization of Surface and Underground Mines; Transition from Surface to Underground Mining; Rock Mechanics and Geotechnical Applications; Mining Equipment: Selection, Operation, Control, Monitoring and Optimization; Mechanization and Automation of Mining Processes; Application of Information Technology; Productivity and Competitiveness of Mining Operations; Sustainability: Improving Health, Safety and Environmental Practice and Performance; Mine Closure and Rehabilitation; and others.

MPES derives its strength from the coalition of various worldwide institutions. MPES 2017 is supported by a number of organizations. To be noted are Lulea University of Technology, Sweden; Department of Mining, Metallurgical and Materials Engineering, Universite Laval; China University of Mining and Technology, Beijing; The National Technical University of Athens, Greece (NTUA); Dipartimento di Geoingegneria e Tecnologie Ambientali, Universita degli Studi di Cagliari, Italy; Western Australian School of Mines, Curtin University of Technology, Australia; National Mining University of Ukraine, Dnipropetrovsk; International Journal of Mining, Reclamation and Environment; American Society for Mining and Reclamation; School of Mining and Petroleum Engineering, University of Alberta, Edmonton, Canada; Faculty of Mining and Geology, VSB - Technical University, Ostrava, Czech Republic; Hokkaido University, Mineral Resources Engineering Department, Japan; Faculty of Geoengineering, Mining and Geology, Wroclaw University of Technology, Poland; Department of Mining, Metals and Materials, McGill University; Department of Energy and Geo-Environmental Engineering, The Pennsylvania State University; DIGET-Politecnico di Torino, Italy; School of Chemical, Environmental and Mining Engineering University of Nottingham, UK. Mining Engineering Department, University of British Columbia; Middle East Technical University Mining Engineering Department, Istanbul, Turkey; Kyushu University, Fukuoka, Japan; University of Cagliari, Italy; Institute of Land Reclamation and Ecological Restoration, China University of Mining and Technology, Beijing, China; Columbia University, USA; Virginia Polytechnic Institute and State University; and Division of Mining & Resources Engineering, Monash University, Australia.

The organization and success of such a symposium is due mainly to the tireless efforts of many individuals, authors included. All members of the Organizing Committees and Conference Chairpersons have contributed greatly. The support of our plenary session, invited speakers, and co-chairs is gratefully acknowledged. In addition, recognition is accorded to co-editors and Chair Person of this symposium, Professor Uday Kumar, who together with his local organizing committee made MPES 2017 a success. My greatest appreciation goes to Dr. Behzad Ghodrati who has worked very hard to support Professor Uday
Kumar by taking control of MPES matters during his absence. I also wish to acknowledge the contribution of Mohini (Mona) Singhal (my wife) who has been involved with MPES since its inception. She is a committee member of the MPES organisation and is an associate editor of the International Journal of Mining, Reclamation and Environment. She shares my workload and maintains the continuity of our work in my absence. We both are committed to make each symposium a successful one.

As the international chair and founder of this series of symposia, I would like to recognise the guidance and support of the symposium chairs from MPES 2011, 2012, 2013, 2015, and 2016 who are present.

This symposium provides a forum for the presentation, discussion and debate of state-of-the-art and emerging technologies in the field of mining. Authors from over 20 countries with backgrounds in computer sciences, mining engineering, research, technology and management representing government, industry and academia concerned with mining and mineral production have contributed to these proceedings. The contents of this volume of proceedings will be of interest to engineers, scientists, consultants and government personnel who are responsible for dealing with the development and application of innovative technologies to the minerals industries.

**Raj K. Singhal**
Chair, International Organizing Committee

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MPES2017

Session 1 – Mine Design & Planning I

13:00-15:00

August 29, 2017
Innovation and Modernization of the South African Mining Industry

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Abstract— In June 2016, the Southern Africa Institute of Mining and Metallurgy (SAIMM) held a colloquium in Johannesburg South Africa of which the the purpose was to create a dialogue between industry, government, research institutes and academia in the area of new technology and innovation. One of the key findings presented at the colloquium was the Chamber of Mines (COM) study indicating that the South African gold reserves dependent upon conventional mining methods will be depleted by 2031 with Platinum Group Metals (PGM) reserves signaling a similar fate.

The South African industry considers mechanization, automation and continuous mining operations as critical to extend gold reserves beyond 2046 or in the case of PGMs to 2042. To date, mechanization, automation and continuous mining in the gold and platinum environments has had limited success in the South African mining industry, generally achieving lower than planned productivity at higher than budgeted costs. Thus, the development of innovative mining systems based on new technology is critical for the continuance of the South African mining industry.

This paper discusses the strategies and research initiatives required to support a mining industry beyond 2040 and investigates the role that innovation will play in sustaining a mining industry that is currently in decline.

Keywords: South African mining industry, mining research and innovation, new technology

1. Introduction

Mining in general, and South African hard rock mining in particular, is associated with risk. Narrow reef mining entails risk from fall of ground, seismic events, fires, and other hazards to the mineworker via the actual mining process. However, another risk which is influencing the South African mining industry is the every apparent risk of mine closure due to economic factors or social and statutory constraints. These concerns affect the entire spectrum of employment in the mining industry from the general labourer to top management; from students just entering university studies in the field of mining to mineworkers with decades of experience; from surface operations exploiting coal and iron ore to underground gold, PGMs, chrome, and manganese operations.

In the past, labour was inexpensive and plentiful, and combined with the narrow nature of the orebodies, this led to labour-intensive mining methods (Figure 1). Many of the deposits were shallow and conducive to short travelling distances and high productivity. This is no longer the case – the reality is medium- to deep-level mining that involves travelling through two or three shaft systems with working face times often less than four hours per shift.

The industry has attempted to mechanize the South African narrow-reef environment – in the late 20th century in the gold mines and since the 21st century for the platinum industry. The former initiative largely failing due to the increase in dilution and the subsequent reduction of the head grade and the current initiative making some progress but not conclusively successful (e.g. Lonmin’s experience). The implementation of mechanization in the narrow-reef environment requires favourable geology with up to 50% of the mineral resources being unsuitable for mechanized mining systems (Rupprecht and Rapson, 2004) and the mechanized mining systems requiring a considerable amount of capital.

These concerns are in context with a labour force that is highly unionized with demands for significant increases in wages; a regulatory environment that is often viewed as over-zealous in regard to safety stoppages and social and community responsibility; and a business environment lacking sufficient power infrastructure, where many new mines are coming on line (for example the eastern and northern limbs of the Bushveld Complex) combined with double digit annual increases for electricity tariff and diesel for a number of years (Figure 2). All of these issues are combined with concerns such as the future availability of water and changing of the environmental regulations.

The view of the authors is that the South Africa mining industry, as we now know it will be finished within the next 15 to 20 years if significant changes are not made with immediate
effect. The South African mind-set must change – unfortunately, too many affected people still consider the South African mining industry as an industry provided with cheap labour, noting that labour equates to 50% to 60% of the on-mine mining costs.

Figures 1 and Figure 2 illustrate the decline of the South African mining industry since 1980. As can be seen, the gold industry accounted for over 60% of all mining-related employment in the early and mid-1980s. Currently, gold represents just over 20% of the employment for the South African mineral industry, with PGMs employing the larger proportion of mineworkers. More importantly, Figure 3 demonstrates the decline of the industry with direct mining-related employment steadily decreasing from 800 000 in the 1980s to 495 000 in 2014.

Figures 4 and Figure 5 depict the results of a recent South African Chamber of Mines study and presented by Turner (2016) indicating the tonnages to be mined from South African gold and platinum mining companies based from conventional, mechanised and continuous (24/7 mechanised) mining operations. These figures indicate that by 2025 gold tonnage sourced from conventional mining will largely be depleted; similarly platinum tonnages sourced from conventional drill and blast operations will be exhausted in 2015. The report suggests that through the continued implementation of mechanised mining, these reserves, both gold and PGMs, can extend beyond 2035 and with continuous operations mining should be able to continue beyond 2040.
The above discussion has painted sombre picture of the mining industry. Regrettably this picture is further reinforced by the fact that productivity (gold and PGMs), as shown in Figure 6, has reduced to its 1990 levels, indicating a stagnant (2000) and even negative trend (2006) in productivity.

The picture is clear – the South African mining industry, even with the commodity boom of the early 21st century, is shrinking and does not present a picture of health or vibrancy. The data referred to does not take into account the major challenges of 2015 and 2016, where further mine closures and retrenchments have taken place in many of the South African mining commodities, for example iron ore, PGMs, manganese, and coal.

Since the late 1980s mining research has been sporadic with much work having been done in safety and health issues (SIMRAC) with a brief period when the DeepMine Collaborative Research Programme produced over 100 reports to support deep-level gold mining. Although the DeepMine research programme strove to transfer the knowledge produced – ‘research reports being the primary product of DEEPMINE’ (Durrheim and Diering, 2002) – the author has observed numerous examples of mine studies and other research work duplicating work already conducted by DeepMine or the other collaborative research programmes. Regrettably, many of these mines are owned by companies that did not belonged to the collaborativeresearch programmes, and in other cases participating groups have failed to fully disseminate this knowledge to all operations and therefore research work is often being repeated.

Going forward, any new research initiative must take stock of the current research knowledge base to ensure fundamental work is not duplicated. Furthermore, critical to the success of research initiatives is a comprehensive method of distributing the results and finding of all research.
2.2 Mechanization

Narrow reef mechanization is critical and one only needs to review the six SAIMM International Platinum Conferences from 2004 until 2014 to understand the volume of work that has gone into introducing mechanization to the southern African platinum industry (Figure 7). One cannot raise the issue of mechanization in the platinum industry without discussing Lonmin and its 2004 vision to achieve 50% of reef production and 100% for development metres by 2010 using mechanized mining methods (Webber et al., 2010). The actual level of mechanization achieved in 2010 was only 20%. The Lonmin mechanization programme failed to meet its target with equipment underperforming, difficulties in regard to labour and supervision, and mining dilution higher than planned due to the size of on-reef development and the inability of the extra-low profile (XLP) equipment to handle rolls in the reef. Thus, Lonmin changed its strategy, deciding that mechanized mining utilizing the XLP equipment needed to be proved on a smaller scale before implementing it mine-wide. Mechanization is a non-debatable area requiring further research, innovation, and development. Probably the biggest shortcoming with the development of mechanization has been the inability of the industry to manage this in a collaborative manner. There has been too much duplication between mining groups, a lack of proper documentation and a reliance on original equipment manufacturers to develop the equipment. A strong requirement is for the local manufacturing of mobile equipment, focused primarily on the southern African underground mining environment. There is a need to apply relevant industrial engineering principles to the mining industry while still being cognisant that mining is not a factory as inputs (i.e. the mining work environment) change on a regular basis, requiring workers and supervisors to readily adapt as and when required. Importantly, primary research must also be conducted by dedicated researchers having the proper experience and skills – this in itself offers challenges to the required change process.

2.2 Continuous Operations

Continuous mining (24/7 mechanised mining) is an initiative in narrow vein stoping to achieve increased production by applying low amounts of energy to the drill hole to break rock (Figure 8). If successful, continuous mining potentially offers a quantum leap in face advance when compared with conventional mining methods. Thus, the success of a continuous mining system could have a huge impact on the South African mining industry.

Continuous mining offers the possibility of stoping continuously throughout the 24 hours, and monthly face advance in excess of 40 m. Traditionally research has focused on the actual breaking of the blast hole (i.e. the micro process) and achieving acceptable emission levels. At this stage continuous mining appears to be applicable to stope layouts including down dip, up dip, and breast, with mining face lengths between 30 m and 90 m. To ensure accuracy, holes should be drilled utilizing stope drill rigs with low profile dozers to remove the broken rock from the stope face. Research on continuous mining has largely been dormant since around 2003. At the time continuous mining failed to consistently and repeatedly break rock utilizing low energy charges. Further concerns were the levels of noxious fumes and dust generated by low energy charges. If the above technical issues can be resolved, there remains the issue of the feasibility of continuous mining as a rock breaking system, as indications are that the process is considerably more expensive than conventional blasting.
3. Conclusions
The need for change has been around for a long time combined. The COMRO conducted research on proposed change in the 1980s. Hustralid and Nilsson in 1998 discussed technology development as a function of time, with Pukkila and Sarkka (2001) illustrating the development steps towards the intelligent mine. In 1998, the South African gold mining industry (AngloGold, GoldFields and DRD) established the DeepMine Research Programme, which was followed by Futuremine, Platmine and CoalTech 2020 in the early 2000’s. In 1998, Bobby Godsell, the Chief Executive Officer of AngloGold commented on the transformation of mine labour: ‘Work structures have remained remarkably unchanged for many decades because of static technology, the impact of apartheid, and the previously closed nature - in times past - of the South African economy.’ (Godsell, 1998).

In 2001, the implementation of new technology is explained in MacFarlane’s SAIMM Journal paper: ‘The implementation of new technology in southern African mines: pain or panacea’ (MacFarlane, 2001). And some 15 years on, the mining industry is at a crossroads either to develop solutions to extend the life of mines beyond 2040 or to continue in its current path resulting in the South African mining transforming into a insignificant producer of gold and PGM metals.

This paper discussed the justification for the mining industry to make significant advancements in the way it conducts mining. The industry must ‘leapfrog’ the current mind-set of conventional mining. Advances in technology, namely mechanization and continuous mining, are proposed as a means to extend South African gold and platinum mineral reserves beyond 2040.

Fundamental to the South African mining industry is the development of mechanised and continuous mining. Crucial to this research is the basic understanding that meaningful research takes time and it may take several years before the industry sees applicable results. A good analogy may be the growing of an olive orchard, which may take five to 12 years before first fruiting. Key to the success will be the transfer of knowledge to the mining industry.

In conclusion, the mining industry must embrace change and recognize the need for meaningful change. Only visionary thinking, hard work, and commitment from all interested and affected parties will enable the industry to navigate through these troubled times and create a sustainable industry that will continue past 2030.

References


Risk Management of Locating Primary Crusher at the Edge of Open-Pit Mines under Price Uncertainty

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Abstract—Primary crusher is one of main open-pit mine’s facility whose location is very important from an economic point of view. The location mainly depends on the pit limits and haulage costs. Due to rising fuel, tire and maintenance costs in line with extracting the ore from deeper earth by open pit mining, economic conditions have forced the mine designer to minimize the truck haulage distance by bringing the primary crusher closer to the pit and, thus, utilize conveyors to perform a larger proportion of the ore transport requirements. The ultimate pit limit is highly subjective to the commodity price. An overestimation of pit limit causes an increase in haulage costs, while pit limit underestimation will impose relocation cost or leave tied up ore. In this paper, a quantitative simulation-based approach is developed for managing the risk raised from locating the primary crusher at the edge of open-pit mine because of price uncertainty. A hypothetical 2D geological block model was used to evaluate the crusher location. The results show that the edge of pit calculated by average price has the lowest risk level. Sensitivity analysis of truck haulage and crusher relocation cost shows the robustness of the results in the presence of 20% increase or decrease in the parameters.

Keywords: primary crusher, risk management, open pit mine, probable pits, price uncertainty

1. Introduction

Site selection is increasingly recognized to play a significant role in mine design particularly in open-pit mines where ultimate pit limit has a vital effect on facility’s location. The solution of the final pit limit problem is usually used to justify the point where the mining operation can progress economically and it is also a guide for the mine planner to locate the mine site facilities. In addition to having a logical relationship, the facilities must be located as near the pit as possible to minimize the transportation costs. This is the most popular strategy of the mine designer to be able to locate the facilities close to the pit, but this desire is always constrained by topographical, geotechnical and environmental conditions. The Aitik copper mine (Sweden's largest open-pit copper mine) is a good example to illustrate this statement. Figure 1 shows two aerial photos of the mine and industrial site placed at the pit rim.

Figure 1. Aerial photos of the Aitik copper mine and its industrial site location

The primary crusher is the most important facility for which decision-making related to its location has a strategic nature. The operational experiences showed that the transportation cost would be managed significantly better following the proper decision in this regards. The crusher location dictates the size of the truck fleet and the length of the conveyor transportation path and its other design criteria such as belt width and conveyor speed. Since material handling costs are usually the largest single component (almost half) of the mining cost [1, 2], a well-designed location of primary crusher plays a very important and critical role, affecting the expense of the total operation. This is why the crusher location has a strategic essence. The term primary crusher refers to the in-ground crushers or rim mounted first crushers that are usually located external to the pit at one place for the entire life of mine and are not intended to relocate.
There are a lot of investigations which highlight the importance of this location problem. In more than 447 mines around the world, the primary crushers have been moved into the pits to decrease the transportation costs [3]. The number of these transitions today (moving the rim mounted crusher into the pit) is more than twice the number of year 2003. From the year 1956 up to 2003, just 200 transitions had occurred [4]. When the mines get deeper, the haulage distance and the haulage cost begin to rise in a way that the marginal ore blocks near the pit limit are going to stay outside of the pit (Inside the pit limit, mining operations are economical and outside the pit limit, open pit mining is not likely to be profitable). In most cases, the crushed materials are moved by conveyors. The operating cost of conveying is about one-fourth to one-third of the trucking cost per kilometer [5, 6]. In addition, conveyer mode of transport does not react to increase in the length of haul as dramatically as the truck transport does. That is why, in order to reduce costs associated with transportation, moving the crusher inside the deep pits is advisable when the mine becomes deeper. However, new open-pit mining operations need to place the crushing plant as near the pit as possible to minimize the overall haulage cost (i.e. truck and conveyor cost). We call it the pit edge or pit rim.

Looking through this lens at the problem leads us to conclude that the primary crusher location follows the pit limit more than the other mine facilities. On the other hand, the pit limit is strictly pursuing the commodity price. Hence, this embeds an uncertainty into the primary crusher location problem. It is this need that this paper aims to address. An overestimation of the pit limit causes an increase in haulage costs, whereas pit limit underestimation will impose relocation cost or leave tied up ore. Therefore, there is a need for risk management to evaluate different crusher locations. Figure 2 shows both cross-sectional and plan views of an open pit with probable future expansions, which cause multiple candidates of primary crusher location at the rim of each probable expansion. The question is: which location is the best in the entire life of mine when considering price uncertainty?

Literature review shows that the focus on primary crusher location problem is more conducted on in-pit crushers [7-12] and the risk associated with the crusher location has never been considered. The mine’s facilities positioning studies are dominantly belong to the processing plant, tailing dam and waste dump [13-23]. Considering these, there is a particular need for a sound approach to managing the risk of primary crusher location.

Figure 2. A schematic cross-sectional and plan views of an open pit and its probable future expansions as well as candidate primary crusher locations

This paper provides a new quantitative simulation-based risk management approach to determine the primary crusher location because of commodity price uncertainty. Firstly the framework for the proposed method is explained and then it is applied in a 2D block model step by step. It is assumed that topography is flat and smooth to permit the mine designer to locate the primary crusher at the pit rim exactly. The environmental aspects are not concerned here.

2. Proposed methodology

Risk is often expressed as the effect of uncertainty on the objectives. The effect is any deviation from the expected either positively or negatively. In other words, risk is defined in terms of a combination of the consequences of an event and the associated likelihood of occurrence [24, 25]. In this research, risk is the chance of price fluctuation that will have negative impacts on mine profitability. These impacts are: increased haulage cost, crusher relocation or sacrificing the tied up ore. Risk management is the coordinated activities to direct and control the risk. The main reason for conducting a risk management is to support decision-making; so that the system has the lowest vulnerability against the future unexpected events [26]. The risk management procedure proposed in this paper consists of six steps: establishing the context, risk identification, risk analysis, risk evaluation, As Low As Reasonably Practicable (ALARP) level and risk treatment associated with the crusher location at the edge of open-pit mine. The main individual components of the proposed simulation-based risk management process are
depicted in Figure 3 and explained in detail in the next subsections.

2.1. Establishing the context
The first step of a risk management is to define the aims and objectives of the analysis. The objective of this risk analysis is to provide a well-organized decision-making tool based on risk management principle to locate the primary crusher in open pit mining. In addition to the objective, any limitation to the scope of the analysis must be taken into consideration, such as lack of available resources, time limit and lack of data and information when formulating the objectives. Furthermore, the environment, operating conditions and the rules governing the system must also be determined.

2.2. Risk assessment
Risk assessment is the central and execution part of the risk management process, which purposes to establish a proactive safety strategy by investigating potential risks. In our research, risk assessment is a process of assessing the likelihood of the causing or contributing to any change in the size of the pit; and the consequence of the pit size change on primary crusher location.

In essence, risk assessment includes three steps: risk identification, risk analysis and risk evaluation. That means, to assess a risk, potential sources of harm should be identified first and then the likelihood and consequence of them occurring should be estimated to analyze the risk. Thereafter, risk should be evaluated which means comparing the estimated risks (level of risk) against risk criteria (predetermined standards or target risk levels) to determine the significance of risk [26].

2.3. Risk identification
Once the context of the risk management is properly established, the next step in the risk management process is to identify the hazards and the situations that have the potential to cause harm or losses, sometimes called ‘unwanted events’ [27]. A hazard is defined as a source of potential harm, which can be a risk source [24]. In this paper, the commodity price uncertainty as the main hazardous element of mine design is evaluated. In fact, decreasing (DP) or increasing (IP) the commodity price often causes a risk on crusher location. With the aim of identifying the price behavior, the historical data are gathered and assessed to be used in the risk analysis process.

Figure 3. The key elements of the proposed risk management methodology
2.4. Risk analysis

Risk analysis includes analyzing the magnitude of the risk that may arise from the unwanted event. The objective of a risk analysis is to describe the risk that is to present an informative risk picture [26], which can be illustrated in the form of a bow-tie diagram. In another sense, providing helpful data to assist in the evaluation and treatment of risks is the main purpose of risk analysis. A comprehensive examination of the hazards can be done as the overall evaluation of interactions between different parts of a single bow-tie diagram or consideration of a range of bow-tie diagrams together. A bow-tie is a simple and effective diagram that is shaped like a bow tie and visualizes the risk you are dealing with in just one easy to understand picture.

The diagram clearly displays the links between the potential causes (hazards), preventative and mitigating controls and consequences of a major initiating event. Figure 4 shows the simplified bow-tie diagram resulted from our research on primary crusher location because of price uncertainty. The hazards and associated preventative controls are on the left-hand side of the bow-tie flowing in a center "initiating event" and then flowing out to the mitigating controls and significant consequences. In Figure 4, the initiating event is changing the size of the pit limit which is mainly followed by decreasing (DP) or increasing (IP) the commodity price. As it was previously mentioned, DP or IP are possible hazards and crusher relocation, leaving the tied up ore and increasing the haulage cost are the important consequences. Along the lines feeding into and out of the center are vertical ellipses that represent controls or barriers that either prevent the initiating event (left side) or the consequence (right side). Topography, ore-body shape, stripping ratio and type of mineral are the main barriers that control the effect of DP and IP on the pit limit while not to locate the crusher at the pit rim is the only barrier that mitigates the effect of the pit size on the three consequences.

Risk is analyzed by combining estimates of consequences and likelihood. In other words, the level of risk equals likelihood multiply by consequences of an occurrence. Risk analysis may be undertaken to various degrees of refinement depending upon the risk information and data available. The analysis may be qualitative, semi-quantitative or quantitative or a combination of these, depending on the circumstances [26]. Generally, in our proposed methodology a quantitative simulation-based risk management is examined. The concept of probable pits is used for calculating the probabilities; also, crusher relocation cost and truck haulage cost increases are considered as the main consequences.

2.4.1. Probable pits generation

The ultimate pit limit (UPL) defines an economic excavation limit of a given deposit beyond which the commodity value will not support the costs. It identifies which block should be left in the ground and which one should be mined. In other words, the optimum final pit limit will guarantee maximum value gained from a given input geological block model and a given set of economic condition whilst satisfying the operational requirement of safe wall slopes. With an increase in price, the pit would expand in size assuming all other factors remained constant. The inverse is obviously also true. By knowing this fact, some probable UPLs can be construed for a given deposit where each one has a distinct probability of occurrence. Figure 5 shows the proposed framework in which the probable UPLs are generated during a simulation. The historical commodity price data and geological block model were identified previously in the first step.
Afterward, the cumulative distribution function (CDF) is created from the historical commodity price data. Next, the price scenarios are randomly generated using Monte Carlo Technique (MCT). The block economic model is created from the geological block model and takes into account production and processing costs and price scenario. If the metal content of a block is zero or not enough to extract then block value will be negative, thus, this block is considered a waste block otherwise it is considered an ore block. Each ore and waste block can take one of the two following values respectively:

\[
\text{BEV}_{\text{ore}} = M \left[ 10gy(p(r) - s) - c_0 - c_w \right] \quad g \geq g_c \quad (1)
\]

\[
\text{BEV}_{\text{waste}} = -c_w M \quad g \leq g_c \quad (2)
\]

Where, \( g \) is average block grade in percent, \( M \) is block tonnage, \( y \) is recovery in percent, \( p(r) \) is commodity price scenario ($/kg cu), \( s \) is smelting and refining cost ($/ kg cu), \( c_0 \) is processing cost, \( c_w \) is mining cost ($/ ton rock), and \( g_c \) demonstrates the break-even cut-off grade. Subsequently, the UPL needs to be identified subject to the maximum profit through the set of Equations 3 through 11. Equation 3 is the objective function, used to maximize the revenue. Equations 4-6 define the slope requirements and the Equation 7 sets the binary condition for the variables.

\[
\max \sum_i \sum_j c_{ij} x_{ij} \quad \text{max} \sum_i \sum_j c_{ij} x_{ij} \quad (3)
\]

\[
x_{ij} - x_{i-1,j} - 1 \leq 0 \quad (4)
\]

\[
x_{ij} - x_{i,j-1} \leq 0 \quad (5)
\]

\[
x_{ij} - x_{i-1,j-1} - 1 \leq 0 \quad (6)
\]

\[
x_{ij} = 1 \text{ or } 0 \quad (7)
\]

Where \( c_{ij} \) is the block economic value and \( x_{ij} \) is a binary variable that gives 1 if the block located at row \( i \) and column \( j \) is extracted, otherwise, it is 0. Following each scenario, the block in \( i \)th row and \( j \)th column \( (b_{ij}) \) would be assigned by 1 or 0. Therefore, in the first run (replication), the \( b_{ij} \) would be inside of the final pit limit with the probability of \( 1/N \) or 0, where \( N \) is the number of repetition. Therefore:

\[
P_{ij,1}=1/N \text{ or } P_{ij,1}=0. \quad (8)
\]

Later, in the second run, the probability can be calculated by:

\[
P_{ij,2}=P_{ij,1}+1/N \text{ or } P_{ij,2}=P_{ij,1} \quad (9)
\]

And in the same way for the third run:

\[
P_{ij,3}=P_{ij,2}+1/N \text{ or } P_{ij,3}=P_{ij,2} \quad (10)
\]

Generally, the \( b_{ij} \) will be inside of the final pit limit in \( r \)th run with the probability of:

\[
P_{ij,r}=P_{ij,r-1}+1/N \text{ or } P_{ij,r}=P_{ij,r-1} \quad (11)
\]

Since each run provides a single estimate of the UPL; several replications are required to assure a reliable estimate of the probable pits. Hence, the process is repeated and the new results are added to the previous ones. The simulation would be repeated for a sufficient number of times \( (N) \) until the probable pits are generated with no significant changes in their probabilities of occurrence. By this way, the probable pits are easily achievable.

### 2.5. Risk evaluation

The purpose of risk evaluation is to make decisions, based on the outcomes of risk analysis, about which risks need treatment and what the treatment priorities are. The process is performed by comparing the level of risk against predetermined standards, target risk levels or other criteria. The output of a risk evaluation is a prioritized list of risks for further action [26]. If the calculated risk is lower than a pre-determined value, then the risk is acceptable (tolerable). Otherwise, the risk is unacceptable (intolerable), and risk-reducing measures are required. The risk should be reduced to a level that is As Low As Reasonably Practicable (ALARP). In the case which is concerned here, the levels of risk raised from locating the crusher at different probable pit rims are prioritized. Then, because of the lack of any pre-determined acceptable risk level in this regard, the probable pit rim with the lowest practicable risk level is selected as the best option.
2.6. Risk Treatment

As it is depicted in Figure 3, the risk assessment is followed by risk treatment wherein the measures to avoid, reduce, optimize and transfer risk are processed. How one chooses to treat risk will depend on which type of strategy the organization has in place for the risk management [26]. Low and accepted risks should be monitored and periodically reviewed to ensure they remain acceptable. The risk associated to the crusher location can be minimized by putting the crusher in a location with the lowest risk level. In other words, a proactive policy in the preliminary designing process must be taken before locating the crusher. In the second level of treatment, relocating the crusher during the mine life needs a cost-benefit analysis.

3. Applying the proposed methodology

3.1. Two-dimensional hypothetical case study

In the following, the suggested risk management process will be implemented in a two-dimensional hypothetical geological block model of a copper deposit. Figure 6 shows the geological block model containing 210 blocks. Block dimensions are assumed as 12.5 m × 25 m in height and width, respectively. The perpendicular to the plane expansion of the model is assumed as 25 m. The rock density (waste and ore) is assumed to be equal to 2.43 tons per cubic meter.

3.2. Estimating the CDF of the price data

To model the price volatilities, the MCT was applied. The MCT creates the random numbers and links them with corresponding estimated variables using specified statistical distributions. Thus, the main step is to generate a CDF from subjective or historical data for the variable. In the first step, the historical changes in mineral price are collected in order to simulate future possible price forecasts. The 316 monthly price data from Jan 1990 to April 2016 were collected. Table 1 shows the descriptive statistics of all price data.

As it is pictured in Figure 7, the price data have two different populations with averages of two and seven. These characteristics should be embedded into the CDF function appropriately. For this propose, the variation interval of the data (1.37-9.87 $/kg) was divided into twenty equal intervals and then the frequency of the data in each interval was calculated separately. Subsequently, the best curve fitted to the cumulative frequencies of the data is selected as the finest CDF function for generating the price scenarios.

3.3. Calculating the probabilities

The framework containing the before mentioned formulas (Equations 1-11) is coded in MATLAB. The technical and economic data presented in Table 2 are used for calculating the blocks economic value. The simulation is done one thousand times. The results of the simulation are illustrated in Figure 8 and Table 3. In this hypothetical 2D model, eight different pits were calculated using the proposed simulation-based method. The numbers inside the blocks demonstrate the likelihood of the pit occurrence. The number 1 means that the block was within the pit limits in all

<table>
<thead>
<tr>
<th>mean</th>
<th>maximum</th>
<th>minimum</th>
<th>Std. deviation</th>
<th>variance</th>
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<td>4.08</td>
<td>9.87</td>
<td>1.37</td>
<td>2.52</td>
<td>6.39</td>
</tr>
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</table>

Figure 7. Probability distribution function of the price data
Cut-off grade of 0.066 were calculated to be within the one thousand simulations. Similarly, the purple blocks (blocks with numbers 0.066) were calculated to be within the one thousand simulations. Similarly, the purple blocks were increased 25 meters with the probability of 2.5% if the other interior pits. On the other hand, the haulage cost would be relocated with the probability of 97.5%. This is just one certain pit. Assuming that locating the crusher at the right half of the pits is considered, then, if the crusher is placed on the rim of the probable pit with 100% confidence (the first pit), then, it would be relocated with the probability of 97.5%. This condition is governing the crusher location at the edge of other interior pits. On the other hand, the haulage cost would be increased 25 meters with the probability of 2.5% if the crusher had been placed on the rim of pit with 97.5% confidence (the second pit). This is also true for the other outer pits. As the same way, assuming that the crusher intended to be at the rim of fifth pit, there exists 53.2% chance that the crusher would be relocated. Also, there are 30.9%, 27.3%, 5.5% and 2.5% chances that the haulage length would increase up to 25, 50, 75 and 100 meters, respectively. There may exist some more scenarios for relocation process if the relocation cost is a function of the distance intended. It is worth noting that the probable pits A and B are independent. In probability theory, events A and B are independent if $P(A|B, C) = P(A)$ and $P(B|A, C) = P(B|A)$. Similarly, $P(A|B, C) = P(A)$. In such a case, having knowledge about either pit does not change our knowledge about the other pit. Therefore, the above chances related to haulage length increments of the fifth pit are not conditional probabilities. It should be noted that the first pit is a benchmark for all scenarios and does not create any risk for locating the crusher because it will certainly happen. In other words, the crusher location would not endure risk when there is just one certain pit.

The risk levels associated with locating the crusher at different seven pit rims (i.e. at the right half of the pits) are examined based on the above methodology. The results of the fifth pit are shown in Table 4. The computational results of the risk levels for other probable pits are depicted in Figure 9 (i.e. the green chart named normal condition). The parameters relocation cost and unit haulage cost are assumed as $30000 and $0.05 per ton per 100 meters, respectively.

### Table 2. The assumed technical and economic data

<table>
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<tr>
<th>symbol</th>
<th>Description</th>
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<th>unit</th>
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<td>$y$</td>
<td>Recovery</td>
<td>95</td>
<td>%</td>
</tr>
<tr>
<td>$s$</td>
<td>smelter and refinery cost</td>
<td>0.7</td>
<td>$/kg cu</td>
</tr>
<tr>
<td>$c_0$</td>
<td>Processing cost</td>
<td>7</td>
<td>$/ton ore</td>
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<td>$c_w$</td>
<td>Mining cost</td>
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<td>$/ton rock</td>
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<td>$g_c$</td>
<td>Cut-off grade</td>
<td>8.5</td>
<td>$/ton ore</td>
</tr>
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</table>

### 3.4. Calculating the risk levels

Up to now, the effects of hazards (price uncertainty) on ultimate pit limit were assessed and the probabilities of different outcomes were calculated using the proposed simulation-based methodology. It commenced as a set of probable pits with a unique likelihood of occurrence. In addition, the possible outcomes (i.e. crusher relocation, leaving tied up ore or haulage cost increase) were demonstrated following the realization of the probable pits. Assuming that locating the crusher at the right half of the pits is considered, then, if the crusher is placed on the rim of the probable pit with 100% confidence (the first pit), then, it would be relocated with the probability of 97.5%. This condition is governing the crusher location at the edge of other interior pits. On the other hand, the haulage cost would be increased 25 meters with the probability of 2.5% if the...

### Table 3. The characteristics of the probable pits

<table>
<thead>
<tr>
<th>Pit</th>
<th>Probability (%)</th>
<th>No of block</th>
<th>Pit depth (m)</th>
<th>No of ore block</th>
<th>No of waste block</th>
<th>Ore tonnage</th>
<th>Waste tonnage</th>
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<tr>
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<td>100</td>
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Figure 8. The probable pits (the numbers show the probability of pit occurrence)
Table 4. Calculating the risk level related to crusher location at the rim of fifth pit

<table>
<thead>
<tr>
<th>Pit</th>
<th>Probability of pit occurrence (%)</th>
<th>Effective probability (%)</th>
<th>Relocation cost ($)</th>
<th>Haulage length increment (m)</th>
<th>Haulage cost added ($/ton)</th>
<th>Ore tonnage</th>
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<th>Risk ($)</th>
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Total risk level 20172

Figure 9. Sensitivity analysis of relocation cost and unit haulage cost on ranking the candidate crusher locations

4. Sensitivity analysis

In the previous section, the risk levels associated with the crusher location were calculated for each probable pit rim with constant parameters. However, these parameters may deviate when the operation starts. Therefore, the critical parameters should be identified and their influence on risk levels should be measured using sensitivity analysis. The two parameters: relocation cost and unit haulage cost per 100 meters were examined to evaluate different outcomes of ranking the candidate crusher location. Figure 9 shows the prioritization in the presence of 20% variation for each parameter when the other parameter is constant. As it is depicted, locating the crusher at the rim of the fifth pit has the lowest risk level. This pit is the same pit as one computed by average commodity price ($4.08/kg cu). To understand the dependence of the risk levels of each pit rim on the parameters, the sensitivity analysis was computed separately for each probable pit rim. The computational results are illustrated in Figure 10. Because the pits 2 and 3 both had almost the same result, the results of the second pit are not shown. The results describe the fact that the risk raised from crusher location is very sensitive to the relocation cost for the interior pits with a high likelihood of occurrence, while the risk value is varied by the unit haulage cost more aggressively for the outer pits. In the case of the middle pits (i.e. 4, 5 and 6), the risk levels are more sensitive to both parameters. Hence, choosing the best strategy cannot be easily achieved following a rough estimation of the parameters. For this purpose, the ratio of relocation cost ($) to unit haulage cost per 100 meters ($/ton.100m) named trade-off ratio (TOR) was defined to evaluate a different aspect of the problem and make the methodology more generic. When the ratio exceeds 1.2 million, the outer pits have the lowest risk level and the interior pits have the lowest risk level if the ratio is lower than 60 thousand. The ratios between these two numbers are to be evaluated more accurately. In real cases, in which the location of the crusher is mainly affected by the main downstream processing plant, the crusher should be in the same direction (the same area) as plant location. The topography, ore-body shape, depth of the ore and type of mineral may avoid probable pits in a special section of the pit. In these cases, the crusher would not endure any risk because of price uncertainty if it is located at the pit rim. In addition, the haulage cost increment is also a function of the pit exit position and existing or planning haul road while in this research; it is computed based on block dimensions. Furthermore, the crusher relocation cost may be interrelated to intended displacement length, which imposes some modification on the risk computation process.
5. Conclusion

The most popular assumption in conventional open pit mine planning is that metal prices is fixed (i.e. does not change during life-of-mine and is known with certainty). Looking at the past history of metal price, obviously, this assumption is far from realistic. Therefore, assuming that this variable will not change throughout the life of mine, mine planning will result in either over- or under-valuation of mining blocks or, subsequently, the pit limits. In both cases, deviations from scheduled plan and sub-optimized project value are likely outcomes. The tonnages of mineable reserves, the production scale of the operation, and size and location of surface facilities, such as, ore stockpiles, waste dumps, and primary crusher are established based on the preliminary pit area and its expected future expansions. Some of these facilities, such as, primary crushers are likely to be positioned at the pit rim exactly due to the haulage cost limitations, even though; their locations are restricted by topographical circumstances. In this paper, a quantitative simulation-based risk management approach was proposed to help decision maker to overcome the risk raised from locating the crusher at the pit rim because of commodity price uncertainty. A widespread risk picture is prepared to identify the main elements of the problem. Afterward, to validate the proposed approach.
methodology, it was applied in a two dimension hypothetical copper example. Furthermore, the robustness of the design parameters was evaluated through a sensitivity analysis to determine the relative importance of selected parameters. The results show that, according to the given parameters, the pit calculated by average price has the lowest risk level. Also, the inner pit rims are very sensitive to the relocation cost and thus if the relocation cost increases by 200%, the risk level for the first pit will be doubled. Inversely, the outer pit rims are more sensitive to the unit haulage cost in a way that the risk level increases by 200% if the unit haulage cost increases by 200%.

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Recommended technology of open-pit mining low power steeply dipping parts of Ekibastuz trough wings

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²LLC «Angrensor Energo», Ekibastuz, Kazakhstan

Abstract — Latest achievements in controlling stability of open-pit edges allow increasing slope angles of operating open-pit edges while mining inclined and steeply dipping mineral occurrences and coal mines. However, minimization of stripping operations often does not allow sustaining project capacity of open-pit mines. Conducting mining operations on limited space decreases safety of service staff and using mining equipment. Absence of platforms with sufficient dimensions limits implementation of short-delay blasting, as a result, output of oversized pieces is exceeded after large-scale blasts, which in turn negatively affects the performance of excavators. These problems are worsened by continuing economic crisis. Decrease in company profits, especially on active open-pit mines, limits funds for repair and service of mining equipment; thus, technical readiness coefficient of such equipment would be decreased.

Keywords: open-pit mining, steep open-pit edges, double open-pit edge deepening system, working zone formation, extended open-pit fields

1. Introduction

Technological development rate of domestic mines depends on right choice of mechanical equipment, technology and organization of their usage [1]. Kazminerals group successfully develop unique copper mineral occurrences of Bozshakol and Aktogai in Kazakhstan. Construction of two mines with high productive capacity is ending in short period. Development projects ensure high rhythmicity of stripping and extraction operations with construction of processing plants on platforms. If high power excavator-truck complexes (ETC) are used on mines, then ore is transported to a processing plant by conveyors.

Extraction volume has decreased on active iron ore and coal mines compared to 2014. It is mainly associated with non-fulfillment of projected volume of stripping operations. Operating platforms are usually narrowed in the lower parts of a stripping zone. Transition to cyclical and continuous method is delayed on many open-pit mines due to the economic crisis. Thus, mining operations in this period are conducted on steep open-pit edges, which makes it hard to keep required dimensions of active part of deep open-pit mines’ working zone, which in turn is needed for extraction of projected amount of minerals and safe implementation of accepted calendar schedule of mining operations.

Many scientific papers on solving complex organizational and technical problems of creating new methods and ways of conducting mining operations on limited space have been written recently, especially Russian and Ukrainian scientists have done a lot of work in this area [2-4].

2. Materials and Methods

2.1 Offered design of working zone formation

A design of a working zone along steep open pit edges developed by us assumes mining high pit banks from two levels of an excavator position by cross panels and perpendicular orientation of an operating front of working zone to operating front of pit banks of steep open pit edges [5-9]. These methods of open-pit mining are protected by innovational patents № 26485 and 29038 in the Republic of Kazakhstan. Known elements of phased development of deep open pit mines by steep layers, development of open pit edges by cross and diagonal stopes, mining stripping pit banks by a high pit face on extended open-pit fields, decreasing average stripping ratio in the main period of mining steeply dipping mineral occurrences, are taken into account to justify offered development of working zones along steep open pit edges.

The purpose of the scientific work is a conduct of pre-project studies on adaptation of safe and intensive technology of working zone formation along steep open pit edges on limited mining space of steeply dipping parts of Ekibastuz trough wings.

The main advantage of mining low power steeply dipping parts of Ekibastuz trough is simultaneous extraction of low-ash and high-ash seams. Longitudinal double pit edge deepening system using excavator and truck complexes is implemented, mainly of a constructional type. The feature of mining reserves of field 11 of “Ekibastuzskyi” mine is adjacency of untouched mining allotments of other subsoil
users from two ends of the mine. Thus, length of extraction operating front is decreased during intensive development of mining operations to the depth. Therefore, relatively small productive capacity is taken in projects. However, volumes of overburden increasing with depth and limited space could limit the depth of open-pit mining operations.

### 3. Results and Discussion

#### 3.1 Possible implementation of offered design in conditions of “Ekibastuzskyi” mine

The feature of offered technology of working zone formation on “Ekibastuzskyi” mine is a creation of safe conditions of mining operations production on steep longitudinal open pit edges. Conditions ensuring perpendicular orientation of operating front of working zone pit banks to operating front of pit banks of steep open pit edges are created during transition to mining high pit faces by cross panels from two levels of an excavator position. Haulage benches with width of 18-20 meters are left between pit banks in hard bedrock and along coal, and safety benches with width of 4-5 meters are left between upper and lower parts of pit banks. Mining upper parts of 10-12 meters high pit banks is conducted by downward digging of hydraulic excavators with backhoe shovel, and lower parts are mined by an excavator with face shovel (Fig. 1). Optimization of mining operations regime is achieved through the transfer of part of stripping operations to later periods of overburden excavation and narrowing of final contours of coal mine.

![Figure 1](image.png)

**Figure 1.** Outline of mining 20-24 meters high pit banks by cross panels using excavators with backhoe and face shovels (eastern end of south open pit edge)

Angle of hade of steep open pit edges should not exceed maximum acceptable values due to stability conditions. Calculations were made for “Ekibastuzskyi” mine by analogy to calculations of “NTC-Geotechnology” Ltd conducted for area 12 of Ekibastuz mineral occurrence with the same position. Stable and constructive angles of hade of its longitudinal pit edges are shown in Table 1. Stability coefficient is taken as \( n = 1.5 \).

<table>
<thead>
<tr>
<th>Open pit mine depth, meters</th>
<th>Slope angle of south longitudinal open pit edge with ( n = 1.5 ), degrees</th>
<th>Slope angle of north longitudinal open pit edge with ( n = 1.5 ), degrees</th>
</tr>
</thead>
<tbody>
<tr>
<td>130</td>
<td>19.5, 29</td>
<td>19.5, 29</td>
</tr>
<tr>
<td>145</td>
<td>22, 29</td>
<td>22, 29</td>
</tr>
<tr>
<td>150</td>
<td></td>
<td></td>
</tr>
<tr>
<td>160</td>
<td>23, 28.5</td>
<td>23, 29</td>
</tr>
<tr>
<td>170</td>
<td>24, 28.5</td>
<td>24, 29</td>
</tr>
<tr>
<td>180</td>
<td>24, 28.5</td>
<td>24, 29</td>
</tr>
<tr>
<td>200</td>
<td>25, 27.5</td>
<td>25, 28.5</td>
</tr>
<tr>
<td>225</td>
<td>25, 27.5</td>
<td>25, 28.5</td>
</tr>
<tr>
<td>250</td>
<td>25, 27</td>
<td>25, 28</td>
</tr>
<tr>
<td>275</td>
<td>25, 26.5</td>
<td>25, 27</td>
</tr>
<tr>
<td>300</td>
<td>25, 26</td>
<td>25, 26</td>
</tr>
<tr>
<td>325</td>
<td>25, 25.5</td>
<td>25, 26</td>
</tr>
<tr>
<td>350</td>
<td>25, 25</td>
<td>25, 25</td>
</tr>
<tr>
<td>375</td>
<td>25, 25</td>
<td>25, 25</td>
</tr>
<tr>
<td>400</td>
<td>25, 25</td>
<td>25, 25</td>
</tr>
</tbody>
</table>

* Note: Periodic excavation using hydraulic excavators should be carried out in central part of north longitudinal open pit edge to avoid landslides. If pit edge is deformed from daylight area then wells are drilled in that part of a mine and they are filled with cement.

Implementation order of recommended mining technologies in conditions of “Ekibastuzskyi” mine consists of several parts. Initially, it is necessary to mine high pit faces by cross panels from two levels of an excavator position. Stripping operations in upper and partially in the middle parts of stripping area are stopped. They are conducted only in lower parts of stripping area (Fig. 2-7). New working zones would be formed on each pit edge after creating new design of longitudinal open pit edges on stripping area, operating front of pit banks of those pit edges would have perpendicular orientation to operating front of pit banks of active pit edges. Optimal width and length of panels on each pit bank of a working zone would be established after detailed mining-geometric analysis depending on coal capacity of a mine.
Figure 2. Construction of cross panels on exploratory line 75

Figure 3. Longitudinal section of south open pit edge of “Ekibastuzsky” mine from its western end

Figure 4. Longitudinal section of south open pit edge of “Ekibastuzsky” mine from its eastern end

Figure 5. End of south open pit edge with location of excavators and system of access tracks in a design

Figure 6. Location of excavators on north open pit edge of “Ekibastuzsky” mine with minimum separation in a design

Figure 7. Outline of mining operations development on “Ekibastuzsky” mine
Table 2 shows calculations of required capacity by external overburden by mining phases of panels. Extracted coal of 1.68 million tons and overburden of 3.4 million m³ from the period of September-December of 2015 were excluded from coal reserves and overburden volume.

### Table 2 – Required mine capacity by external overburden by mining phases of panels

<table>
<thead>
<tr>
<th>Panels formation phases</th>
<th>Coal reserves, million tons</th>
<th>Mining period of coal reserves, years</th>
<th>External overburden volume, million m³</th>
<th>Required external overburden capacity, million m³</th>
</tr>
</thead>
<tbody>
<tr>
<td>00-00+11-12</td>
<td>6.03</td>
<td>1.1</td>
<td>4.07</td>
<td>4.48</td>
</tr>
<tr>
<td>21-22</td>
<td>3.56</td>
<td>0.65</td>
<td>6.27</td>
<td>9.65</td>
</tr>
<tr>
<td>31-32</td>
<td>3.15</td>
<td>0.57</td>
<td>8.29</td>
<td>14.54</td>
</tr>
<tr>
<td>41-42</td>
<td>3.12</td>
<td>0.57</td>
<td>9.71</td>
<td>17.04</td>
</tr>
<tr>
<td>Total</td>
<td>15.86</td>
<td>2.89</td>
<td>28.34</td>
<td></td>
</tr>
</tbody>
</table>

Annual capacity by external overburden in calendar years is taken to be as following: 2016 – 8.5 million m³; 2017 – 10.5 million m³; 2018 – 12.35 million m³.

Calculations show that in 3 years (from 2016 to 2018) volumes of external overburden compared to projected mining operations schedule could be decreased by 8.043 million m³. Required annual capacity of mine by external overburden in this period changes from 4.48 million m³ in 2016, to 10.356 million m³ in 2017 and 16.515 million m³ in 2018. Considering delay of stripping operations, transition to offered technology of working zone development in the period of 2016-2018 allows decreasing these volumes by 14.28 million m³.

Savings from decreased amount of stripping operations, considering delay in the period of 2016-2018 with design costs of 0.3 million USD by LLC «Angrensor Energo», would be 4.6 million USD. Hence, costs on implementation of offered technology of working zone development due to optimization of mining operations regime would pay off more than 15 times.

### 4. Conclusions

Technical possibility and economic expediency of transition to mining high pit faces by cross panels from two levels of an excavator position with perpendicular orientation of operating front of working zone pit banks to operating front of pit banks of steep pit edges are proven on a base of “Ekibastuzkyi” mine while conducting pre-project studies.

Hade angle of operating open pit edges is limited by parameters of panels being mined and does not exceed its stable value. If operating front of working zone pit banks oriented perpendicular to operating front of pit banks of active open pit edges, then constructive hade angle of the latter could reach its stable value. In such case, working zone kind of slides along steep active open pit edge. Angle of hade of an active open pit edge would be increased if only haulage benches are left between pit banks with construction of access tracks for technological transport on certain places. Safety benches are left between parts of high pit faces.

Constructive angle of hade reaches its maximum value when width of panels is equal to width of haulage benches. However, amount of minerals being extracted would be limited in such case. Minimum width of panels is usually used while mining extended open-pit fields by double pit edges deepening system with small horizontal breadth of minerals and round shaped mineral occurrences with limited dimensions in design with low productive capacity. Wherein, excavator and truck complexes with low capacity are used, such usage predetermines replacement of traditional dump trucks with hinged dump trucks with capacity of 50 tons, which leads to an increase of longitudinal angle of hade up to 15-18 degrees (3.3-4 times more). It should be noted, that offered design of working zone development along steep open pit edges could be safely implemented only with hard bedrock, and overburden should be stripped by low pit banks, leaving working platforms between them.

### References


An Investigation Into Effect of Tailing Particle Size in the Strength Evolution of Mine Back Fill

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Abstract: In this paper, the effect of the particle size of tailings and the binder concentration on the rheological and mechanical properties of cemented mine paste backfill were investigated by conducting a series of laboratory experiments. Different mine backfill samples were prepared and tested. Large numbers of samples with various mixture designs were cast and cured over 56 days. The mechanical properties of the samples were investigated using the uniaxial compression test (UCS), and the results were compared with those of reference samples made. The test results indicated that the particle size distribution has a great effect on the mechanical (uniaxial compressive strength and rheology) properties of backfill. The rheology analysis revealed that the modification of the particle size of tailings particles can modify the rheological properties of mine backfill.

Keywords: Rheology; Particle Size; Tailings, Paste Backfill, UCS

1. Introduction

The process of filling the void created by underground mining activities with waste materials is defined as mine backfilling. Mine backfill has become an integral part of underground mining methods. Cemented paste backfill (CPB) is used worldwide for underground backfilling of mined-out stopes. It is a highly efficient technique for providing secondary ground support and increasing long term mine stability. Mine backfilling is primarily used to increase ore extraction, increase ground mine stabilization, and deposit waste materials. [1, 2]. CPB mainly consists of tailings (78–85% solid content, C_s), a binding agent (3–7% by dry mass of tailings), and water. The binder hydration process leads to self-consolidation of the backfill and the development of adequate mechanical strength [3, 4].

CPB is usually mixed on surface and then transported to the underground stopes by the means of gravity and/or pumping [5]. The uniaxial compressive strength (UCS) of the CPB is widely used in practice to evaluate its strength and stability. A high rate of early strength gain is very important since it can reduce the mining cycle and increase production. Reducing the moisture content or the water to cement ratio of the paste can increase the strength of CPB in the short term. However, CPB needs to have a certain level of fluidity in order to be properly pumped and transferred in the mine [6].

Rheological behaviour of the CPB must be understood in order to optimize its flow characteristics and improve transportation parameters. Optimizing the CPB may help improve transportation parameters and reduce the risk of pipe clogging. Factors which contribute to rheological behavior of CPB are the solids content, the Particle Size Distribution (PSD), the temperature and the level of pH [7, 8].

The yield stress of CPB has been found to increase with increasing solid content [8]. However, for a given binder type and content, increased solid content improves mechanical performance [9]. Higher yield stress also translates to lower workability so it is crucial to find the right balance between the two.

The particle size is another important physical property of backfill. It has been found to affect the mechanical properties and the rheological properties [10, 3]. Fine particles in tailings have been found to increase the yield stress by increasing the packing density [11]. It has also been observed that decreasing the amount of fine particles in tailings can increase the uniaxial compressive strength of cemented backfill due to the improvement in the gradation of tailings [12]. Another study has found that a mixture of two materials with different particle size distributions produces a dense packing, which renders higher values for the apparent yield stress as compared to a single-component mixture [13].

This study focuses on the effects of paste compositions such as solids content, binder content, and variations of Particle Size (PS) on mechanical and rheological properties of CPB. Variations of PS were achieved by mixing different ratios of two different sized tailings with similar chemical compositions.

2. Material and Methods
2.1. Tailings

Tailings are waste materials produced in ore processing plants. The materials consist primarily of finely ground host rock. The physicochemical properties of tailings have a
significant effect on the mechanical performance of mine backfill [3, 12]. In this research, two tailings namely Tailings A (TA) and Tailings B (TB) with different particle size distributions were used to investigate the effect of particle size on the mechanical and rheological properties of CPB. Both tailing samples were delivered from a mine in Canada. The mineralogical content of the tailings generally consists of quartz, albite and a slight quantity of calcite, muscovite, pyrrhotite, chalcopyrite, anorthite, and chlorite. The particle size distribution of the tailings was determined by using the laser diffraction methods (ASTM, 1996). The result is presented and compared with the average size of 11 mine tailings (T Ave.) from the provinces of Quebec and Ontario, reported by Ouellet et al. (2008) in Figure 1 as well as Table 1 and it was observed that the Tailings B is courser than Tailings A, as well as the average size of the 11 mine tailings.

![Figure 1: Particle size distributions of tailings and average size of 11 mine tailings.](image)

Table 1. Physical properties of the tailings.

<table>
<thead>
<tr>
<th>Material</th>
<th>( D_{10} ) (µm)</th>
<th>( D_{30} ) (µm)</th>
<th>( D_{60} ) (µm)</th>
<th>( D_{90} ) (µm)</th>
<th>( C_a )</th>
<th>( C_c )</th>
<th>Specific gravity, ( G_s )</th>
</tr>
</thead>
<tbody>
<tr>
<td>TA</td>
<td>5.7</td>
<td>35.1</td>
<td>132.5</td>
<td>9.04</td>
<td>0.98</td>
<td>2.77</td>
<td>2.77</td>
</tr>
<tr>
<td>TB</td>
<td>2.97</td>
<td>16.8</td>
<td>22.8</td>
<td>101</td>
<td>7.68</td>
<td>1.48</td>
<td>2.90</td>
</tr>
<tr>
<td>T Ave.</td>
<td>2.2</td>
<td>20</td>
<td>29</td>
<td>102</td>
<td>13.2</td>
<td>1.24</td>
<td>NA</td>
</tr>
</tbody>
</table>

\( D_{10} \) = particle diameter size that 10% of the sample particles are finer than
\( D_{30} \) = particle diameter size that 30% of the sample particles are finer than
\( D_{60} \) = particle diameter size that 60% of the sample particles are finer than
\( D_{90} \) = particle diameter size that 90% of the sample particles are finer than
\( C_a \) = \( D_{60}/D_{10} \) = coefficient of uniformity
\( C_c \) = \( (D_{30})^2/(D_{60} \times D_{10}) \) = coefficient curvature
NA= Not available

2.2. Binder

Binders are mainly used to increase the mechanical stability of fill materials. The most expensive part of mine backfill is the binders, and the cost of binder used in backfill could represent up to 75% of backfill costs [14]. Normal Portland cement, fly ash and blast furnace slag have been mainly used for mine backfill. In this research Type 10 Portland cement provided by Lafarge Canada, was used.

Type 10 Portland cement is one of the main binders, and is used in different mines in Ontario, Canada. The density of the Portland cement used was 3.07 g/cm³ and the Blaine specific surface area of the Portland cement was 3710 cm²/g, respectively. The chemical compositions of the Portland cement is shown in Table 2.

Table 2. Chemical composition of the Portland cement and blast furnace slag provided by Lafarge.

<table>
<thead>
<tr>
<th>Chemical composition</th>
<th>Blast furnace slag (wt%)</th>
<th>Portland cement (wt%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>CaO</td>
<td>37.129</td>
<td>61.13</td>
</tr>
<tr>
<td>SiO₂</td>
<td>36.127</td>
<td>19.39</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>10.385</td>
<td>4.61</td>
</tr>
<tr>
<td>MgO</td>
<td>13.246</td>
<td>3.3</td>
</tr>
<tr>
<td>SO₃</td>
<td>3.362</td>
<td>2.27</td>
</tr>
<tr>
<td>Fe₂O₃</td>
<td>0.668</td>
<td>2.01</td>
</tr>
<tr>
<td>Na₂O</td>
<td>0.424</td>
<td>2.03</td>
</tr>
<tr>
<td>K₂O</td>
<td>0.489</td>
<td>0.71</td>
</tr>
</tbody>
</table>

3. Experimental setup

3.1. Sample Preparation and Mixing Design UCS Test

In order to investigate the effect of the tailings particle size on the UCS evolution of CPB, 16 different mixtures were prepared. Tailings A and Tailings B were mixed by different ratios and different cement contents. The yield stress of the samples was kept constant at 250 Pa as is practiced in the mine in Canada. 4 samples were made with different yield strength to observe the effect of yield stress on mechanical strength. Portland cement was added to the mixture by weight of dry tailings. Sample mixtures were prepared in small batches in a 5-L stainless steel bowl. The mixtures were mixed for 5 min and then casted into Cylindrical, polyvinyl moulds. They were then cured in a curing chamber where the relative humidity was kept constant at (90±2)% and the temperature was adjusted to (25±1) °C. They were then tested at 7, 14, 28 and 56 days. The compressive strength of the cured specimen was measured by conducting the unconfined compressive strength (UCS) test. The test was conducted with a “Wykeham Farrance 100 kN” pressure equipped with a 50 kN load cell by conducting uniaxial compression tests (ASTM, 2006). A linear variable displacement transducer (LVDT) sensor was used to obtain the samples’ vertical deformation rate (strain). UCS tests were conducted at 7, 14, 28 and 56 days. Below shows the number, the ingredients and the yield stress of each sample.

3.2. Rheological Characterization

In order to measure the rheological properties of the materials, a desktop shear vane rheometer was used (Figure 2). The specification of the rheometer used in this research is...
presented in Table 4. Initially, in order to measure the yield stress of the samples, two 3-kg representative samples were carefully selected and prepared from Tailings A and Tailings B. Then, the 3 kg-tailings samples were mixed with 5% of the cement by total dry weight of the sample. Water was added to the mixture in order to make a thick paste. The samples were sheared with the rheometer and the yield stress and solid concentration were measured. A small amount of water was added to the mixture and the test repeated till the yield shear stress dropped below 70.

7 different batches with different ratios of Tailings A and Tailings B, as well as different cement contents were prepared to investigate the effect the tailings particle size and binder addition on the yield stress of cemented backfill. Table 5 shows the batch number and its contents.

Table 3: Binder mixtures characteristics of backfill samples.

<table>
<thead>
<tr>
<th>sample #</th>
<th>TA (%)</th>
<th>TB (%)</th>
<th>Cement (%)</th>
<th>Yield Stress (Pa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>99.0%</td>
<td>0.0%</td>
<td>1%</td>
<td>250</td>
</tr>
<tr>
<td>2</td>
<td>95.0%</td>
<td>0.0%</td>
<td>5%</td>
<td>250</td>
</tr>
<tr>
<td>3</td>
<td>91.0%</td>
<td>0.0%</td>
<td>9%</td>
<td>250</td>
</tr>
<tr>
<td>4</td>
<td>65.9%</td>
<td>33.1%</td>
<td>1%</td>
<td>250</td>
</tr>
<tr>
<td>5</td>
<td>63.3%</td>
<td>31.7%</td>
<td>5%</td>
<td>250</td>
</tr>
<tr>
<td>6</td>
<td>60.6%</td>
<td>30.4%</td>
<td>9%</td>
<td>250</td>
</tr>
<tr>
<td>7</td>
<td>27.2%</td>
<td>71.8%</td>
<td>1%</td>
<td>250</td>
</tr>
<tr>
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<td>14</td>
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<td>68.9%</td>
<td>5%</td>
<td>350</td>
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</tbody>
</table>

Table 4: Rheometer Specification

<table>
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<tr>
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</tr>
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<tbody>
<tr>
<td>Vane</td>
<td>FL 100</td>
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<tr>
<td>Vane Diameter (mm)</td>
<td>22</td>
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<tr>
<td>Vane Height (mm)</td>
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<tr>
<td>Rotation Rate (deg/min)</td>
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<tr>
<td>Rotation Rate (1/s)</td>
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</tr>
<tr>
<td>Rate of Shear (mm/s)</td>
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<tr>
<td>Room Temperature (°C)</td>
<td>22</td>
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</table>

Table 5: different Mixtures and rheological properties

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<tr>
<th>Batch #</th>
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<th>TB (%)</th>
<th>Cement (%)</th>
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<tr>
<td>1</td>
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</tr>
<tr>
<td>6</td>
<td>26.10</td>
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</tr>
<tr>
<td>7</td>
<td>0.00</td>
<td>95.00</td>
<td>5</td>
</tr>
</tbody>
</table>

3.3. Unconfined compressive strength (UCS) tests
The mechanical strength of the cured specimens was measured. The test was conducted with a “Wykeham Farrance 100 kN” pressure equipped with a 50 kN load cell by conducting uniaxial compression tests (ASTM, 2006). A linear variable displacement transducer (LVDT) sensor was used to obtain the samples’ vertical deformation rate (strain). Samples were taken out from the humidity room just prior to conducting the unconfined compressive strength test. A data acquisition board and a computer setup were used to record and display the data. On a given curing day for each mixture, three samples underwent the unconfined compression testing and the average value of the three results was recorded as the overall result of the UCS test.

4. Results and Discussion

4.1. Effect of tailings particle size on the backfill yield stress strength
The shear yield stress profiles of the samples are presented in Figure 3. The yield stress versus the solids concentration curves were developed by fitting exponential trendlines to the data and presented in Figure 3. The equations for these curves are presented in Table 6. It was noticed that the shear yield stress of the samples increased exponentially with increased solids concentration. Furthermore, by comparing different mixtures it
is observed that moving from coarse tailings (Tailings A) to more fine tailings (Tailings B), the solids concentration decreases at the same yield stress. The same statement is true for 100, 250 and 300 Pa yield stresses. Comparing mixtures with the same ratio of Tailings A to Tailings B (mixtures 1 and 2, mixtures 4 and 5). As the addition of binder decreases, the yields stress of the CPB samples increases. For instance, at 100 Pascal yield stress solid concentration of mixture 2 with just 5% binder, drops from 74.25% to 73.94%. Replacing the 33.4% of Tailings A with Tailings B in mixture 3, drops solid concentration from 73.94% to 72.29% at 100 Pascal yield stress.

4.2. Effect of particle size distribution of tailings on the strength of CPB

In order to investigate the effect of the tailings particle size on the uniaxial compressive strength, the result of the UCS test of the CPB samples prepared with Tailings A and Tailings B with 1%, 5% and 9% binder concentrations are presented in Figure 4 and Figure 5 respectively. It can be observed that the samples prepared with fine tailings (TA) has lower mechanical strength in comparison to the samples prepared with coarse tailings (TB). It could be due to the higher surface area of fine particles in comparison to coarse tailings. Therefore, an elevated amount of binder is required to cover the higher surface area of fine particle tailings. It can also be observed that the UCS evolution of CPB samples prepared with coarse tailings is more rapid than the samples prepared with fine tailings. Figure 7 and Figure 8 show the UCS evolution of CPB samples prepared with various ratios of Tailings A and Tailings B containing 5% and 9% binder concentration respectively. It can be observed that the size gradation of the combined tailings has a great impact on the strength evolution of CPB samples.

Table 6: The yield stress versus solids concentration curve equations

<table>
<thead>
<tr>
<th>Batch #</th>
<th>Equations</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>( \tau_y = 5.6593 \times 10^{-13} e^{44.178x} )</td>
</tr>
<tr>
<td>2</td>
<td>( \tau_y = 1.2034 \times 10^{-13} e^{46.456x} )</td>
</tr>
<tr>
<td>3</td>
<td>( \tau_y = 2.9869 \times 10^{-10} e^{36.707x} )</td>
</tr>
<tr>
<td>4</td>
<td>( \tau_y = 1.8916 \times 10^{-9} e^{35.014x} )</td>
</tr>
<tr>
<td>5</td>
<td>( \tau_y = 1.173 \times 10^{-9} e^{30.166x} )</td>
</tr>
<tr>
<td>6</td>
<td>( \tau_y = 2.5241 \times 10^{-9} e^{31.446x} )</td>
</tr>
<tr>
<td>7</td>
<td>( \tau_y = 2.4359 \times 10^{-9} e^{31.854x} )</td>
</tr>
</tbody>
</table>
4.3. Effect of CPB yield stress (solid concentration) on the backfill strength

The UCS test results of CPB samples prepared with different combinations of Tailings A and Tailings B and 5% Portland cement binder at various yield stresses are shown in Error! Reference source not found. and Error! Reference source not found. respectively. The UCS values of CPB samples increase with the increase of curing time due to hydration of binder agents. Additionally, the UCS values of CPB samples increase with the increase of yield stress. This expected behaviour can be due to the reduction of the water to binder ratio. In fact, by reducing the water to binder ratio, the total porosity is decreased and therefore, the density and the strength of the material are increased.
5. Conclusions
The effect of the particle size of tailings and binder dosage as well as different binders on the mechanical properties of CPB is presented in this paper. The investigation confirmed that CPB specimens produced by fine tailings (TA) have lower mechanical strengths in comparison to CPB specimens made with coarse tailings (TB). Moreover, it was observed that the strength acquisition of backfill samples prepared with Tailings B (coarse tailings) is more rapid than the samples prepared with Tailings A (fine tailings). Furthermore, the research also shows that the particle size of tailings has a great effect on the UCS of the samples. It was also demonstrated that binder dosage and binder types strongly influence the UCS evolution of CPB samples. Finally, it was demonstrated that the yield stress of CPB can mainly be changed by the particle size of tailings.

Acknowledgement
The authors acknowledge the financial support given by SERC for the mine backfill distribution system wear study. The authors are also grateful for the help and support of Glencore, Vale, Barrick, Lake Shore Gold, IAMGOLD and HudBay Minerals. The help and contribution of other graduate and undergraduate students at McGill University must also be recognized. The authors thank Glencore for the donation of the roller machine for the wear study.

References


Optimization of mining schedule during safe development of working zones along steeply inclined open pit edges

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Abstract— An algorithm for optimizing the mining schedule has been developed using the example of the “Vostochnyi” coal mine of the Ekibastuz region to implement the technology of safe intensive development of working zones along the steep open pit edge. Optimization operations of the mining operations mode and the transformation of its results into a mining schedule are combined in one algorithm. It will allow technical service to adjust the approved project schedule of mining operations at the end of each calendar year, depending on the actual volumes of coal mining and overburden excavation with recalculation of the required stripping volumes. Thus, this method of rapid assessment of design solutions while combining optimization of the open pit-mining mode and the schedule of their production allows, with a change of demand on commercial products, to correct the required volumes of stripping operations, taking into account the production rhythm of overburden, mining and mining preparatory operations.

Keywords: coal mining, inclined mineral occurrences, optimization operations, excavator and automobile complexes, cyclical and continuous method

1. Introduction
The "Vostochnyi" open pit mine of the Ekibastuz region was put into operation in 1989 with advanced continuous coal mining and transportation technology. The feature of its implementation was the use of rotary excavators with conveyor lifts while mining inclined coalbeds. Projects of the long-term development of the "Vostochnyi" open pit mine are scheduled to be continued for a long period. In 2010, a cyclical and continuous method was implemented in the lower part of the stripping zone using hydraulic excavators, high capacity dump trucks, a crushing and conveyor complex and a console spreader.

The intensity of the production of stripping operations of the continuous coal mining technology was not reached after transition to combined automobile-conveyor transport in the lower part of the stripping zone at the “Vostochnyi” open pit mine. Mining overburden benches in the lower part of the stripping zone by longitudinal panels with narrow operating platforms does not allow using excavator automobile complexes (EAC) efficiently. The delay in stripping operations limited the coal production capacity.

The width of the operating platforms should also ensure high performance of excavators while mining inclined and steeply dipping mineral occurrences. It is known that, during reduction of its size and operation of excavators with a dead end stope, the time of cycle of excavators increases by 40-50% and the duration of the exchange of dump trucks is almost 2 times higher.

2. Material and Methods
It is possible to increase the design angle of the operating open pit edge up to 30-32° at a depth of 260 m with a reduction only to 28° with an increase in depth to 400 m based on the research ordered by LLP "Eurasian Group" and performed by LLC "NTC-Geotechnology". Stability margin was taken to be equal to 1.3. We offered a technology of transition to mining high pit banks by cross panels from two levels of an excavator position (innovative patent No. 26485 of Kazakhstan) with the construction of temporary openings on the sides of the open pit field [1-3] as one of the options of EAC implementation. The efficiency of the transition to the technology of working zones development along a steep open pit edge with a significant delay of stripping operations is also proved, when the operational front of working zone pit banks is perpendicular to the operational front of pit banks of the operating open pit edge (innovative patent No. 29038 of Kazakhstan). At the same time, only haulage benches are left between high pit banks along the operating open pit edge, and there are safety benches between the parts of pit banks. The length of the operational front of the working zone pit banks is assumed to be equal to the width of the cross panel of the cut-off technological layer (mining phase). Its size should ensure two-way access of dump trucks to excavators in stopes, even with the dead-end development of the operational front and create conditions for the application of multi-row short-delayed blasting of borehole charges.
The feasibility of switching to mining high pit banks by cross panels from two levels of an excavator position has been justified at the pre-design phase in agreement with JSC "Eurasian Energy Corporation".

A transverse section of the "Vostochnyi" open pit mine is shown in Fig. 1 with mining the lower part of the stripping zone with high pit banks (30 m high) from two levels of an excavator position with 50 meters wide cross panels.

Figure 1. Order of mining cross panels of EAC in lower part of stripping zone of «Vostochnyi» open pit mine

The development rates of the lower part of the stripping zone would correspond to the continuous technology of coal mining by adjusting the depth of initial cut during 50% bench development of lower part of the stripping zone and the upper platform of the extraction bench.

Optimization of the position of the operating open pit edge of the “Vostochnyi” open pit mine during the transition to the offered technology of EAC implementation is achieved by the sequential determination of the parameters of its design on mining and stripping zones by mining phases. The required annual volumes of stripping operations are calculated and their averaging is performed after calculation of phased volumes of overburden and coal reserves, ensuring a smooth gradual change of the current stripping ratio values by calendar years.

Average stripping ratio \( k_{\text{ave}} \) is taken as an efficiency indicator on the mined boundaries of open pit in \( k_j \) phases of mining. Minimum efficiency indicator is given by the following expression:

\[
k_{\text{ave}} = \frac{\sum_{j=1}^{k_j} V_{ij}}{\sum_{j=1}^{k_j} Z_j} \rightarrow \min
\]

Minimum efficiency indicator is reached with certain value of \( B_b \) considering fulfillment of the following conditions:

\[
B_{tba} + 15 \leq B_b \leq B_{pa}
\]

\[
H_dj - H_dzz + h_van > H_vzj > H_dj - H_dzz,
\]

\[
k_{vz(i-1)} \leq k_vti \quad \text{and} \quad A_{xy} = Q_{xy} = \text{const}
\]

\[
V_{vti} - V_{vz(i-1)} \leq 4.0 \times 10^6
\]

where, \( k_{\text{ave}} \) - average stripping ratio, m\(^3\)/tons; \( V_{ij} \) - overburden volume within \( j \)-th phase of mining, m\(^3\); \( k_j \) - number of mining phases in studied boundaries of coal mine, pc.; \( Z_j \) - coal reserves within \( j \)-th phase of mining, tons; \( B_{tba} \) - haulage bench width between pit banks, meters; \( B_{pa} \) - working platform width while implementing EAC by design, meters; \( H_dj \) - open pit depth within \( j \)-th phase of mining, meters; \( H_dzz \) - extraction zone height within \( j \)-th phase of mining, meters; \( h_van \) - height of extraction sub-bench in bedrock, meters.

Limiting condition (2) characterizes limits of cross panel width \( B_b \) change; limiting condition (3) ensures correspondence of stripping operations rate to technology of coal extraction with decrease of coal clogging; (4) and (5) optimizes mining operations mode by equal smooth change of current stripping ratio values \( k_{\text{amn}} \) in each \( i \)-th year with gradual increase while reaching annual capacity by coal \( A_{xy} \) by productive capacity \( Q_{xy} \) and elimination of significant increase of annual capacity by external
overburden $V_{inj}$. Parameters of coal beddings are given in Table 1.

<table>
<thead>
<tr>
<th>Number of $j$-th mining phase</th>
<th>$M_j$ – horizontal thickness of coal beddings on $j$-th phase of mining, meters</th>
<th>$\beta_j$ – angle of dip by footwall of lower coal seam being mined within $j$-th mining phase, degrees</th>
<th>$\beta_kj$ – angle of dip by hanging wall of upper coal seam being mined within $j$-th phase of mining, degrees</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>600</td>
<td>19</td>
<td>19</td>
</tr>
<tr>
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<td>18</td>
<td>18.5</td>
</tr>
<tr>
<td>5</td>
<td>580</td>
<td>18</td>
<td>18</td>
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3. Results and Discussion
Currently, the productivity of the “Vostochnyi” coal mine is 18 million tons of coal, the projected production capacity is 20 million tons. The decrease in coal production is due to a significant stripping delay, especially in the lower part of the stripping zone within the boundaries of EAC implementation. It will be possible to organize a two-flank scheme of opening the lower part of the stripping zone and switch to mining high pit banks by cross panels from two levels of an excavator position after setting the second line of the cyclical-and-continuous method complex. Only in this case, the rates of stripping operations development would correspond to the designed coal production capacity. EAC implementation boundaries are given in Table 2.

Mathematic model, implementation algorithm and necessary formulas are described in scientific works [4-6]. Capacity of an open pit mine depending on the phase is shown in Table 3.

<table>
<thead>
<tr>
<th>Number of $j$-th mining phase</th>
<th>$H_{2j}$ – depth of open pit mine, meters</th>
<th>$n_{maxj}$ – number of stripping sub-benches during mining with EAC, pc.</th>
<th>$H_{adj}$ – height of lower part of stripping zone within boundaries of EAC operation, meters</th>
<th>$B_{adj}$ – Initial cut width on overburden within EAC operation boundaries, meters</th>
<th>$N_{n0j}$ – number of safety benches, pc.</th>
<th>$n_{n0j}$ – number of intact pit banks within EAC operation zone, pc.</th>
<th>$\gamma_{adj}$ – slope angle of lower part of stripping zone, degrees</th>
</tr>
</thead>
<tbody>
<tr>
<td>Actual position</td>
<td>225</td>
<td>6</td>
<td>90</td>
<td>6</td>
<td>5</td>
<td>3</td>
<td>25</td>
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<table>
<thead>
<tr>
<th>$j$-th phase of mining</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
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</thead>
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<tr>
<td>$A_{kij} \times 10^6$</td>
<td>18.5</td>
<td>20</td>
<td>20</td>
<td>20</td>
<td>20</td>
</tr>
</tbody>
</table>
Bucket-wheel excavators mine extraction benches with 50 meters wide stopes; hence the width of the cross panel for the EAC is taken as 50 meters (Tables 4-7) for the check computation. It should ensure a rhythmic production of drilling and blasting operations and a high quality of crushing through the implementation of multi-row short-delayed blasting. The minimum limit of the cross panel width is taken for the conditions of the open pit mine operation with the coal capacity below the production capacity in the initial data.

The results of calculating the phased and total reserves of coal along the upper, intermediate, lower extraction pit banks and in the initial cut of the newly opened extraction horizon, as well as for each phase and in general within the open pit limits from 250 to 350 meters of its depth are given in Table 4.

Results of calculating the parameters of the structural elements of the lower part of the stripping zone, the volumes of the lower and upper parts of the stripping zone, respectively, within the boundaries of EAC and ERC (excavator railway complex), external overburden, phased stripping ratio, and in the row “Total” - the average stripping ratio are given in Table 5.

The results of calculating the time spent on mining coal reserves during each phase, their sum, the estimated annual overburden capacity during each phase, its first and second corrections, followed by the current stripping ratio within each mining phase and like in Table 5, value of the average stripping ratio are given in Table 6. Adjustments to the estimated required annual overburden capacity during each phase are performed in accordance with the constraining conditions of the model and are aimed to eliminate the stepwise increase in its size from one mining phase to another by uniform distribution, beginning with the start of the correction phase, with transfer to subsequent mining phases.

The processing of the results of mining-geometric analysis is performed with obtaining the optimum schedule of mining operations mode in Tables 4-6. Afterwards, a schedule of mining operations is compiled while filling the Table 7. The economic justification is given in Table 8.
### Table 4 – Beginning of calculations with cross panel width $B_d=50$ meters within EAC operation zone

<table>
<thead>
<tr>
<th>Number of j-th mining phase</th>
<th>$H_{ij}$ – depth of coal mine, meters</th>
<th>$S_{kj}$ – coal reserves on upper extraction pit banks, tons</th>
<th>$L_{ykj}$ – coal reserves on intermediate extraction pit bank, tons</th>
<th>$S_{aj}$ – number of intact pit banks within EAC operation, meters</th>
<th>$L_{vyj}$ – slope angle of lower part of strippin g zone, degrees</th>
<th>$S_{cj}$ – area of lower part of stripping zone within EAC operation, $m^2$</th>
<th>$L_{yij}$ – volume of lower part of stripping zone within EAC operation, $m^3$</th>
<th>$S_{hij}$ – number of stripping banks, pc.</th>
<th>$L_{yij}$, m</th>
<th>$Z_{hij}$ – coal reserves in initial cut, tons</th>
<th>$Z_j$ – coal reserves, tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>250</td>
<td>187</td>
<td>2793,3</td>
<td>370</td>
<td>2764,5</td>
<td>16672699</td>
<td>4700</td>
<td>2742,5</td>
<td>21912575</td>
<td>1158</td>
<td>2720,3</td>
</tr>
<tr>
<td>2</td>
<td>275</td>
<td>377</td>
<td>2764,5</td>
<td>4700</td>
<td>2720,3</td>
<td>2109526</td>
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<td>21735197</td>
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</tr>
<tr>
<td>4</td>
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<td>377</td>
<td>2720,3</td>
<td>4700</td>
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<td>20670910</td>
<td>4700</td>
<td>2676,1</td>
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<td>1158</td>
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</tr>
</tbody>
</table>

### Table 5 - Continuation of calculations with cross panel width $B_d=50$ meters within EAC operation zone

<table>
<thead>
<tr>
<th>Number of j-th mining phase</th>
<th>$H_{ij}$ – depth of coal mine, meters</th>
<th>$N_{naj}$ – number of stripping sub-benches during mining with EAC, pc.</th>
<th>$H_{naj}$ – height of lower part of stripping zone within EAC operation, meters</th>
<th>$B_{naj}$ – Width of initial cut within EAC operation, meters</th>
<th>$N_{naj}$ – number of safety benches, pc.</th>
<th>$\gamma_{naj}$ – slope angle of lower part of stripping zone, degrees</th>
<th>$S_{naj}$ – area of lower part of stripping zone within EAC operation, $m^2$</th>
<th>$V_{naj}$ – volume of lower part of stripping zone within EAC operation, $m^3$</th>
<th>$S_{vaj}$ – area of upper part of stripping zone within EAC operation, $m^2$</th>
<th>$V_{vaj}$ – volume of upper part of stripping zone within EAC operation, $m^3$</th>
<th>$k_{vaj}$ – phase d stripping ratio, $m^3$/ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>250</td>
<td>8</td>
<td>120</td>
<td>38</td>
<td>7</td>
<td>4</td>
<td>25</td>
<td>18510</td>
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<td>54017733</td>
<td>11250</td>
</tr>
<tr>
<td>2</td>
<td>275</td>
<td>10</td>
<td>150</td>
<td>34</td>
<td>9</td>
<td>5</td>
<td>24</td>
<td>20160</td>
<td>2953,2</td>
<td>59536512</td>
<td>11250</td>
</tr>
<tr>
<td>3</td>
<td>300</td>
<td>11</td>
<td>165</td>
<td>28</td>
<td>10</td>
<td>5</td>
<td>23</td>
<td>13710</td>
<td>2975,1</td>
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<td>11250</td>
</tr>
<tr>
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<td>325</td>
<td>13</td>
<td>195</td>
<td>24</td>
<td>12</td>
<td>6</td>
<td>23</td>
<td>26835</td>
<td>2996,1</td>
<td>80400343</td>
<td>11250</td>
</tr>
<tr>
<td>5</td>
<td>350</td>
<td>15</td>
<td>225</td>
<td>66</td>
<td>14</td>
<td>7</td>
<td>22</td>
<td>42450</td>
<td>3031,0</td>
<td>128665950</td>
<td>11250</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Table 6 - Completion of calculations with cross panel width $B_6 = 50$ meters within EAC operation zone

<table>
<thead>
<tr>
<th>Number of $j$-th mining phase</th>
<th>$H_j$ - depth of coal mine, meters</th>
<th>$Z_j$ - col reserves, tons</th>
<th>$A_{kij}$ - annual capacity by coal within $j$-th mining phase, tons</th>
<th>$t_{ij}$ - duration of coal mining within $j$-th mining phase, years</th>
<th>$V_{ij}$ - external overburden volume, m$^3$</th>
<th>$V_{vij}^\prime$ - calculated required annual capacity by overburden within $j$-th mining phase, m$^3$</th>
<th>$V_{vij}^\prime\prime$ - first iteration of required annual capacity by overburden within $j$-th mining phase, m$^3$</th>
<th>$V_{vij}^\prime\prime\prime$ - second iteration of required annual capacity by overburden within $j$-th mining phase, m$^3$</th>
<th>$K_{vij}$ - current stripping ratio within $j$-th mining phase, m$^3$/tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>250</td>
<td>52110858</td>
<td>18500000</td>
<td>2.81</td>
<td>87876858</td>
<td>31272903</td>
<td>28030197</td>
<td>28030197</td>
<td>28030197</td>
</tr>
<tr>
<td>2</td>
<td>275</td>
<td>64336539</td>
<td>20000000</td>
<td>3.21</td>
<td>93786012</td>
<td>29216826</td>
<td>28030197</td>
<td>28030197</td>
<td>32097448</td>
</tr>
<tr>
<td>3</td>
<td>300</td>
<td>63816999</td>
<td>20000000</td>
<td>3.19</td>
<td>75286746</td>
<td>23600861</td>
<td>28030197</td>
<td>3616498</td>
<td>32097448</td>
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<tr>
<td>4</td>
<td>325</td>
<td>63297459</td>
<td>20000000</td>
<td>3.16</td>
<td>115134718</td>
<td>36435037</td>
<td>44299199</td>
<td>3616498</td>
<td>3616498</td>
</tr>
<tr>
<td>5</td>
<td>350</td>
<td>62777919</td>
<td>20000000</td>
<td>3.14</td>
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<td>52163360</td>
<td>44299199</td>
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<td>Total</td>
<td></td>
<td>306339774</td>
<td></td>
<td></td>
<td>535877284</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Table 7 – Scheduling of mining operations

<table>
<thead>
<tr>
<th>Indexes</th>
<th>Calendar years</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>$A_{ki}$, mln. tons</td>
<td>18</td>
<td>18</td>
</tr>
<tr>
<td>$V_{ami}$, mln. m³</td>
<td>28</td>
<td>28</td>
</tr>
<tr>
<td>$V_{vti}$, mln. m³</td>
<td>28</td>
<td>28</td>
</tr>
<tr>
<td>$k_{vti}$, m³/tons</td>
<td>1,55</td>
<td>1,55</td>
</tr>
<tr>
<td>Pre-design $V_{vti}$, mln. m³</td>
<td>33</td>
<td>35,2</td>
</tr>
<tr>
<td>Pre-design $k_{vti}$, m³/tons</td>
<td>1,83</td>
<td>1,95</td>
</tr>
<tr>
<td>Decrease of auto overburden volumes, mln. m³</td>
<td>5</td>
<td>7,2</td>
</tr>
</tbody>
</table>
Table 8 – Broad calculation of expected economic effect from implementation of offered technology of mining high pit banks by cross panels with automobile transport on “Vostochnyi” open pit mine

<table>
<thead>
<tr>
<th>Indexes</th>
<th>units</th>
<th>By years until the open pit depth of 350 meters</th>
</tr>
</thead>
<tbody>
<tr>
<td>Reduction of auto overburden</td>
<td>mln. m³</td>
<td>5</td>
</tr>
<tr>
<td>Economy from reduction of auto overburden</td>
<td>mln. tenge</td>
<td>718.8</td>
</tr>
<tr>
<td>Preparation to excavation</td>
<td>mln. tenge</td>
<td>139.2</td>
</tr>
<tr>
<td>excavation</td>
<td>mln. tenge</td>
<td>153.7</td>
</tr>
<tr>
<td>transportation</td>
<td>mln. tenge</td>
<td>287.9</td>
</tr>
<tr>
<td>design</td>
<td>mln. tenge</td>
<td>54.8</td>
</tr>
<tr>
<td>Road construction</td>
<td>mln. tenge</td>
<td>37.2</td>
</tr>
<tr>
<td>Road maintenance</td>
<td>mln. tenge</td>
<td>46.0</td>
</tr>
<tr>
<td>Implementation year</td>
<td></td>
<td>0</td>
</tr>
<tr>
<td>Resulted economy</td>
<td>mln. tenge</td>
<td>718.8</td>
</tr>
<tr>
<td>The resulted economy with a growing total</td>
<td>mln. tenge</td>
<td>718.8</td>
</tr>
</tbody>
</table>
The approbation of the offered algorithm of optimizing the position of the operating open pit edge shows that, by the offered scheme of the stripping zone development, the volume of stripping operations from the 250 meters depth to a depth of 350 meters in comparison with the prospective design scheme is 141.1 million m$^3$ less, while the average stripping ratio decreases from 2.2 to 1.75 m$^3$ / tons (25.7%).

The analysis of the mining schedule (Table 7) shows that the values of the current stripping ratio vary from 1.69 to 1.87 m$^3$ / tons for 16 years with a mean operating stripping ratio of 1.75 m$^3$ / tons. The maximum value of the current stripping ratio exceeds the average stripping ratio only by 6.4%; i.e. an optimal mode of mining is achieved.

4. Conclusions

Designed and actual performance of the open pit mine often differ from the project schedule of mining operations, depending on demand changes for marketable products and the reliability of mining equipment. It is possible to consider the residual prepared coal reserves in all three layers in each of the four coal blocks by the first mining phase and the overburden parameters within the boundaries of the EAC and ERC operation while obtaining data on the actual position of the mining operations in the initial data.

The option of the formation of the operational front of the working zone pit banks perpendicular to the operational front of the pit banks of the steep operating open pit edge leaving only haulage and safety benches would be studied on the cyclogram when providing data on the performance of excavator automobile complexes. It is recommended for construction of a stable hole, at this stage, on one of the flanks of the open pit field and switch to the option of mining high pit banks by cross panels from two levels of an excavator position.

The broad calculation of the expected economic effect from the implementation of the offered technology of mining high pit banks by cross panels in the lower part of the stripping zone using excavator automobile complexes shows that in 5 years the economic effect would be no less than 4.9 billion tenge, over 10 years - 11 billion tenge, for 16 years - 14.7 billion tenge.

References


MPES2017

Session 2 – Mine Equipment I

13:30-15:00

August 29, 2017
Comparison of Alternative Truck Maintenance Strategies using Simulation

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University of Western Australia, Perth, WA, Australia

Abstract— Recently the Australian mining industry has focussed on improving the efficiency of existing operations and lowering production costs. This study explored alternative surface mining truck maintenance strategies, namely condition based maintenance (CBM) and clustered maintenance (CM). The strategies were compared using performance metrics such as truck availability, workshop utilisation and efficiency.

The study examined the application of CBM and combined CB and CM maintenance (CBCM) through computer modelling and simulation. Multiple models, each representing a single maintenance strategy, were constructed. Initial results were analysed before the completion of a sensitivity analysis, which established the influence of key variables on maintenance strategy performance.

The study shows that both CBM and CBCM reduced workshop utilisation, increasing truck availability by 3% and 5%, respectively, over the base case. CBCM provides the greatest reduction in individual maintenance bookings, substantially reducing workshop related delays and increasing workshop efficiency.

Further analysis of CBCM showed that the critical condition threshold governs maintenance strategy performance, while the diagnostic condition threshold largely affects clustering efficiency.

The study concludes that operators who implement CBM should realise significant benefits in terms of both increased production, due to higher truck availability, and lower operating costs.

Keywords: condition cluster based maintenance strategy

1 Introduction

Changes in the economic drive and sensitivity of the Australian mining industry have caused many operators to focus on lowering production costs. Cost centres usually considered in this process are the number of employees, training, production expansions and contractor requirements, however, the cost of mobile plant maintenance is often not considered.

Perhaps the most untouched production cost is that of load-haul fleet maintenance, which may account for up to 60% of total operational cost (Chamzini, et al., 2012).

2.1 Objectives and Scope

The objective of this project was to explore the combination of alternative Surface Mining Trucks (SMT) maintenance strategies; particularly the combination of preventative maintenance, condition based maintenance and clustered maintenance. This was achieved through mine site computer modelling, simulation, and fleet performance analysis.

2.2 Maintenance Strategies Currently In Use

Currently, the global mining industry uses two broad types of SMT maintenance strategies; breakdown and preventive maintenance. Each strategy attempts to achieve the same goal; to maintain expected SMT performance while being cost effective. Maintenance strategies currently in use are a variation of these two broad types. Strategies based on company standards and operational conditions are typically selected (Mueller, 1995).

2.2.1 Breakdown Maintenance

Breakdown strategies call for maintenance works only when an asset fails. This approach is highly unpredictable, making workshop resource allocation difficult and affecting mine production significantly (Chamzini, et al., 2012). From a production standpoint, this approach is highly undesirable. Both maintenance interval and service times are unknown, making cost control and budgeting difficult. Additional complications include resource allocation to particular assets, prioritising of SMTs and maintaining a sufficient parts inventory without unnecessarily tying up capital (Mueller, 1995). Breakdown maintenance was excluded from this study.

2.2.2 Preventative Maintenance (PM)

Preventative maintenance strategies, based on asset service hours and time of last maintenance, take many forms. Perhaps the most common example is when assets are serviced every 250 operating hours. This approach is well developed and is almost exclusively the approach specified by manufacturers’ recommended maintenance strategies. Preventative maintenance aims at mitigating asset breakdown by inspecting and replacing fluids and components prior to
their complete degradation (Mueller, 1995). This approach incurs extra cost as the component replacement occurs whilst it is still operating normally. However, even by following the manufacturers’ service schedule, or other preventative maintenance schedules, ad hoc breakdowns still occur (Chan & Kuruppu, 2006).

2.2.3 Condition Based Maintenance (CBM)

A variation of preventative maintenance is CBM, where the condition, or health, of an asset is monitored. Equipment is serviced based on its actual and current requirements (Faitakis, et al., 2004). The focus of CBM is to reduce unnecessary maintenance works, as maintenance actions are necessary only when there is sufficient evidence to suggest abnormal equipment operation (Banjevic, et al., 2006). In this approach, maintenance occurs when equipment shows signs of decreased performance or catastrophic failure, leading to the establishment of a dynamic service schedule (Marzio, et al., 2002). The dynamic schedule and the predictive nature of maintenance helps to reduce spare part inventory, optimise maintenance resources, minimise equipment downtime, and most importantly, eliminates premature replacement of spare parts. Thus, CBM works towards maintaining equipment availability while reducing overall maintenance cost, over the life of the asset.

CBM determines equipment health through non-invasive, and often remotely monitored, measurement of key mechanical components. Maintenance is scheduled when key performance indicators exceed previously determined ‘safe’ limits. This approach permits the prediction of failure before it has occurred, allowing the equipment to be shut down normally and serviced as required, as opposed to failure before service (Faitakis, et al., 2004). The CBM approach is shown diagrammatically in Figure 1, where maintenance is triggered for any component condition metric entering the orange zone. Only the component that triggered the maintenance is serviced.

2.2.4 Clustered Maintenance (CM)

Another variation of preventative maintenance is CM, where individual maintenance tasks are grouped into larger workshop bookings.

Clustered maintenance is a relatively new strategy, and as a result, little literature is available. After review it is apparent that effective implementation of clustered maintenance is limited to the airline industry. Useful literature is limited to a single paper, written by Muchiri & Smit (2011), with the detailed study occurring on a Boeing 737.

The result of re-clustering maintenance tasks, and the subsequent variation of maintenance intervals, resulted in a reduction in maintenance cost of 7.3% across the life of the asset, whilst maintaining aircraft availability (Muchiri & Smit, 2011).

The variation of CM that was considered in this study has been called Condition Based Clustered Maintenance (CBCM). CBCM is similar to CBM in that a service is triggered whenever a component condition metric exceeds a critical threshold value. But with CBCM, differently to CBM, other components, which exceed a different and lower threshold, are also serviced. The CBCM approach is shown diagrammatically in Figure 2, where maintenance is triggered for any component condition metric entering the orange zone. Differently from CBM, any component with a condition in the yellow zone is also serviced.

2. Model Development

The key to exploring the application of CBM and CBCM to SMTs is computer modelling and simulation. This study uses three representative models to study the effects of these maintenance strategies.

Realistic representation of truck shovel operating systems requires consideration of multiple model elements. Investigation into the application of CBM and CM strategies requires three models, each modelling a specific SMT maintenance strategy. The following section focusses on the development of each model.

The software used for this study was Arena from Rockwell Automation.

2.1. Mine Site Model

As a representation of a real world scenario, the Mine Site Model (MSM) represents a typical layout of open pit mines. The following load haul tasks were modelled:

- Loading of SMTs via shovels, with the inclusion of a loading queue;
- Inclined, declined and flat SMT haulage routes to tipping points (crushers or waste dumps), shovels and the workshop; and
- Dumping loads.

The following aspects of the workshop were also included:
• Travel time to and from the workshop; and
• Workshop queue and bay allocation.

The model included a fleet of 20 SMTs and three shovels, serviced by a three bay workshop. This was the basic framework for all model variations. Variations were limited to implementation of maintenance decision criteria and were essentially logic statements.

2.2. Base Case Model

The base case model incorporates the MSM with the inclusion of the industry standard preventative maintenance strategy. Routing of a SMT to the workshop occurs every 250 operating hours. Maintenance work is carried out on all maintenance intensive subsystems of the SMT. This maintenance strategy was the basis for comparison of this study as it is widely used within the mining industry. Figure 3 details model logic concerning maintenance strategy decisions.

2.3. CBM Model

Modelling CBM requires splitting a SMT into its major subsystems, imitating condition monitoring of multiple components:
• Structural (body and frame integrity);
• Electrical;
• Hydraulic;
• Mechanical; and
• Cooling (engine).

The model logic is almost identical to that of the base case model, the only difference being the initiation of a maintenance booking. Figure 4 details this new model logic.

Figure 4: Logic used in the CBM model.

MTBF data, sourced from literature and past performance, forms the basis of maintenance interval estimation for each subsystem.

This condition monitoring and maintenance initiation process occurs simultaneously, and independently, for each SMT subsystem. Maintenance triggers are essentially point sources for maintenance initiation. Thus, all subsystems will be maintained separately, unless two or more triggers occur simultaneously. Evidently, the nature of CBM is averse to grouping maintenance tasks.

2.4. CBCM Model

Modelling CBM and CM (CBCM) requires the implementation of additional maintenance decision logic. The CBCM model utilises the CBM model, including only additional CM attributes, in the form of diagnostic and critical condition thresholds (DCT, CCT).

CM works by applying DCTs and CTTs, which exist between ideal operation and the development of a functional fault. Figure 5 shows typical DCTs and CCTs in relation to ideal system operation and a functional fault.

Figure 5: Logic used in the CBCM model.

Determination of the DCT and CCT depends on how aggressively CM is implemented. An aggressive
implementation will set the CCT value close to that of the functional fault, seeking to increase availability, but with a higher risk of failure.

The value of the DCT is determined largely by economic decisions regarding maintenance cost. A value close to ideal operation increases maintenance trigger range and decreases the minimum maintenance interval. This means that a subsystem will be included in a greater number of maintenance bookings and may undergo unnecessary maintenance works, incurring unnecessary additional costs. Setting the DCT value further from ideal operation reduces the maintenance trigger range and increases the minimum maintenance interval. This reduces the number of bookings a subsystem is included in and incurs less cost over the life of the asset. It is worth noting that the risk of breakdown is not affected by the values of the DCT, as the CCT governs the maximum maintenance interval.

3. Model Verification and Validation

Performing model verification and validation was crucial to this study. Without performing verification and validation, any results obtained from the model are questionable and likely to be erroneous (Law, 2008).

3.1. Model Verification

Arena simulation software allows model verification to occur graphically. This is especially useful when checking the complex programming involved in CBM and CM model logic. The following process was utilised to verify each model:

- Reduce the SMT fleet size to one, making it easier to track entities throughout the simulation;
- Replace all PDFs with fixed numbers and calculate when a SMT should enter the workshop and for how long. Run the simulation and ensure this happens;
- Increase fleet size to 40 in order to ensure balkng and queues operate correctly. A fleet of this size is deemed sufficient to verify model operation without making entity tracking difficult; and
- Replace fixed numbers with appropriate PDFs and use estimates to verify Arena correctly calculates PDFs.

This process exposed multiple errors in model programming, particularly the methods Arena uses to calculate PDF values. It was realised that Arena uses non-standard PDF equations; therefore, manipulation of source data was required to ensure correct execution.

3.2. Model Validation

Validation of models can take many forms, ranging from comparison with historical data to comparison with well-documented theories. Comparison of the MSM to specific historical data would produce misleading results, as the MSM represents a generalised mining system. Thus, validation against historical data is not possible for this study. Given the study’s focus on maintenance, and all maintenance related parameters, it is unnecessary to validate other aspects of the MSM (SMT queuing, utilisation, dispatching, etc.). Additionally, validation of CBM and CBCM models is impossible due to the lack of implementation of these maintenance strategies to SMTs.

Thus, model validation utilises the base case as this is a well known, well-documented maintenance strategy in use throughout the mining industry. Typical implementations of this maintenance strategy, in industry, realise fleet availabilities of 85 - 90% (Woodrow, 1992).

The base case model reports fleet availability of 87.3%, well within industry expectations of a 250-hour preventative maintenance schedule. This validates maintenance aspects of the base case model, allowing real comparison of CBM and CBCM models.

3.3. Experimental Conditions

Experimental conditions must be established for stochastic models run in Arena. Arena has two major settings, warm-up period and the number of model replications.

3.3.1. Warm-up Period

The warm-up period refers to the time a simulation runs prior to data collection. This is effectively utilised when studying the steady state of a modelled system, as no data recording occurs during the warm-up period. Additionally, warm-up periods are utilised when the initial state of a modelled system is unlikely or does not represent the initial state of the real system, thus data collection should not occur (Grassman, 2008).

The MSM reaches a steady state after approximately 15 hours. This is an insignificant period considering a simulation time of 87,600 hours, thus negating the need for a designated warm-up period.

3.3.2. Simulation Replications

Stochastic characteristics of each model create variation and a degree of randomness in each execution. Multiple replications of each simulation eliminate statistical bias. Thus, the required statistical accuracy of the study determines the number of replications required. This study will use a two-sided 95% confidence interval (CI), calculated using:

$$CI (95\%) = \bar{x} \pm t_{\alpha/2} \left( \frac{\sigma}{\sqrt{n}} \right)$$

The required number of replications was determined using the base case model, with each replication being the full simulation time (87,600 hours). Key outputs subject to statistical variability are utilised in determining satisfaction of CI. Additionally, the margin of error associated with each output was calculated. This is summarised in table 1.

Table 1: Margin of error for a 95% CI (10 replications).
In order to satisfy a 95% CI and reduce error this study utilised 10 replications per model.

4. Results and Discussions

Various performance metrics were used to compare the different models of maintenance strategy.

4.1. Maintenance Bookings

The number of individual maintenance bookings, including a per subsystem breakdown.

Maintaining reliability of SMTs when demonstrating alternative maintenance strategies is fundamental to widespread adoption in the industry. The number of maintenance bookings may be treated as a proxy to fleet reliability, and, logically, increased chance of breakdown follows a significant reduction in maintenance events. However, this claim assumes that components do not suffer wear-in failures.

Figure 6 summarises the number of maintenance bookings per SMT for each maintenance strategy.

Figure 6: The number of maintenance bookings, and total service time per SMT over 10 years

4.2. Workshop Delays

There are three major components to workshop delays; sign on time, sign off time and queue time. These occur with each booking, and thus are a function of the number of bookings. These delays represent SMT idle time and thus should be minimised as much as practicable. Figure 8 displays a breakdown of total workshop delay for each maintenance strategy.

Figure 8: A breakdown of the total SMT delay time for each maintenance strategy.

4.3. Workshop Utilisation & Efficiency

Ideally, alternative strategies raise workshop efficiency and require little or no change to existing workshop infrastructure (capacity) or staffing. Workshop utilisation and efficiency is calculated using the following equations.

Figure 9: Workshop utilisation and efficiency for each maintenance strategy.

4.4. SMT Availability

---

**Table 1:**

<table>
<thead>
<tr>
<th>Output</th>
<th>Mean</th>
<th>Standard Deviation (σ)</th>
<th>Margin of Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fleet availability</td>
<td>87.2%</td>
<td>0.001</td>
<td>0.06%</td>
</tr>
<tr>
<td>Fleet service hours</td>
<td>189,056.8</td>
<td>15.99</td>
<td>2.71%</td>
</tr>
<tr>
<td>Fleet sign on delay</td>
<td>8,722.84</td>
<td>16.46</td>
<td>2.43%</td>
</tr>
<tr>
<td>Fleet sign off delay</td>
<td>8,315.43</td>
<td>11.42</td>
<td>2.55%</td>
</tr>
</tbody>
</table>

4.1.1. Subsystem Analysis

**Figure 7:** The number of services for each subsystem per SMT over 10 years.

**Figure 9:** Workshop utilisation and efficiency for each maintenance strategy.

---
4.5. Sensitivity Analysis

4.5.1. Effects of changing Workshop Capacity

Figure 11 shows the effect of workshop capacity on utilisation and efficiency in the CBM model.

Figure 11: Workshop utilisation and efficiency for varying workshop capacity (CBM).

Figure 12: SMT availability for varying workshop capacity (CBM).

4.5.2. Effects of changing CCT

The CCT defines the maximum maintenance interval for the subsystem. As discussed in Section 2.4, the value of this threshold is largely determined by how aggressively CM is implemented. The model used in this analysis utilises the same DCT of 3% and only varies the CCT. Figure 15 shows how the number of maintenance bookings for each subsystem varies with changing CCT.

Figure 15: Number of services for each subsystem for varying CCT (per SMT over 10 years).

Increasing the CCT, thus increasing the maximum maintenance interval, results in reduced maintenance bookings for all subsystems. The model becomes decreasingly sensitive as the CCT increases. This is due to increased clustering flexibility, a result of a wider maintenance trigger
range. This is echoed in figure 16, which shows the number of individual maintenance bookings per SMT.

Figure 16: The number of individual maintenance bookings for changing CCT (per SMT over 10 years).

The effect of clustering is apparent with increasing CCT as the number of individual maintenance bookings per SMT falls. However, it is worth noting that the effect of clustering has increased sensitivity to changes in smaller CCT values. Figure 17 shows the effect on workshop delays.

Figure 17: Workshop delays for varying CCT.

As discussed above, sign on and sign off times are proportional to the number of maintenance bookings. This is reflected in figure 17. Queue time decreases rapidly, by 74.8%, between CCTs of 0.5% - 1.5%, after which the rate of reduction slows. Clearly, increasing the CCT realises decreased service time and workshop related delays. Figure 18 shows the relationship between CCT and SMT availability.

Figure 18: SMT availability with changing CCT.

SMT availability increases by 5.4% over the range 0.5% - 2.5%, after which it plateaus. Again, the greatest change is seen at lower CCTs, with availability increasing 2.4% over the interval 0.5% - 1%. It is important to note that these SMT availability figures (particularly 97% availability using a CCT of 2.5% - 3%) are unrealistic, however, results effectively demonstrate the effect of the CCT on CBCM. Workshop utilisation and efficiency is shown in figure 19.

Figure 19: Workshop utilisation and efficiency for varying CCT.

Efficiency of the workshop increases gradually with increasing CCT values and is due to reduced delay times. Workshop utilisation decreases with increasing CCT values, as expected. The rate of reduction is relatively consistent across the modelled interval; however, smaller CCTs still offer the greatest rate of reduction, 13.6% over the interval 0.5% - 1% compared to 7% over the interval 1% - 1.5%. Increased sensitivity over these intervals will mostly affect operators implementing a conservative CBCM strategy. Careful consideration of required workshop capacity is necessary, as workshop utilisation will climb significantly with slight reductions in the CCT.

The trend in sensitivity raises one major question when increasing the CCT beyond 1.5%, 'is the marginal benefit of further increasing the CCT worth the increased risk of catastrophic SMT failure?' The answer to this question is beyond the scope of this study, as it requires an extensive SMT reliability study. However, this is an aspect of CBCM that needs to be investigated prior to a physical implementation.

4.5.3. Effects of changing DCT

The DCT defines the minimum maintenance interval for the subsystem. As suggested in Section 2.4, the value of this threshold is largely determined by economic decisions regarding maintenance cost. In this analysis, the CCT was increased to 2% (from 0.5%) due to computational restrictions present in Arena software. However, the effect of changing the DCT may still be analysed. Figure 20 shows how the number of maintenance bookings for each subsystem varies with changing DCT.
As the DCT increases so do the number of maintenance bookings for each subsystem. The model is consistently sensitive to changes with no uncharacteristic jumps in service numbers noted. The overall trend is consistent with the suggestion in Section 2.4 that increasing the DCT will cause a subsystem to be included in a greater number of maintenance bookings. Again, this is due to increased clustering flexibility, a result of a wider maintenance trigger range. This is consistent with the overall trend shown in figure 21.

Figure 21: The number of individual maintenance bookings for changing DCT (per SMT over 10 years).

Similar to variation in the CCT, the effect of clustering is increased with increasing DCT. However, the interval of greatest effect has shifted. The greatest decrease in clustering effect is seen in the 1.5% - 2% interval, decreasing bookings by 2.9% compared to 1.4% in the 1% - 1.5% interval. This suggests that the DCT has more power in governing the efficiency of clustering than the CCT. Figure 22 shows the effect on workshop delays.

Figure 22: Workshop delays for varying DCT.

Sign on and sign off delays decrease 3.9% and 4.6%, respectively, over the interval 1% - 2%. Queue time fluctuates by up to 18.8% with increasing DCT. The exact source of this fluctuation is unknown, however, it can be postulated that it is related to the duration of maintenance bookings and workshop queuing criteria. Figure 23 shows the relationship between CCT and SMT availability.

Figure 23: SMT availability with changing DCT.

Surprisingly, variation in the DCT has no effect on SMT availability. The increase in subsystem services with increasing DCT, seen in figure 20, would suggest a reduction in SMT availability. The exact reason for constant availability is unclear; however, it is possibly due the fact that the DCT governs only the minimum maintenance interval. This implies that it is only the model’s efficiency of clustering that may affect workshop utilisation and efficiency. Workshop utilisation and efficiency is shown in figure 24.

Figure 24: Workshop utilisation and efficiency for varying DCT.
Workshop efficiency remains relatively constant, only increasing 0.4% over the DCT interval of 1% - 3%. Workshop utilisation increases 1% over the same interval, as a direct result of an increased number of subsystem services. The most pertinent observation is insensitivity of key CBCM performance metrics to changes in the DCT. The only aspect of CBCM sensitive to changes in the DCT is the ability to cluster services. However, maintenance cost must still be analysed. Figure 25 shows the relationship between maintenance cost and DCT.

![Figure 25: Maintenance cost variation with changing DCT.](image)

Clearly, the cost of consumables, as a result of an increase number of services per subsystem, outweighs cost savings made through clustering of maintenance tasks. The cost relationship is linear with maintenance cost increasing 0.1 cents per tonne (3.5%) over the interval 1% - 3%. This amounts to approximately $0.2 million cost increase per year. However, this cost increase is minimal when compared to cost savings made by increasing CCT.

It becomes increasingly clear that the DCT should be set by decisions based on the desired clustering performance. CBCM clustering performance is important as decreased clustering gives rise to increased workshop throughput and may introduce logistical issues on site, such as difficulty in SMT dispatching. Evidently, performance of CBCM, in terms of SMT availability and cost is primarily governed by the CCT, not the DCT. Additionally, the cost increase of increasing the DCT represents only 5% of potential cost savings made by increasing the CCT. Ultimately, the decision of the DCT is the operator’s preference, as performance of CBCM is largely independent of this value.

4. Research Summary

As seen in the literature review, there is a lack of application of CBM and CBCM strategy to SMTs. This study contributes to the development of maintenance strategies in the mining industry by studying the effects, through simulation, of CBM and CBCM on SMTs.

Arena simulation software was used to construct representative models of a mine site and allowed the implementation of these alternative maintenance strategies to be studied. Models were validated by comparison to typical industry values and expectations. Three models were constructed, each modelling a different maintenance strategy. These were:

- The base case, which used the industry standard 250 hour preventative maintenance schedule;
- CBM, which modelled condition monitoring and maintenance of five SMT subsystems; and
- CBCM, which included additional maintenance decision logic in order to cluster maintenance bookings.

The construction of these models allowed effective comparisons and conclusions to be drawn with relation to SMT and workshop performance. The results conclude that;

- CBCM provides the greatest reduction in individual maintenance bookings;
- Workshop delays are reduced substantially when using CBCM, resulting in increased efficiency and lower utilisation when compared to both the base case and CBM;
- CBM and CBCM positively affect SMT availability; and
- CBM is the most cost effective maintenance strategy, offering annual savings of $8.3 million when compared to the base case.

The implications of these conclusions are;

- Operators will realise SMT production gains due to increased SMT availability by implementing CBM or CBCM;
- Operators are able to increase SMT fleet size and maintain availability without upgrading workshop capacity, infrastructure or increasing staff numbers. This is particularly pertinent given the recent push in the industry to ramp up production;
- Greenfields operations can reduce capital expenditure by initially building smaller workshops and thus requiring less maintenance staff, whilst upholding fleet maintenance requirements;
- When implementing CBCM operators need to actively consider the marginal cost benefit versus the risk of catastrophic SMT failure when assigning condition thresholds; and
- By implementing alternative SMT maintenance strategies operators are able to make considerable cost savings while improving the efficiency of the maintenance operation.

4.1. Research Contributions and Limitations

This research has successfully simulated the alternative maintenance strategies of CBM and CBCM to SMTs. A number of models have been constructed and conclusions drawn in investigating the potential benefits of such strategies. Ultimately, this research contributes to the mining industry by;

- Providing initial confirmation that CBM and CBCM is advantageous when applied to SMTs;
- Demonstrating that alternative maintenance strategies are beneficial to SMT performance and workshop efficiency; and
• Providing a foundation for development of alternative maintenance strategies, which are able to take advantage of current technology while satisfying performance requirements of the contemporary mining industry.

However, when moving forward, it is important to recognise the limitations of this research. Limitations are due to assumptions in model construction and inputs. Additionally, results are based entirely on stochastic models, and while every effort has been made to ensure they represent reality, there will always be small dissimilarities. Further limitations include:

• Data suitability
  MTBF and TTR data was sourced from two academic papers. While this data is based on tangible reliability studies of SMTs it is not necessarily suited for application in modelling.

• Lack of primary data
  It was not possible to attain primary, real world data, for analysis and use in maintenance interval and TTR parameters. Primary data is preferred as this allows better model validation and improves the overall quality of the study.

References
Cyber-Physical Mining Systems

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Abstract—In times of deeper and more complex ore bodies, more competitive economic settings and increasing requirements regarding health, safety and environment, there is a need for the development of advanced mining technologies to meet the future challenges and needs. With recent advances in robotic sensing and understanding of the environment, current approaches focus on providing full automation and remote-control operation of existing mining machinery with fallback options of manual operation. However, the smart and intelligent, safe and high-performance “Mine of the Future” demands more advanced and integrated systems.

Our innovative approach “Cyber-Physical Mining Systems” uses a systems engineering approach to cover the entire value chain from the mine site to the processing plant as a system with several subsystems. All subsystems should be fully autonomous instead of fully automated, meaning that they are able to learn and find solutions within given boundaries. Hence a complete redesign of the whole mining process from scratch is reasonable and required.

This presentation gives an overview of mining concepts, designs and off the shelf technology to be reengineered for more flexible, efficient and fail-safe operations suitable for future land-based mining operations as well as projects far beyond, like deep-sea and asteroid mining. Furthermore, recommendations for a roadmap towards autonomous mining as well as training requirements for future Mining Engineers will be given.

Keywords: Autonomous Mining Systems, CPMS, Mining Engineering education, Mine ventilation

1. Introduction
The global mining industry is facing perceptible challenges due to an increase in the depth of mining operations, an increase related to the complexity of deposit structures with a more extensive network of excavations, rising energy and operational costs as well as stricter regulations on mine health and safety, environment as well as social-license-to-operate and not least in terms of an upcoming aging workforce and skills crisis [1–8]. Therefore, the mining industry is obliged to develop advanced and innovative mining technologies and solutions for ensuring safe, environmentally-friendly and economically feasible mining operations while at the same time ensuring a safe and secure access to and supply of raw materials. Examples for holistic concepts addressing these challenges are the “Mine of the Future” [9] or “Smart Mine of the Future” [5,6]. However, future developments such as deep-sea mining or extra-terrestrial Mining (Moon, Mars, Asteroids) demand for truly autonomous systems because of the remoteness and therefore for very limited or nor real-time communication for remote control. With advances in computer science and engineering there is a clear trend towards ubiquitous cyber physical systems as a tight conjoining of and coordination between computational and physical resources. Cyber-physical systems of tomorrow will “far exceed those of today in terms of adaptability, autonomy, efficiency, functionality, reliability, safety, and usability” [10]. Furthermore, it is of vital importance that the whole mining process is treated holistically as an integrated process from exploration to processing.

Therefore, section two introduces the concepts and components of cyber-physical mining systems and delimitations to typical systems for mine automation. Due to the importance of mine ventilation systems, which can be described as the “lifeblood of a mine” [11] and which makes up a major fraction of a mine’s total energy consumption - 40% to 50% in most cases [12,13]. In section three the general concept of CPMS will be exemplary applied to the area of mine ventilation. For a successful implementation and integration of autonomous systems and CPMS education concepts for the qualification of future mining engineers need to be reviewed; section 4 will illuminate some implications to the training and education requirements for future Mining Engineers before the paper will conclude with a summary and outlook on further activities in section 5.

2. Cyber-Physical Mining Systems: from Mine Automation towards Autonomous Mining Systems
2.1. Cyber-Physical Systems (CPS)
The term cyber-physical systems (CPS) refers to the “tight conjoining of and coordination between computational and physical resources” [10]. CPSs build upon the core technologies CCC – computation, communication, control – and are fully integrated and embedded in the systems environment (Figure 1). Information will be obtained from physical processes, modeled, simulated and optimized in real-time, dynamically controlled and made available to all objects in the network. Physical components comprise all objects in
the system, sensors and actuators, requiring dynamics, concurrency as well as timing. The (mining) system itself includes a holistic systems engineering approach taking into consideration the context management as well as relationships between the different objects and entities in the system. All information will be made available within the cyber system, comprising process modeling, (real-time) simulation, optimization and the communication within the network. In contrast to embedded systems CPSs are not bound to predetermined classical process measuring and control techniques and therefore proceed further in terms of functionalities, robustness, adaptation, intelligence and interconnectivity [14,15].

![Figure 1: Components of Cyber-Physical Mining Systems (CPMS)](image)

2.2. Differentiation to Mine Automation

Mine automation focuses on enabling machines to perform repetitive tasks on its own or use remote control for more difficult tasks. Therefore, mine automation is very limited in highly complex and variable environments (Figure 2) since as a system it reacts on sensor input.

However, CPMS comprises the development and deployment of fully autonomous mining systems. The main difference is to use true self learning autonomous systems based on machine learning and other tools from artificial intelligence instead of using a traditional systems engineering approach with handcrafted decision rules within strong operational boundaries. The AI focused approach aims at cloning human behavior and leads to increased robustness regarding system disturbances and dealing with unknown situations [16–21].

In the following chapters examples are given how CPMS could be applied.

![Figure 2: CPMS in the Context of Mine Automation and Traditional Mining](image)

2.2.1. Redesign of Mine Planning & Mining Processes

In general, future mining processes can be more adaptive due to real-time data acquisition and using predictive models for data analysis. Therefore, a mining operation should be treated as a system itself. The application of CPMS on a mining value-chain level comprises more than just machinery. A key component is improved real-time exploration resulting in continuous self-adjusting mine planning within the overall limits of the mining operation.

When it comes to excavation machinery, self-learning and self-configuring systems will be able to handle complex and highly variable situations and therefore realize “zero-entry production areas” [22]. Moreover, the excavation performance can be measured and therefore the haulage can be fitted appropriately.

In a surface mining operation the deployment of autonomous haul trucks is expected to improve efficiency by 14 % without changing other parameters [23]. Moreover, with real-time monitoring of road conditions, the performances of haul trucks can be predicted and considered in haulage planning. Hence with truly autonomous and integrated haul trucks, material handling with discrete transport elements can achieve a quasi-continuous state which is more robust to system disturbances.

2.2.2. Redesign of Mining Machinery

Another key component of CPMS is the total redesign of mining machinery. Today’s machinery is designed to be operated by humans, hence key elements rely heavily on eye vision and a person and therefore a cab. In case of a LHD truck with a side mounted cab, the required space for visibility limits the width of the scoop. Today’s remote-control solutions are usually build on top of existing machinery leaving general design of mining machinery unchanged. Considering a truly autonomous the cab itself is just an empty space. Hence a cab of a drilling jumbo or a roadheader could be reequipped with rock logging devices for exploration while drilling.

Real-time analysis and predictions based on structural self-monitoring of all machinery and structures [24–27] allows for preventive maintenance and therefore reduce break
downs. With sophisticated data analysis, maintenance is not only predictive but it is possible to be scheduled to minimize systems disturbance as well.

3. CPMS Applied to Mine Ventilation

As outlined in the introduction, mine ventilation systems can be described as the “lifeblood of a mine, the intake airways being arteries that carry fresh air to the working areas and the return veins that conduct pollutants away to be expelled to the outside atmosphere” [11]. Therefore, an effective and efficient ventilation system concept tailored to individual operation is necessary for guaranteeing safe working conditions in underground mining operations. However, with increasing complexity and simultaneous efforts in terms of CPMS while at the same time stricter regulations for mine health, safety and environment, like the discussion about significant reductions of threshold limit values for nitrous gases [28], new holistic ventilation concepts and planning as well as control tools need to be developed. Current approaches focus mainly on the development and implementation of ventilation on demand (VOD) concepts and systems [29]. To further enhance the integration of and interaction between production, infrastructure and logistic, maintenance and ventilation system in terms of CPMS the following approach, which is demonstrated as a conceptual sketch in figure 3 is proposed. In general, the system proposed is not reactive based on sensor input as most of the VOD approaches are, but a utilizes and integrates predictive modeling, simulation and optimization.

4. Implications to Mining Engineering education

The Higher Education environment has been significantly reformed during the last decade while at the same time employee’s expectations towards graduates changed tremendously. Furthermore, due to the 4th industrial revolution, which was outlined above, with the creation of “smart mining factories” with cyber-physical mining systems, in which computers, automation, robotics and humans will be connected in an entirely new way, also the skillset of future mining engineers needs to be adapted for coping with these new challenges and future work environments. The overall objective is to educate confident graduates with excellent professionalism, leadership, critical thinking, communication and additional skills important for a successful transition to the world of work. Resulting demands for university education in Mining Engineering nowadays cannot be sufficiently addressed by traditional teaching and learning approaches, so that there is a significant requirement for

- the development and integration of innovative, (inter)active and cooperative teaching and learning concepts and methods
- fostering self-regulated and self-driven students learning
- consistent and consequent alignment to learning outcomes and acquisition of competences according to the theory of Constructive Alignment.

An example for an innovative approach in higher education for mine ventilation and climatization is the holistic teaching and learning concept InVent (“Innovations in Mine Ventilation Education”). InVent, which was established and consequently enhanced since several years by the lead author, is a holistic approach by meeting the requirements mentioned above and making a deliberate alignment between well designed learning activities suitable and appropriate for achieving the intended learning outcomes and well-considered assessment and feedback criteria and strategies. All elements, especially development and integration of active, cooperative and collaborative learning environments, build-up and implementation of a ventilation laboratory, development and integration of concepts and structures with and at a Teaching and Research as well as the integration of project-based learning with real case scenarios were established and embedded in the overall approach, each aiming at specific objectives [31–33].

Future developments will focus on the further enhancement and development of modern teaching labs specialized for innovation in key areas for the mining sector. This will also include the integration of techniques of virtual (VR) and augmented (AR) reality. Through the layering of virtual content onto the real world a completely new and applied approach can be designed [34–36]. In addition, these technologies could be used not only for higher education purposes but also for the training of professionals, especially for emergency cases.
However, any educational application requires technological, pedagogical and psychological aspects to be carefully investigated before their implementation [37].

5. Conclusions and Outlook

With the advent of ubiquitous cyber-physical systems and artificial intelligence, moving from remote control and automation towards truly autonomous systems becomes relevant also for the mining industry. Therefore, a mine should be treated as a system with numerous interconnected subsystems. The AI focused approach increases systems robustness and hence will be able to deal with more complex situations than automation. Since autonomous systems have different requirements to environmental perception and a major redesign of mining processes and planning is reasonable. Finally, this could then lead to more adaptive mining systems.

As an example, the application of CPMS to mine ventilation was given. With upcoming challenges, like meeting regulatory limits of air quality, CPMS is a suitable approach and concept for coping with these. Moreover, aside an increase in functionalities, robustness, adaptation, intelligence and interconnectivity significant energy savings are possible.

In addition, the skillset of future mining engineers and professionals need to be adapted to these arising changes and requirements. Therefore, an example for a holistic education concepts was introduced; further developments and enhancements both for production as well as education and training purposes is seen in the integration of augmented and virtual reality.

References


The main geometrical parameters of the bucket-wheel excavators SchRs 1550 used for the management of mining in real time

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Abstract—In the opencast coal mining we face the excavation, transport and the storage of a substantial volume of materials. One of the main and regular tasks for surface mines are therefore the calculations of volumes and the monitoring the progress of extraction. The main objective of the measurement is to determine the geographic position of bucket-wheel excavators and lay down their basic geometric parameters, which define their position within three-dimensional space. This way we can monitor, plan and manage mining processes better on stripping areas in coal mining sections. Thus, we can minimize the risk of soil landslides on an excavator, and avoid collision with drilling equipment and the underground mine workings. The geometric parameters of bucket-wheel excavators obtained using 3D laser scanning is one of the important parts of the whole process, which enables the management of mining in real time. The measurement is taken at the site of Doly Nástup Tušimice Mines (DNT), which come under the company Severočeské doly a.s., the largest producer of coal in the Czech Republic.

Keywords: 3D laser scanning, bucket-wheel excavator, digital terrain model, GNSS, real-time.

1. Introduction

The system of mining management in real time has been developing since 2006, and during this time it has already undergone a number of innovations. It concerned not only the area of development for the functionality of the system itself, but also the number of implementations to these huge machines. Currently, in the company Severočeské doly a.s. the system for determining the spatial position of the bucket-wheel excavators in real time is used for 20 such excavators. This article focuses on the construction’s important nodal points and the geometrical parameters of the bucket-wheel excavator, 3D laser scanning technology. Namely, it is the bucket-wheel excavator SchRs 1550/4x30 (Figure 1, Table 1), which operates in the conditions of the first stripping cut in Doly Nástup Tušimice mines, where the easily separable cap rocks prevail. The excavator is equipped with a non-telescopic wheel boom, placed on a caterpillar chassis. It consists of three basic elements. They are the bucket-wheel excavators themselves, the linking belt bridge and the loading vehicle. The linking belt is telescopic.

Figure 1. The bucket-wheel excavator SchRs 1550, the 3D laser scanning of boom using Leica ScanStation C10
Table 1 – Basic technical parameters of the bucket-wheel excavators SchRs 1550

<table>
<thead>
<tr>
<th>Parameter</th>
<th>SchRs 1550</th>
</tr>
</thead>
<tbody>
<tr>
<td>Operating weight</td>
<td>3558 t</td>
</tr>
<tr>
<td>Max. height</td>
<td>59.4 m</td>
</tr>
<tr>
<td>The length of the bucket-wheel boom</td>
<td>70 m</td>
</tr>
<tr>
<td>Cutting Height</td>
<td>30 m</td>
</tr>
<tr>
<td>The depth of the cut under the sliding ground</td>
<td>4 m</td>
</tr>
<tr>
<td>The diameter of a wheel</td>
<td>11.8 m</td>
</tr>
<tr>
<td>Theoretical output</td>
<td>5500 m³/h</td>
</tr>
</tbody>
</table>

With its dimensions and other parameters it significantly exceeds the hitherto measured bucket-wheel excavators utilizing the 3D laser scanning. For comparison e.g. the bucket-wheel excavator K800, whose weight is 1440 t, its height is 31.4 m, and the diameter of the bucket-wheel is 8 m.

2. A brief description of the system

The system for the spatial position calculation for the bucket-wheel is composed of three basic elements [8]:

- Measuring segment (GNSS receiver, inclinometer, control unit),
- communication segment (data transfer - GPRS, radio modems),
- users segment (evaluative software - Báňský model, technological model KVAS).

The technical details of the individual parts in this system are described in detail e.g. In December 2014 the bucket-wheel excavator SchRs 1550/4x30 (manufacturer Vitkovice-Prodeco and ThyssenKrupp Fördertechnik) was equipped with GNSS receivers and other sensors (Figure 2). Using these GNSS receivers (GNSS Trimble SPS 851) the position of two points on a bucket-wheel excavator can be measured, using a valid coordinate system S-JTSK, and its altitude with the height system Balt, after its adjustment (Bpv). The GNSS receivers are fixed so that they are in the same vertical plane with the centre axis of the wheel. With the inclinometers (SITALL Electronic) the bank slope of the entire excavators and the bucket-wheel boom are measured, utilizing the incremental revolution sensor (IRC LARM; a.s.) [6].

The IRC sensor is not a part of the SchRs 1550 monitoring system here, as the excavator is equipped with a fixed boom. Therefore, an important nodal point is the axis of the boom joint. During the installation of the individual components for the measurement segment it was important to find for them proper locations, so that it is possible tu use these partial measurements to determine the spatial position of the wheel axis centre for any position of the bucket-wheel excavator and its boom. If the spatial coordinates for the bucket-wheel centre are known, as well as its radius and shape, you can define the border area between the extracted and non-extracted materials [3].

With the 3D laser scanning a detailed computer model of the entire bucket-wheel excavator was created. Its evaluation was focused on the vector construction of its important structural elements, measuring the distance between the individual gauges in relation to one another, and in relation to certain mechanical ‘nodes’ in this excavator construction. These parameters are the basis for a mathematical model used for the excavator, defining its movement within the three-dimensional space. The creation of a mathematical model for the bucket-wheel excavators is further specified in the source [5].

The data are used to visualize the spatial position of the excavators and to monitor their movements in real time, using the programme Báňský model developed by KVASoftware company. This application works with a dynamic model, describing the area affected by mining for the continuous calculations of excavated materials. The calculation for the volumes and derived calculations are based on the existence of the two known 3D planes (DMT). The excavators operate using the common data space. The position of the excavators is updated every 5 seconds. The values measured by individual measuring instruments, which are transmitted from an excavator to a computer server, are stored in a database file *.DB. The 3D model of the bucket-wheel excavator and its hull is gradually cut into a digital quarry model, and thus a new surface resulting from excavation is created. The digital model of quarry is updated regularly (monthly duty cycle), based on the air digital photogrammetry (digital chamber UtraCam X) [4].

With each provided calculation of the bucket-wheel excavator position the time data of these calculations are also recorded, including the values of counted volumes and tonnage levels. Knowledge of the position for this excavator, or the position of its bucket-wheel towards the digital ground model (quarry) and its geological model, is important for extraction management and for the calculation of volumes excavated in real time (Figure 3).
3. 3D laser scanning

The laser scanning of bucket-wheel excavator SchRs 1550 in the Doly Nástup Tušimice mines took place on 11. 6. 2015, it was cloudy and temperature was 23°C; the 3D laser scanner used was Leica Scanstation C10. Before this scanning a reconnaissance of the terrain took place, a detailed photographic documentation was taken, and the bucket-wheel excavator was inspected, focused on determining all important nodal points. On the bucket-wheel excavator were signalled all the fitting points to link the model into the reference system, and to join the individual scans, i.e. registration. The position of these fitting points was selected with regard to the sufficient coverage of the individual scans and to the field of this laser scan vision. All fitting points were placed in order to cover the entire measured area evenly. There were 13 fitting points signalled in the form of HDS circular targets, and 3 of them were tilting targets equipped with magnetic holders with six inches diameters (Figure 4) [2].

Despite continuous operation for 24 hours, the excavators face regular downtimes. During these periods they are maintained and upgraded. The measurement can take place only during several downtimes planned over the year, and moreover it must take into account the current weather conditions. During the downtime period some lorries are constantly moving in the surrounding area, as well as cranes, bulldozers and other machinery. These block some parts of the excavator and cause vibration during the process of measurement. The bucket-wheel excavator needs to change the belts and other structural elements. So during these maintenance activities the excavator is in its movement mode.

For each stand it was necessary to see at least four fitting points. The fitting points were located using the polar method from the stands determined by the GNSS instrument Trimble R8 in the S-JTSK coordinate system. The horizontal and vertical resolution of the scan was set for the values from 1 cm to 2.5 cm. The boom and the bucket-wheel were scanned using the resolution of 1 cm, for the rest of the excavator the value of 2.5 cm was set for seven stands (Figure 5). A tripod was carried with the scanner to suitable places in order to obtain the required even coverage for the area scanned using the laser points. The entire field of vision was not scanned (360° x 270°), only its section in which the bucket-wheel excavator was placed. A "Quick Scan" mode was used, preserving the 270° vertical field of vision. Only the horizontal field of vision was adjusted. Taking photos using the integrated camera was not necessary. For vectorization it is sufficient to display the point cloud, which is coloured only in accordance with their reflection intensity [1].

The point cloud, which represented the structure of the bucket-wheel excavator, contained some 41 mil. points. The maximum deviation for registration did not exceed the value of 1,3 cm, the medium average error (MAE) was 0,005 m.

The next step was to filter and edit data. All unwanted objects were eliminated, i.e. the surrounding terrain, maintenance vehicles, undesired noise etc. The point cloud...
was segmented and placed into levels, or layers. These individual blocks of point clouds were always related to the surrounding areas of important nodal points for the excavator (wheel, joint of boom, the centre of the spherical truck, caterpillar chassis, GNSS receiver, etc.) (Figure 6).

Figure 6. The point cloud division into segments. The centre of wheel, the centre of spherical truck

4. Establishing key bucket-wheel excavator geometric parameters

Using the Cyclone programme an approximation of the point cloud was done for the determined structural elements on the excavator (190 lines, 26 key points). The vector design was exported into the *.dxf format, and subsequently imported into the Microstation V8 programme (Figure 7).

Figure 7. Vector model and auxiliary dimensioning elements of the structural elements on the excavator

Using the Microstation V8 programme the key geometric parameters were measured from the 3D space vector model (Figure 8). For this purpose the vector model was updated with the dimensioning elements.

The geometric parameters are measured and related to the subsequent structural elements [7]:

- GNSS receivers location,
- the bucket-wheel centre,
- the centre of the spherical truck,
- the bucket-wheel boom incline direction,
- the joint of bucket-wheel boom,
- the undercarriage bottom edge.

To define the movement of the excavator using a 3D mapping during the excavation it is necessary to know the geometric parameters (Figure 8, Table 2), which are illustrated in the bucket-wheel excavator scheme.

Figure 8. The scheme of the bucket-wheel excavator SchRs 1550

The following geometric parameters must be determined (Table 2).
Table 2 – The geometric parameters of the bucket-wheel excavator SchRs 1550

<table>
<thead>
<tr>
<th>Geometric parameters</th>
<th>Symbol</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bucket-wheel boom length</td>
<td>( l )</td>
</tr>
<tr>
<td>Vertical distance of the GPS1 sensor from the joint of bucket-wheel boom</td>
<td>( h_1 )</td>
</tr>
<tr>
<td>Vertical distance between the bucket-wheel centre and the joint of bucket-wheel boom</td>
<td>( h_2 )</td>
</tr>
<tr>
<td>Horizontal distance between the wheel axis and the GPS1</td>
<td>( L )</td>
</tr>
<tr>
<td>Horizontal distance of the GPS1 sensor from the joint of bucket-wheel boom</td>
<td>( l_1 )</td>
</tr>
<tr>
<td>Horizontal distance of the bucket-wheel axis from the joint of bucket-wheel boom</td>
<td>( l_2 )</td>
</tr>
<tr>
<td>Horizontal distance from the joint of bucket-wheel boom to the axis of excavator</td>
<td>( l_3 )</td>
</tr>
<tr>
<td>Horizontal distance of the GPS2 sensor from the joint of bucket-wheel boom</td>
<td>( l_4 )</td>
</tr>
<tr>
<td>Horizontal distance from the GPS1 sensor to the axis of excavator</td>
<td>( l_5 )</td>
</tr>
<tr>
<td>Horizontal distance between the GPS sensors</td>
<td>( h_{GPS} )</td>
</tr>
<tr>
<td>Vertical distance between the GPS sensors</td>
<td>( Z_{GPS} )</td>
</tr>
<tr>
<td>Vertical distance between the joint of bucket-wheel boom above the spherical truck</td>
<td>( Z )</td>
</tr>
<tr>
<td>Bucket-wheel boom incline in the measured position of the excavator</td>
<td>( \alpha )</td>
</tr>
<tr>
<td>Maximum diameter of the bucket-wheel up to the tooth edge</td>
<td>( D_k )</td>
</tr>
<tr>
<td>Height of the spherical truck from the undercarriage bottom edge</td>
<td>( Z_{KD} )</td>
</tr>
<tr>
<td>Height of the bucket-wheel axis from the undercarriage bottom edge</td>
<td>( Z_k )</td>
</tr>
<tr>
<td>Height of the GPS1 sensor from the undercarriage bottom edge</td>
<td>( Z_{GPS1} )</td>
</tr>
<tr>
<td>Height of the GPS2 sensor from the undercarriage bottom edge</td>
<td>( Z_{GPS2} )</td>
</tr>
</tbody>
</table>

Selected geometric parameters can be adjusted in accordance with the current position of the bucket-wheel boom. All variables and some constant parameters will change during the rotation of the upper body of the excavator, if this is its incline position. That is why it is important to take into account the influence of the excavator incline on the change of lengths reflected into the horizontal and vertical planes. The operating inclines of the travel plane for excavators range within several \%. The structural design of the upper construction does not allow for greater incline, as this could cause loss of stability. Mathematical relations for the models take into account various alternative inclines of the excavator and also the position of the bucket-wheel boom, and they are dealt with using the following sources.

Selected geometric parameters obtained using laser scanning are checked for each bucket-wheel excavator against parameters obtained via geodetic, GNSS and inclinometric recordings. Stands for this geometric measurement of these parameters were determined using the GNSS apparatus, using the same transformation key and reference stations as for the GNSS receivers, which are a part of the excavator monitoring system. When determining the length parameters for the excavator SchRs 1550 the maximum deviation between the geodetic measurement (by electronic tachymeter Leica TCR 1202) and the laser scanning reached 3 cm (parameter \( l_1 \)). Determining the incline of the bucket-wheel boom \( \alpha \) with a deviation of 0.5°. Not all parameters were tested, as some nodal points are placed inside the structure of excavator, and it is technically impossible to measure them using the traditional geodetic methods.

Here, the most useful was the similar computer model of the excavator, created and based on the laser scanning. We monitor the hull area of the complex elements, and thus we can e.g. by using the cross-sections easily identify the elements, which are placed inside the structure of this excavator. These are for example the centre of wheel and the spherical truck, from which other important geometric parameters are derived.

At the same time the database of the system allowed us to monitor the wheel in real time and determine the coordinates for the wheel centre \( X_K, Y_K \) and \( Z_K \), calculated using the measuring instruments on the excavator in real time of the check measurement (Figure 9). The comparison of the coordinates, obtained from the system for calculating the centre of wheel axis and from the geometric measurement, can be found in our Table 3. The deviations \( O_{xy} \) express the deviation of position and \( O_z \) is the deviation in height. For the sake of simplicity the coordinates obtained via checking geodetic measurement are considered to be correct.

![Figure 9. Centre of wheel (point K)](image)

Table 3 – Geometric parameters of the bucket-wheel excavator SchRs 1550

<table>
<thead>
<tr>
<th>Excavator</th>
<th>Day</th>
<th>Time</th>
<th>( O_{xy} ) [m]</th>
<th>( O_z ) [m]</th>
</tr>
</thead>
<tbody>
<tr>
<td>SchRs1550</td>
<td>21. 5. 2015</td>
<td>10:05</td>
<td>0.20</td>
<td>0.27</td>
</tr>
</tbody>
</table>
Checking measurement for the location of the bucket-wheel on the excavator, which contains a monitoring system, was done from 13. 5. 2015 to 2. 6. 2015. These bucket-wheel excavators are located in the Doly Nástup Tušimice and Bílina mines.

During the checking measurement of the bucket-wheel position in the plane X, Y, there were some deviations found ranging from 0.08m to 0.73m. The deviations in height measurement was ranging from 0.01m to 0.27m. In the Table 4 there are the numbers of bucket-wheel excavators with respect to the interval reached both for the position Oxy, and the height Oz. For the checking measurements of the position some 70% of excavators reached deviations of 0 - 40cm, for the measurements of heights it was all 100% [8].

### Table 4 – Geometric parameters of the bucket-wheel excavator SchRs 1550

<table>
<thead>
<tr>
<th>Interval [cm]</th>
<th>Number of excavators Oxy</th>
<th>Number of excavators Oz</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 - 20</td>
<td>4</td>
<td>17</td>
</tr>
<tr>
<td>20 - 40</td>
<td>10</td>
<td>3</td>
</tr>
<tr>
<td>40 - 60</td>
<td>2</td>
<td>0</td>
</tr>
<tr>
<td>&gt;66</td>
<td>4</td>
<td>0</td>
</tr>
</tbody>
</table>

### 8. Conclusions

Using the 3D technology of laser scanning we can determine the basic geometric parameters of the bucket-wheel excavators to build a mathematical model, which can define its movement within three-dimensional space. From this detailed computer model we can identify with sufficient details all the important nodal points, which are placed inside this structure. This reached accuracy must be considered in relation with the complexity of this excavator and its structure, its size, and impossibility of direct measurement of these important nodal points. No mathematical model is able to define the dynamics of movement with a sufficient accuracy, and copy the real movement of such an excavator. All geometric parameters of these excavators are being measured in their idle states. During the process of excavation the individual parts are bending and making some torsion movements. These deformations change with regard to the position of such excavator and its boom travel. The bucket-wheel diameter, measured up to the edge of tooth, is changing as the teeth of this wheel are being worn out during the excavation process. The final values are also influences by the installation of the measuring systems (declinators, GNSS antennas). The calculation also influences the overall accuracy of the system measuring devices, composed of individual measuring devices. Despite these deviations the above mentioned system, used to control and manage the process of excavation in real time, as well as to minimize the risks, proved to be very efficient. The differences between the starting volume, obtained using the photogrammetric data and by the method applying the „GNSS Báňský model”, was during the checking calculation in 2011 ranging from -0.39 to +0.91 % [4].

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### References


Failure Mode and Effect Analysis (FMEA) in Automated Mining Machinery

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Abstract — Type Failure Mode and Effect Analysis (FMEA) is a common method of failure analysis and is often the first step of a system reliability study. It involves reviewing as many components, assemblies, and subsystems as possible to identify failure modes, and their causes and effects. When FMEA is implemented in automated mining machinery, a new aspect of analysis appears, namely, FMEA application to the hardware and software of automation systems. If the critical failure mode and its related subsystem comprise mechanical or electrical parts of a machine, classic FMEA is appropriate. But in the case of automation failures (hardware and software failures), FMEA needs to be done in a slightly different way which needs a deep knowledge of failure mechanism in automation systems. Since, these kinds of failures are new concepts in mining machinery, in this paper it is tried to discuss and investigate the associated challenges in applying FMEA method in automated mining machinery. Finally, some new practical guidelines are presented for increasing the success of FMEA analysis and carrying out the effective reliability centered maintenance (RCM) program in automated or semi-automated mining fleet.

Keywords: automation, FMEA, software, hardware.

1. Introduction

During the last two decades, researchers have tried to develop new technologies to enable mining companies to maintain their assets at the highest possible safety levels, with the greatest accuracy and at the lowest costs. However, none of the proposed maintenance methods has been adequately tested or totally implemented in automated mining machinery. Maintenance challenges in automated mines recently have been noted as: safety of maintenance crew, big data and integration of automated systems [1]. Any maintenance solution should be based on a comprehensive understanding of automated machinery and systems and must be able to address one of the mentioned problems. This is achieved by building a strong database of failures and performed maintenance actions, as well as designing efficient condition monitoring systems. Two well-known maintenance solutions which may be able to build up a strong database, improve the safety of maintenance, solve big data problems and enhance system integration are: Reliability Centered Maintenance (RCM) and eMaintenance [1-3].

RCM analysis was first proposed and documented by F. Stanley Nowlan and Howard F. Heap [4]. It is a systematic process integrating Preventive Maintenance (PM), Predictive Testing and Inspection (PT&I), Repair (also called reactive maintenance), and Proactive Maintenance to increase the probability that a machine or component will function in the required manner over its design life-cycle with a minimum amount of maintenance and downtime. Specific RCM objectives, as stated by Nowlan and Heap [4], are as follows:

- To ensure realization of the inherent safety and reliability levels of the equipment;
- To restore the equipment to these inherent levels when deterioration occurs;
- To obtain the information necessary for design improvement of those items where their inherent reliability proves to be inadequate;
- To accomplish these goals at a minimum total cost, including maintenance costs, support costs, and economic consequences of operational failures.

RCM is a step-by-step process, with seven steps proposed to systematically delineate the information required to finalize the maintenance programming [4-6]:

- Step 1: System selection and information collection
- Step 2: System boundary definition
- Step 3: System description and functional block diagram
- Step 4: System functions and functional failures; preserve functions.
- Step 5: Failure Mode and Effective Analysis
(FMEA); identify failure modes that can defeat the functions.
- Step 6: Logic (decision) tree analysis; prioritize function need via failure modes.
- Step 7: Task selection; select only applicable and effective PM tasks.

Satisfactory completion of these seven steps will provide a baseline definition of the preferred preventive maintenance (PM) tasks on each system with a well-documented record of exactly how those tasks were selected and why they are considered the best selections among competing alternatives.

Typically, the following analytical tools are employed in the RCM analysis [7]:

- Failure modes and effects analysis (FMEA),
- RCM task selection flow diagram and risk-based decision making tools (e.g., risk matrix).

FMEA is one of the most important steps of RCM analysis. However, in some high-risk industries such as nuclear power plants, based on the International Atomic Energy Agency (IAEA) RCM document in 2007 [8], failure analysis (the goal of FMEA) is known as the first essential step of RCM (Figure 1).

Typically, the FMEA is thought of as a design tool used extensively to ensure the recognition and understanding of the weaknesses (i.e., failure modes) inherent to a given design in both its concept and detailed formulation [2,5,10]. Armed with such information, design and management personnel are better prepared to determine what, if anything could and should be done to avoid or mitigate the failure modes. This information also provides the basic input for a well-structured reliability model to predict and measure product reliability performance against specified targets and requirements.

FMEA are frequently extended to include other information for each failure mode as listed below [11]:

- Failure symptoms
- Failure detection and isolation steps
- Failure mechanism data (i.e., microscopic data on the failure mode and/or failure cause)
- Failure rate data on the failure mode (not always available with the required accuracy)
- Recommended corrective/mitigation actions

In conducting the FMEA, there are two basic sub-steps [9]: first is identifying the failure mode and its effects, and second is assessing the criticality of the failure mode. After passing the first step, each potential failure mode and effect is rated in each of the following three factors [5]:

- Severity: the consequence of the failure when it happens;
- Occurrence: the probability or frequency of the failure occurring;
- Detection: the probability of the failure being detected before the impact of the effect is realized.

As a common method for selecting the critical failure modes, the above three factors are combined in an index called “Risk Priority Number (RPN)” to reflect the priority of the failure modes identified. The RPN is the critical indicator for determining proper corrective action on the failure modes. The RPN is calculated by multiplying the severity (1–10), occurrence (1–10) and detection ranking (1–10) levels resulting in a scale from 1 to 1000.

The smaller the RPN the better; therefore, the larger, the worse. A Pareto analysis should be performed based on the RPNs once all the possible failure modes, effects and causes, have been determined. The high RPNs will assist in providing a justification for corrective action on each failure mode. The generation of the RPN allows the engineering team to focus their attention on solutions to priority items rather than trying to analyse all the failure modes [11-13].

When we finally turn to the question of defining the

![Figure 1. RCM analysis flowchart by IAEA (adopted from [8])]
required PM tasks, our decisions will be linked to these failure modes. In other words, no failure mode leads the RCM to no PM task. In general, the FMEA process could be briefly presented as shown in Figure 2.

![FMEA Process Flowchart](image)

**Figure 2. FMEA process flowchart (adopted from [13])**

### 3. Failure mode and effect analysis in automated mining machinery

FMEA is a common mode of reliability and failure analysis for mechanical and electrical components and has been applied successfully in non-automated mining machinery [14, 15]. However, when FMEA is implemented in the RCM analysis of automated mining machinery, a new aspect of analysis appears, namely, FMEA application to the hardware and software of automated systems. If the critical failure mode and its related subsystem comprise mechanical or electrical parts of a machine, classic FMEA is appropriate. Nevertheless, in the case of automation (hardware and software) failures, FMEA needs to be done in a slightly different way. From this point of view, the FMEA can generally be classified as either a product FMEA or a process FMEA depending upon the application. The classification and different categories of failure mode and effects analysis of the product and process FMEA are presented in Figure 3.

**3.1. General types of FMEA requirements in automated systems**

During the FMEA analysis in automated systems, the process could go through the four different types or levels of automation as following [9]:

1. General automation
2. Computer-based systems
3. Wireless data communications
4. Integrated automation systems

In integrated and complex automated systems such as mine automation, identifying the possible failure modes and the consequences of them is very demanding. FMEA is purposed to be applied on electrical, hydraulic, electronic, computer-based systems and equipment for control, monitoring, alarm and on board safety instruments. In such systems, identifying the failure modes and their consequences becomes difficult due to the vast amount of different operational states of the system and existence of program errors which is hardly possible to identify [16].

The FMEA is to demonstrate that the integrated system will ‘fail-safe’, and that essential services in operation will not be lost or degraded. A fail-safe concept is to be applied to the design of all control systems, manual emergency control systems and safety systems. In consideration of its application, due regard is to be given to the safety of individual machinery, the system of which the machinery forms a part and the whole automated fleet.

![Types of FMEA](image)

**Figure 3. The classification of failure mode and effects analysis [11]**
3.2. Hardware FMEA:
Automation hardware consists of mechanical, electrical and electronic parts. For the first two components, FMEA is applied in its classic format. However, for the electronic components whose failures occur in random fashion, the FMEA objective is trimmed to simply define the safety consequences of failure.

The failure mode and effects analysis for hardware has certain distinguishing characteristics, as noted by Ristord et al. [17]:
- May be performed at the functional or part level
- Applies to systems considered free from failed components
- Assumes failures of hardware components according to failure modes due to aging, wear or stress
- Analyses the consequences of these failures at the system level
- States the criticality and the measures taken to prevent or mitigate the consequences

3.3. Software FMEA:
To understand the analysis of software failures in automated mining systems, it is useful to consider typical software failure modes during each of the operating modes as listed in the following [9]:

1. *Lack of Functionality:* software provides no output or control action not provided when expected.
2. *Improper functionality:* the programmed control system software performs an unexpected action as defined by the operator of the equipment.
3. *Timing:* software event happens too late or too early or control action mistimed.
4. *Sequence:* software event occurs in wrong order or control action with incomplete sequence concept error.
5. *False alarm/action:* software detects an error when there is no error or control action provided when not expected.
6. *Faulty logic and ranges:* concept error where the software contains incomplete or overlapping logic or control action contains incomplete or overlapping logic.
7. *Incorrect algorithm:* software computes incorrectly based on some or all inputs or control action is based on wrong computation.
8. *Memory Management:* software runs out of memory or memory leakage or control action stops due to lack of memory.
9. *Interface failure:* software failed due to failure of hardware interfaces such as power supply, watch dog timer, clocks, reset circuits, etc.
10. *Software virus:* software did not function on demand due to software virus.

Using the above classification, the failure analysis gets a better direction and leads the maintenance engineers to deeper understanding of failures. As an essential consideration, the failure analysis should map the causes and consequences of the failures, present the controls to detect and indicate the failure, and the fail-safe action or logic that will prevent either the cause or the consequence.

For automation software, as mentioned above, the FMEA process uses a different concept of failures, ones not found in non-automated mining machinery. Therefore, this part of the RCM analysis needs to be more focused and more carefully analysed. The characteristics of software FMEA are as following [17]:
- Is only practicable at functional level
- Applies to systems considered to contain software faults which may lead to failure under triggering conditions
- Postulates failures of software components according to functional failure modes due to potential software faults
- Analyses the consequences of these failures at the system level
- States the criticality and describes the measures taken to prevent or mitigate the consequences. Measures can, for example, show that a fault leading to the failure mode will be necessarily detected by the tests performed on the component, or demonstrate there is no credible cause leading to this failure mode because of the software design and coding rules applied

When software FMEA is properly implemented at the right point in the lifecycle, it can provide the following benefits [18]:
- Make requirements, design and code reviews more effectively
- Identify single point failures caused by software
- Identify defects that cannot be addressed by redundancy or other hardware controls
- Identify abnormal behaviour that might be missing from the design specifications
- Identify unwritten assumptions
- Identify features requiring a fault handling design
- Eliminate several failures by addressing one failure mode

As the above list suggests, the findings could greatly improve the safety of both automated systems and mining machinery. This shows the capabilities of RCM analysis in improving maintenance management in automated mines and enhancing maintenance quality. However, in practice, there are two key challenges; a) finding effective ways of classifying software failures into appropriate failure modes and b) estimating the likelihood of each failure mode.

Note that resources are devoted to single failure occurrences that detrimentally impact function; we do not consider multiple failure scenarios as we do in safety
analyses. If redundancy prevents loss of function, then a failure mode thus shielded by redundancy should not be given the same priority or stature as a failure mode that can singly defeat a necessary function. If we think there is a high probability of multiple independent failures in a redundant configuration, what we have identified is a more fundamental design issue, not one that should be addressed by the maintenance program.

When do we invoke the redundancy rule? When listing the failure modes, we do not introduce it, as our objective is to ensure we initially capture each failure mode (protected or not) that can lead to functional failure. Later in the effects analysis, we apply the redundancy rule. If available redundancy essentially eliminates any effect at the system level (and, it will follow, at the plant level also), we drop the failure mode from further consideration and turn to the run-to-failure (RTF) strategy. Since automated mining machines are often designed with a host of redundancy features to achieve high levels of safety and productivity, it is not uncommon to find that this initial screening with the redundancy rule could relegate 50% or more of the failure modes to the RTF status [5]. Should we encounter this situation, our maintenance program will likely realize significant cost reductions because of foresight in the design phase (even though that foresight was not, in all likelihood, maintenance driven).

An important exception to the preceding rule involves alarms, inhibited or permissive devices, isolation devices and protection logic devices involving some voting scheme. Here, the rule requires an assumption of multiple failures to properly assess the effects or consequences of redundancy loss. In the case of alarms (as well as isolation, inhibited, and permissive devices), a "failure to operate" is, by itself, not significant. It can become significant, however, if the alarmed or protected component has also failed. We assume the alarmed component has failed so we can gain the proper perspective in our effects analysis. The same principle holds with protection logic, where redundant channels are assumed failed to the extent that the next single failure will wipe out the protection logic. We tend to find protection logic systems when dealing with safety and environmental issues or areas where machines must be "tripped" automatically to preclude widespread damage to automated machinery.

In the following, the failure of safety gates sensors as a critical part of underground mine automation systems are reviewed.

### 3.4. Example of a failure in mine automation system

New generation of mining machinery specially the automated ones are instrumented by vast number of sensors which provide a wide range of operational, maintenance, environmental and safety data. For instance, the Rio Tinto mining company does have a fleet of 900 heavy dump trucks which record 4.9 Terabit data per day. This amount data is generated by 192 different on-board sensors which are installed in different parts of machine as illustrated in Figure 4. These sensors have introduced different risk and safety associated failures which are difficult to understand their mechanism considering the harsh mining environment.

![Figure 4. Installed sensors in different part of new generation of dump trucks (adopted from [19])](image)

As another example of failures in automated mining systems, safety gates in an automated underground mine is discussed in the following (Figure 5). Automation safety system and related gates are installed separated from control system and are developed based on fail-safe PLC system. The function of these gates is to limit the load haul dump (LHD) machines’ operation within a security cell (area) [20]. The laser sensors are called failed when they don’t emit and therefore they don’t detect any motion pass through the gate. The consequence of this failure is that LHD machines leaves the operation area and automatically stops and switched off. They should be returned to operation by manual driving. Automatic stopping is control action for limiting the failure consequence of laser gates. If the control system fails simultaneously, the economical consequence of these parallel failures could be catastrophic and the machine might be destroyed completely. It reveals that automation failures can happen in a very different way and can cause economical and safety consequences simultaneously.

![Figure 5. Safety gates and related laser sensors in underground automated mines](image)
4. Conclusions
In this paper the different aspects of FMEA analysis in automated mining machinery were reviewed. This investigation reveals that generally FMEA analysis in automated systems could be carried in four major levels; general automation, computer-based systems, wireless data communications, integrated automation systems. Among these levels, the computer-based systems are the most challenging units of mining machinery view point of FMEA and RCM analysis. The automation software and sensors which provide decisions’ feed for control and computer system are the complex network of instruments and programs which play a dominant role in the quality of RCM process.

Based on the important points which highlighted in this article, it is possible to follow up the FMEA process in automated mining fleets with more concentration and focus on the failure mechanism and long term consequences of failures.

References
Technical condition change detection using Anderson-Darling statistic approach for LHD machines – engine overheating problem

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Abstract – In underground mine with room-and-pillar mining system, so called Load-Haul-Dump machines (LHD) play a key role in horizontal transportation process. LHD machines execute ore haulage from mining faces to dumping points in a cyclic way. Time-varying and harsh environmental conditions determine high workload, so effectiveness and availability demands are big challenges for maintenance staff. One of the most important issue is related to engine overheating, what is the main cause of unjustifiable loader stoppages and unwanted disturbances in production. Operator is obligated to react quickly and switch machine to idle operation until it cools down. Existing on-board monitoring systems dedicated for LHD machines provide data necessary to perform diagnostics of the engine as well as its cooling system. Understanding how load, wear level of machine and ambient temperature influence diagnostic data is the key in development of fault detection algorithms. In this paper authors propose to use long-term temperature data. The Anderson - Darling statistic is applied in order to detect segments of different statistical properties which are related to different technical condition. Analysis of obtained two-dimensional data structure allows to find points of change of technical condition of the machine. It could be considered as training for diagnostic system that could be used for machine monitoring.

Keywords: Load-Haul-Dump machines, Anderson - Darling statistic, condition monitoring, statistical analysis

1 Introduction

Load-Haul-Dump machines (LHD) are common assets used for ore transportation in production area in a mine with room-and-pillar system. Their design is theoretically adapted to full-time operation mode in difficult environmental conditions (high temperature, humidity, dust and salinity) and high time-varying load. Unfortunately, in practice harsh mining conditions significantly shorten their total lifetime specified by the manufacturer.

For the purpose of increasing operational capacity of LHDs, it is necessary to ensure adequate management information. The significant increase in the use of on-board monitoring applications in worlds mining exemplifies current demands in terms of maintenance management. Existing commercial solutions incorporate complete sensor suite installed on the machine. Typical on-board monitoring system provides online acquisition of dozens of signals (pressures, temperatures, speeds, torques, diagnostic error codes etc.) from key components of machine systems (hydraulic system, engine, gearboxes, drive chain) [1, 2]. In the literature, there exist well-known methods of signal processing to assess the LHD effectiveness as well as reliability [3, 4, 5, 6, 7, 8]. Access to operating parameters is a key for further diagnosis and optimization of Load-Haul-Dump operations.

One of the most important LHD issue is engine overheating, that affects the hauling process and the unit operations downstream, resulting in an insufficient flow of ore. When machine becomes overheated, it is good idea to let engine idle for a few minutes (if the temperature of engine coolant exceeds limit value). Such stoppages reduce operating time of machine as well as production rates of a long-term loss of dumping point and mine division, respectively, because of unwanted disturbances in production.

In case of self-propelled machines operating in underground mine condition, overheating syndrome is usually dependent on many factors, such as: (a) type of operations (loading, hauling, driving etc.), (b) ventilation solution used in production area, (c) technical condition, (d) capacity utilization of machines in long-term window, (e) mine atmosphere parameters (humidity, temperature) and corridors condition (mainly presence of mud what is a common cause of plastering the elements of cooling system.

This paper focuses on developing alternative method to differentiate time segments containing data that describe different
technical condition of the machine. Authors take advantage of the fact that statistical distribution of temperature data differs according to technical condition [9]. Appropriate transformation of long-term temperature data and application of Anderson-Darling statistic allow to quantify those differences in the form of two-dimensional map of test statistic. Analysis of this map results in detection of points where technical condition of the machine changes significantly.

1.1 LHD machine

The machine considered in this paper is a loader used in one of Polish copper ore underground mines in loading, hauling and dumping processes (see Fig. 1). Loaders are operated in 4-shift work (5-6 hours per shift), 6 days a week with the exception of short intermissions for maintenance or repair purposes. Nominal capacity of bucket is about 6 Mg. Total distance of haulage (from mining face to dumping point) does not exceed 800 meters. Investigated loader delivers material to dumping point roughly every 1.8-2.4 minutes.

Data used for evaluation of proposed method concerns temperature of engine coolant and has been acquired using built-in data acquisition system (see Fig. 2). For analysis purposes, original data with 1 second sampling have been downsampled to period of 30 seconds. Data describes period of about two and a half months.

In good technical condition case, as mentioned above, the loader operates about 6 hours per single work shift. Variability of temperature measurement is relatively high - after start up of engine, the temperature is equal to the ambient temperature, then it increases to value not exceeding 100 degrees Celsius (alarm threshold) during LHD operations and drops down to 40 degrees Celsius during idle gear, respectively. When machine is diagnosed with overheating engine problem, temperature does not vary significantly but alarm threshold is exceeded, either temporarily or permanently. These situations require quick stoppage of machine and let engine idle for a few minutes in order to cooling down. Because such overheating machine is inefficient, in practice it is used for other purposes. Its operating time is getting much shorter. Frequent idling or switching the engine off as well as long-time breakdowns of machines cause increase of variance of temperature.

2 Methodology

In this section we describe methodology of our algorithm. We propose to present it in four subsections: Pre-processing, Anderson-Darling statistic, Smoothing of AD map and AD map analysis. Fig. 3 present functional diagram of proposed procedure.

Proposed method is based on long-term data observation. Based on initial visual evaluation of the input data one can notice three disjoint processes occurring consecutively:

- **Process 1**: Machine is operating while in bad technical condition, overheating significantly;
- **Process 2**: Machine experiences a cooling system failure, but is still being operated;
- **Process 3**: Failure is repaired and machine operates in good technical condition.

In such case one can describe those processes in terms of signal parameters:

- **Process 1**: High median, low variance (warning state approaching failure);
- **Process 2**: High median, high variance (failure state);
- **Process 3**: Low median, high variance (healthy state).

Despite the relatively good visibility of the events in the input data, it is not easy to define one simple diagnostic criterion.

2.1 Pre-processing

Input data is imported in time domain, so pre-processing is required. Firstly, non-overlapping six-hour segments of the signal, that denote six-hour work shifts, are framed into new matrix as its columns. Then columns with excessive amount of missing values are rejected based on distribution of their
2.2 Anderson-Darling (AD) statistic

Because of different behaviour in observed regimes (changing the median and variance) we propose to calculate the empirical cumulative distribution functions (ECDF) of each work shift and compare statistic based on distances between them. Usually it is calculated in supremum (Kolmogorov-Smirnov [10]) or quadratic (Cramér-von Mises family [11]) norm. Usually it is calculated in supremum (Kolmogorov-Smirnov [10]) or quadratic (Cramér-von Mises family [11]) norm. For signal parameterization authors propose to use statistic based on quadratic norm between two ECDFs. Kolmogorov-Smirnov test is widely recognized and used as a classical method, but AD test is described in the literature as more powerful and hence superior [12, 13]. A class of measures of discrepancy is given by Cramér-von Mises family:

\[
Q^2 = n \int_{-\infty}^{\infty} \left( F_n(x) - F(x) \right)^2 \psi(x) dF(x)
\]  

(1)

where \( F_n(x) \) is ECDF, \( F(x) \) is theoretical CDF that \( F_n(x) \) is compared to, \( \psi(x) \) is a function of weights. \( \psi(x) = 1 \) gives \( \omega^2 \) statistic of Cramér-von Mises. To obtain Anderson-Darling statistic \( A^2 \) the vector of weights is given by \( \psi(x) = |F(x)(1 - F(x))|^{-1} [14] \).

In presented case authors used two-sample Anderson-Darling statistic which is modification of (1):

\[
A^2_{nm} = \frac{mn}{N} \int_{-\infty}^{\infty} \left( \frac{F_{m}^{n}(x) - F_{n}^{m}(x)}{H_N(x)(1-H_N(x))} \right)^2 dH_N(x),
\]

(2)

where \( F_{m}^{n}, F_{n}^{m} \) are ECDFs of \( i^{th} \) and \( j^{th} \) work shifts and \( H_N \) with \( N = n + m \) is the weight function, for \( n \) and \( m \) being the number of observations in work shifts. The \( H_N \) function is given by combining \( F_{m}^{n} \) and \( F_{n}^{m} \) distribution: \( H_N(x) = \frac{n}{N} F_{m}^{n}(x) + \frac{m}{N} F_{n}^{m}(x) \). It is used to test hypothesis \( F_{m}^{n} = F_{n}^{m} \). In presented method authors calculate Anderson-Darling statistic defined by (2) for every pair of shifts and construct 2D map of values. The resulting map is symmetric relative to its main diagonal.

2.3 Smoothing of AD map

After calculating the map of AD statistic, it requires minor smoothing to obtain more consistent levels in distinctive areas of the map [15]. Authors propose to use two-dimensional convolution-based filter with normalized triangular kernel \( K \) based on a vector \( v = [1, 2, 1] \) (3).

\[
K = \frac{v^T v}{\sum (v^T v)} \begin{bmatrix} 1 & 2 & 1 \\ 2 & 4 & 2 \\ 1 & 2 & 1 \end{bmatrix} = \begin{bmatrix} 0.0625 & 0.125 & 0.0625 \\ 0.125 & 0.25 & 0.125 \\ 0.0625 & 0.125 & 0.0625 \end{bmatrix}
\]

(3)

2.4 AD map analysis

Considering knowledge about general behavior of the processes described in previous sections, one can make a few assumptions about the expected structure of AD map (see Fig. 4):

1. ECDFs of shifts within a single process will be similar, hence values of AD statistic in the areas (1,1), (2,2) and (3,3) will be relatively low, because those are the areas where shifts of a given process are compared with shifts of the same process.

2. ECDFs of shifts from warning and failure processes are similar for failure shifts where the machine was not cooled down on purpose, and different for failure shifts where the machine was purposely cooled down (regions (1,2) and (2,1)).

3. Analogous case occurs when comparing ECDFs of shifts from failure and healthy process: they are similar for failure shifts where the machine was purposely cooled down, and dissimilar otherwise (regions (2,3) and (3,2)).

4. Comparing warning and healthy processes, ECDFs are not similar at all, hence those regions of AD map will be characterized with high values (regions (1,3) and (3,1)).
It is hard to find transition points based on two-dimensional data. Hence, we propose to determine threshold based on one-dimensional statistic. Taking advantage of previously mentioned assumptions, it is expected that variance of AD map vectors will vary according to the area. Hence, in a next step one-dimensional sample variance of smoothed map is calculated. Since the second process shares similar features with two remaining ones, which on the other hand are different from each other, one can expect certain behavior of the variance values of particular groups of AD map columns:

- **Group regarding process 1 (columns of areas (1,x))**: relatively high variance values. AD statistic values will be low when comparing process 1 to itself, medium when comparing it to process 2 (similar median) and high when comparing it to process 3 (no similar parameters);

- **Group regarding process 2 (columns of areas (2,x))**: relatively low variance values. AD statistic values will be low when comparing process 2 to itself, and medium when comparing it to processes 1 and 3 (similar median or variance);

- **Group regarding process 3 (columns of areas (3,x))**: relatively high variance values. AD statistic values will be low when comparing process 3 to itself, medium when comparing it to process 2 (similar variance) and high when comparing it to process 1 (no similar parameters);

3 Application to real data

In this section presented method is applied to introduced temperature data. In the first step temperature data is restructured into time-time domain (see Fig. 5a). Vertical axis describes local time of a single work shift, while horizontal axis stands for global time denoting consecutive work shifts. Visible gaps in data occur when machine is not operating. Empty shifts as well as shifts with excessive amount of empty values are disregarded, leaving only the shifts where machine was operating regularly, which happens to be about 40% of total considered data.

Next step interpolates missing values within remaining shifts by nearest neighbor method. The nearest neighbor algorithm selects the value of the nearest point and does not consider the values of neighboring points at all, yielding a piecewise-constant interpolant. The result of this procedure is presented in Fig. 5b. This step is required, because AD-testing procedure cannot operate on data containing missing values.

Figure 6a shows the values of Anderson-Darling statistics for each pair of compared work shifts. Areas connected to described processes are visible, but after filtration properties of the map are enhanced (see Fig. 6b).

After filtering AD map we calculate sample variance of column vectors. The values of variance for each work shifts are presented in Fig. 7b (blue line). Next step is to estimate its distribution by smoothed histogram and calculate local minimum of this function (see Fig. 7a). We propose to take this value as a threshold (red line in Fig. 7b). The area when variance is smaller that threshold correspond with failure state.
Transition points have been overlaid on input temperature data (see Fig. 8). They were identified to be at shifts no. 166 and 238 when counting within all shifts before empty values removal, or shifts no. 72 and 93 when according to interpolated map. It is important to mention that accounting system of the mine reports the repair performed on the machine, that was connected to the engine overheating problem. The actual day of repair is the day after detected change of technical condition. Considering that method presented in this paper based on statistical analysis, smoothing and transforming from two to one-dimensional form has only one day difference between date of reported repair and date of second found transition point is surprisingly small.

4 Conclusions

In this paper we discuss very important issue of LHD machine related to engine overheating. Bad condition of cooling system causes losses of machine operating time. Loader is not able to perform its tasks for a full shift without need of cooling down e.g. by idling. Detection of failure is the best practise to maximize availability of LHD machine and safety reasons. In this paper we propose the algorithm for health state change detection. The algorithm uses data transformation and Anderson-Darling statistic for comparison of empirical cumulative distribution functions in order to parameterize the temperature data. As a result, moments of change of technical condition can be identified. Approach proposed in this paper has been applied to industrial data.
Acknowledgement

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References


Abutment pressure in Longwall Mining through the damage areas

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2. Beijing University of Posts and Telecommunications, Beijing 100876, China

Abstract — Mining pressure behavior is different from ordinary panel face in longwall mining through the abandoned gateroad of small mines. It is easy to induce the accident of face spalling, roof fall and crush the shield support. In this paper, Cuijiazhai E13103 panel was used as the research background. By using mechanical analysis, numerical simulation and field measurement, the stress distribution for the panel in the disturbed area was determined with research on damage range of the vertical, oblique and parallel abandoned gateroads of small mines. For different types of abandoned gateroads, the mining technique was established for passing through the damaged area. Especially for the four parallel abandoned gateroads across the panel, the “swing-inclined face” mining method was employed in this paper. The abutment pressure distribution of “swing-inclined face” was determined as well by using numerical simulation. By analyzing the relation between "roof-shield- abandoned gateroads- pillar - intact coal", the reasonable supporting capacity of shield was calculated when the face was in the damaged areas. It is reveals the problems that the shield selection is not reasonable and it is effectively guide the shield design and shield selection; Technical measures for safety mining were also proposed in longwall mining through the abandoned gateroad of small mines.

Keywords: abandoned gateroad of small mines; abutment pressure; swing-inclined face

1. Introduction

Disorder and cross-border mining of small mines leads to the anfractuous mine gateroads left inside the mine, frequent occurrence of gobs or abandoned gateroads, roof accidents easy for mining in panel face, inducing a large danger of safety and efficiency in longwall machine mining therein [1-2]. During the 70-90’s, there were a lot of small coal mines in China. Due to their disordered mining, most of the small coal mines left a great many abandoned gateroads (defined as AG in this paper).

2. Mining Condition

Panel E13103 is located in Cuijiazhai coal mine in north-central China. Coal seam is 306-320m deep. The average mining height is 4m. The dip angle of coal seam is 5°. The panel length is 1086.5m and panel width is 110m. Panel layout is shown in Fig.1. There are four parallel abandoned gateroads across the whole panel face have a large effect on mining due to the large contacting area and even promoting fully contacting with panel face.

3. Mining technique

The inclined faceline method is employed when mining through AGs that are parallel to the faceline. The angle to the
faceline can reduce the exposure area of AG at any time. Meanwhile, according to the geological condition, the face advancing rate is increased as much as possible so that the face can pass the AG as soon as possible.

4. Abutment pressure distribution of inclined face

The rock parameters in the model is shown in table 1.

Fig. 2 Inclined face mining technique

Table 1 Mechanical parameters of panel E13103

<table>
<thead>
<tr>
<th>Serial number</th>
<th>Strata</th>
<th>Thickness/m</th>
<th>Volume force kN/m³</th>
<th>Tensile strength /Mpa</th>
<th>Elastic modulus /Gpa</th>
</tr>
</thead>
<tbody>
<tr>
<td>31</td>
<td>Loess</td>
<td>44</td>
<td>13.8</td>
<td>0.1</td>
<td>0.01</td>
</tr>
<tr>
<td>30</td>
<td>Pebble</td>
<td>10.8</td>
<td>26.2</td>
<td>0.1</td>
<td>0.02</td>
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<tr>
<td>29</td>
<td>Fine sand</td>
<td>3.6</td>
<td>25.4</td>
<td>1</td>
<td>2.7</td>
</tr>
<tr>
<td>28</td>
<td>Egg gravel</td>
<td>8.3</td>
<td>26.1</td>
<td>0.1</td>
<td>0.02</td>
</tr>
<tr>
<td>27</td>
<td>Clay</td>
<td>21.6</td>
<td>17.3</td>
<td></td>
<td>0.008</td>
</tr>
<tr>
<td>26</td>
<td>Gravel bearing Sandstone</td>
<td>13.7</td>
<td>28.2</td>
<td>6.40</td>
<td>12.2</td>
</tr>
<tr>
<td>25</td>
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<td>42.89</td>
<td>24.1</td>
<td>2.10</td>
<td>10.9</td>
</tr>
<tr>
<td>24</td>
<td>Conglomerate</td>
<td>3.18</td>
<td>24.8</td>
<td>4.30</td>
<td>28.4</td>
</tr>
<tr>
<td>23</td>
<td>Silty claystone</td>
<td>7.84</td>
<td>25.7</td>
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<td>4.6</td>
</tr>
<tr>
<td>22</td>
<td>Fine sand conglomerate</td>
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<td>4.30</td>
<td>28.6</td>
</tr>
<tr>
<td>21</td>
<td>Claystone</td>
<td>9.34</td>
<td>24.1</td>
<td>2.10</td>
<td>10.9</td>
</tr>
<tr>
<td>20</td>
<td>Silty claystone</td>
<td>21.68</td>
<td>25.7</td>
<td>3.20</td>
<td>4.6</td>
</tr>
<tr>
<td>19</td>
<td>Siltstone</td>
<td>12.47</td>
<td>25.2</td>
<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
<td>18</td>
<td>Silty claystone</td>
<td>6.94</td>
<td>25.7</td>
<td>3.20</td>
<td>4.6</td>
</tr>
<tr>
<td>17</td>
<td>Fine sandstone</td>
<td>4.5</td>
<td>27.5</td>
<td>8.64</td>
<td>35.9</td>
</tr>
<tr>
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<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
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<td>24.1</td>
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<td>10.9</td>
</tr>
<tr>
<td>14</td>
<td>6# coal seam</td>
<td>1.45</td>
<td>13</td>
<td>0.01</td>
<td>0.4</td>
</tr>
<tr>
<td>13</td>
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<td>10.01</td>
<td>25.2</td>
<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
<td>12</td>
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<tr>
<td>11</td>
<td>5# coal seam</td>
<td>5.68</td>
<td>13</td>
<td>0.01</td>
<td>0.4</td>
</tr>
<tr>
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<td>25.2</td>
<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
<td>9</td>
<td>Silty claystone</td>
<td>17.17</td>
<td>25.7</td>
<td>3.20</td>
<td>4.6</td>
</tr>
<tr>
<td>8</td>
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<td>8.62</td>
<td>25.9</td>
<td>4.30</td>
<td>28.6</td>
</tr>
<tr>
<td>7</td>
<td>Siltstone</td>
<td>12.94</td>
<td>25.2</td>
<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
<td>6</td>
<td>Fine sandstone</td>
<td>6.95</td>
<td>27.5</td>
<td>8.64</td>
<td>35.9</td>
</tr>
<tr>
<td>5</td>
<td>Claystone</td>
<td>6.85</td>
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<td>2.10</td>
<td>10.9</td>
</tr>
<tr>
<td>4</td>
<td>Fine sandstone</td>
<td>2.99</td>
<td>27.5</td>
<td>8.64</td>
<td>35.9</td>
</tr>
<tr>
<td>3</td>
<td>Siltstone</td>
<td>10.07</td>
<td>25.2</td>
<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
<td>2</td>
<td>Fine sandstone</td>
<td>1.2</td>
<td>27.5</td>
<td>8.64</td>
<td>35.9</td>
</tr>
</tbody>
</table>
Figure 3 a—d show the vertical stress distribution from a three-dimensional model when the face is passing through the AG. During mining, the headgate T-junction is moving 13m more than that of tailgate T-junction so that the faceline is inclined to the AG. It avoids the entire face moving simultaneously toward the AG such that the front abutment pressure is superimposed[3-4].

5. Comparative analysis between inclined face and straight face

The inclined faceline makes the stress concentration occur earlier at the headgate T-Junction side of the pillar. So the front abutment pressure at the headgate side is larger than the tailgate side. The pillar at the headgate side yields and fails first (Figure 4-a). The pillar is in triangle shape in the inclined face. The stress concentration occurs all the pillar when the headgate intersect the AG first. As the inclined face keeping moving, the pillar becomes smaller. The front abutment pressure is dived into two parts. One is in the intact coal outby the face and the other one is in the triangular pillar. They both are subjected to the abutment pressure until the pillar is mined out (Figure 4-b). After passing the AG, the stress concentration transfers to the intact coal outby the face as the front abutment pressure (Figure 3-c). The whole of passing is over and the faceline returns to vertical to the gateroads (Figure 4-d). For the straight face to pass through the AG, the stress concentration distribution is larger and greater than inclined face. So the inclined face method can avoid the serious roof accident of roof fall due to the extensive linkage of panel face and abandoned gateroads[3-6].

<table>
<thead>
<tr>
<th>Layer</th>
<th>Material</th>
<th>SZZ (MPa)</th>
<th>SZZ (MPa)</th>
<th>SZZ (MPa)</th>
<th>SZZ (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Silstone</td>
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<td>25.2</td>
<td>4.52</td>
<td>23.2</td>
</tr>
<tr>
<td>0</td>
<td>1# coal seam</td>
<td>4.60</td>
<td>13</td>
<td>0.01</td>
<td>0.4</td>
</tr>
<tr>
<td>-1</td>
<td>Claystone</td>
<td>1.90</td>
<td>24.1</td>
<td>2.1</td>
<td>10.9</td>
</tr>
<tr>
<td>-2</td>
<td>Oolitic clay rock</td>
<td>13.16</td>
<td>22.5</td>
<td>1.9</td>
<td>10.2</td>
</tr>
</tbody>
</table>

Fig.3 Abutment pressure distribution with inclined panel face
6. Conclusion

(1) The inclined faceline method is employed for passing through parallel AG. This method can reduce the exposure area between panel face and AG at any time.

(2) Inclined face avoids the entire face moving simultaneously toward the AG such that the front abutment pressure is superimposed.

(3) The stress concentration for inclined face is much smaller than the straight face.

Acknowledgement

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Reference


The History and Future of Rock Mass Characterisation by Drilling in Drifting
From sledgehammer to PC-tablet

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Abstract— In underground construction projects problematic rock mass conditions are one of the major issues causing cost overruns during the excavation phase. Before a tunneling project starts the rock mass is roughly characterized by a pre-investigation. However, in many cases, these pre-investigation does not portray the rock mass characteristics accurately and do not predict local anomalies in the subsurface. Therefore there is a need for new rock mass characterization methods that can reduce uncertainties and improve the overall tunneling process.

In the end of the 1880s, rock mass characterization based on manual drill data was investigated and rock masses were quantified using drillability. Since then, the technology has significantly changed with the introduction of hydraulic rock drills, computerized drill rigs, and advanced rock mass classification systems based on drill parameters. Nowadays, automatic drill logging systems and drilling data processing software packages are widely available and commonly used in Scandinavian tunneling projects.

This technology uses drilling parameters to characterize the rock mass. However, monitored drill parameters are influenced not only by the variations in the properties of the penetrated rock mass but also by the operator and the rig control system that continuously control the applied forces to optimize drilling and prevent jamming. In order to be useful for geomechanical purposes, the drilling data needs to be filtered, normalized and analyzed to refine the rock related response from responses caused by other influencing factors. If successful the data might be used to determine hardness, fracturing and water indicators.

Even though the technology has shown high potential in laboratory tests and field trials, it is not an obvious choice for all tunneling projects. In this paper, the background of the technology are described and the potential for the future outlined, concluding that the technique probably will be used more extensively in the future.

Keywords: measurement while drilling, MWD, drilling, rock mass characterization

1. Introduction

In underground construction projects, problematic rock mass properties are one of the major issues causing cost overruns during the excavation [1]. Before a tunneling project starts the rock mass is roughly characterized by a pre-investigation. This is normally based on previous experiences, for example, excavated drifts nearby, surface investigations and a limited number of diamond core (DC) drill holes. These estimated rock properties determine the excavation method, excavation advancement, material wear and rock support required. However, in many cases, the pre-investigation does not portray the rock mass characteristics accurately and do not predict local anomalies in the subsurface. The additional costs for this information shortages include extra mucking due to over-break, additional rock support, e.g. increased number of bolts and increased volume of shotcrete and tunnel lining, due to over-break as well as an irregular tunnel contour; and maybe most important, delays resulting in decreased tunneling advancement rates.

2. Measurement While Drilling

Measurement While Drilling (MWD) technology is used to log drilling parameters during drilling [2]. The drilling parameters include operational pressures, penetration rate, flush flow etc. In most Scandinavian tunneling projects MWD data is collected, but not always utilized during excavation [3, 4]. The major benefits of MWD data compared with DC are the high data density and the low data cost. Since the late 1990s, MWD recording options are available on face drill rigs produced by e.g. AMV, Atlas Copco, and Sandvik. Likewise, a similar system is available for (coal) mine roof bolters from J.P. Fletcher [5]. Nowadays, onboard analysis of drilling data can be performed on the latest generation of drill rigs [6, 7].

2.1. Drill Data Logging

The use of drilling data to characterize rock masses dates back to 1888 [8]. The collected parameters contained information on the amount of work per hammer blow and the number of hammer blows applied for removing one cubic centimeter of rock. This parameter was used for categorizing
different tunnels and rock types [8]. This method of rock mass characterization came to a hold with the introduction of air powered rock drills. In the 1970s mechanized drill rigs were developed and rudimentary machine parameters, e.g. oil pressures, penetration rates, rotary speed etc. could be recorded [9]. However, with the introduction of these new drill rigs, several issues had to be addressed to collect reliable data. Measures taken included: exact positioning of the drill rig and boom, development of a rig control system, optimization of boom interaction, development of an automated drill bit change and data logging system [9]. The first-generation computerized rock drill rigs recorded basic drilling information, e.g. the number of holes, drilled meters, penetration rate, and drill-hole sequence. Furthermore, this new control system had an automatic optimization of feed pressure and rotation speed to optimize the penetration rate. The system showed its benefits with an improved drilling rate in varying rock mass conditions, reduced bit wear and a better tunnel contour quality. All these resulted in an overall reduction of excavation cost [9]. Besides, this development resulted in improved anti-jamming systems [10]. Conjointly, these drill control systems reduced the drill-hole deviation by adjusting thrust, torque, penetration rate and percussive pressure while optimizing drilling rates. With the gradual improvement of drill control systems, the drill rig operator’s work shifted from manual drill control to drilling supervision. Apart from the drill control, drilling parameters started to be used to investigate the existence, location, and aperture of fractures in the rock mass, as well as variations in lithology and estimations of the Uniaxial Compressive Strength of the rock mass [11, 12]. In the 1990s, laboratory tests were undertaken in Sweden [13] and in the USA [14] to predict these rock mass properties. These lab tests were followed by field trials [2, 11, 15-17], during which penetration rate, drilling pressures, and rotary speed were recorded. These measurements were then plotted against hole-length and drilling time. The field tests showed [18], that in homogeneous rock masses the drilling parameters are constant; while in heterogeneous rock masses the drilling parameters show variation in thrust and torque. Based on these and similar experiences, today’s MWD system has been developed [2, 17].

2.2. Measurement While Drilling Parameters

MWD data files contain a complete recorded of the drilling operation. Firstly, it contains basic drilling information, e.g. hole ID, hole type, navigated drill rig location, hole collar location, hole depth, time-stamp as well as the drilling settings and recording settings. Secondly, the collected drilling data, i.e. operational values, are recorded at a set sample distance; ranging from minimum 2 cm and upwards. The recorded drill parameters can be separated in independent and dependent parameters [10]. The independent parameters are not influenced by the rock mass, but solely by the drill settings, operator, and control system. These independent parameters are bit thrust or percussive pressure and rotation speed. The dependent parameters are influenced by the drill systems response to varying rock conditions and are typically penetration rate, torque or rotation pressure, damper or stabilization pressure as well as flushing flow and pressure. The thrust is applied to ensure contact between the hole bottom and drill bit throughout the impact and the torque is the energy required to overcome the friction between the drill steel and hole wall and the rotation resistance between the bit and the hole bottom.

2.3. Geological Factors on Percussive Drilling

Rock mass characteristics are known to have a major influence on the drilling performance [19]. These factors are well known and likewise well described:

- Compressive rock strength has a negative correlation with drilling performance [19-21]
- Rock texture has different influences on the drilling performance [19]. The rock texture at a grain or mineral level is a considerable factor in drill fracture initiation and continuation, e.g. grain orientation to the applied stress field, grain size, grain interlocking and grain boundaries roughness. The fracture propagation requires less energy along the boundary, intersecting with grain boundaries is a barrier for fracture propagation. On the contrary, grain boundary roughness has a negative influence on fracture propagation and therefore drilling performance [19]
- Rock mass structure, e.g. joints, intrusions, and foliation, affect the drill performance [19-22]. In the case of anisotropic foliation or sedimentary layers within the rock mass [21], the drillability can differ depending on the angle between the geological structures and drilling direction, i.e. drilling at a 60° angle to the foliation requires the lowest amount of work [20]. In addition to effects on drillability of the rock mass, the foliation and sedimentary layering in the rock mass might cause drill hole deviation [20]
- Mineral composition will affect the drilling performance [19 20]; hard minerals (Mohr’s scale > 5.5) reduce the drilling rates and increase wear [19]
- Rock cavities cause major increases in penetration rate and water losses, increasing rotary friction along the drill hole and therefore wear [20].
- Porosity and permeability may influence drilling performance to a great extent [20]. High porosity and permeability might drain out the flushing water, increasing rotary friction. Further drilling in sedimentary rock masses only requires the breakage of cemented bonds between the grains, resulting in an increased penetration rate [20]
- Weathered rock masses increase the penetration rate due to their lower hardness, but with a higher risk of drill hole collapses and jamming [5, 20], especially in rock masses with a high clay content [5]

2.4. Drilling Technique Factors on Percussive Drilling

Operational factors are known to have a major influence on the drilling performance. The major factors of influences include:

- Operator skill and experience. During collaring the drilling is performed less aggressive, with reduced percussive pressure and feed pressure, in order to minimize hole
deviation and wear, as well as the proficiency to adjust angle of the drill during drilling [18, 20, 22-25]
• The choice of rig, rig capacity, and hammer type determine the optimal penetration rate in varying hard rock masses [20, 25]
• Flushing capacity used for flushing out cuttings and clay. In clay-rich areas a low flush flow may not flush the hole sufficiently, increasing rotary friction in the drill hole [5, 20, 26] Bit type, button versus cross-bit, rod and bit status as well as bit wear and button breakage, impact the drill rate significantly due to friction at the hole bottom and rock breakage capacity [5, 20, 25, 26]

2.5. Normalization of MWD Data
The rock mass, drill rig, and operator influence the recorded machine data [15, 27]. The influences from the rig control and the operator should be removed, in such manner that only features of the rock mass are taken into account [17, 28, 29]:
• In the beginning of the drill hole, when careful collar drilling is applied, i.e. reduced drilling pressures and speed, and the following ramping up the values towards regular drilling settings should be removed from the data set entirely since they do not represent normal drilling conditions [28]
• The drilling process is heavily influenced by the drill hole length. The hammer energy, rotation energy, and hole flushing become less effective at depth [29], due to increased friction and wearing of the drill bit [29] The rotary pressure increases with depth for all drill holes, due to increased friction between drilling rods and hole wall.
• In longer hole drilling, e.g. grout and probe holes, drilling data shows clear drops of feed pressure and penetration rate. This due to energy losses in the couplings of the extension rods during the drilling of these holes [29].
• One way of data normalization is summarized by [28]:
  1. Remove the data collected during collaring
  2. Eliminate the effects of hole length on the parameters, by using correlations between the drill hole depth and MWD parameters [17].
  3. Normalize the variation due to thrust differences
  4. Remove influence of penetration rate on torque

3. Analysis of Measurement While Drilling Data
Analysis of the MWD data can be performed in different ways:
• Single parameter analysis, where the independent drilling parameters as far as possible are kept constant and the dependent parameters, i.e. penetration rate, are observed and correlated to rock mass conditions [29]
• Dual parameter analysis, where the interaction between two parameters is taken into account, i.e. feed pressure and penetration rate [20], the interaction will be used to assess the rock mass quality
• Rock mass indices, produce a calculated rock mass parameter based on the drill performance, e.g. Specific Energy for Drilling [30, 31], Alternation Index [24], Rock Quality Index [24], Drillability index [32], and Drilling Energy Index [6]
• Pattern recognition uses a training image or patterns from previous data to identify rock mass characteristics by recognizing these data patterns in the drilling data [33]. The training images contain “standardized” drill behavior for when drilling e.g. fractures, different strata or cavities. The recorded data is then compared with training image
• Multivariate approach incorporates all recorded parameters in order to describe the rock mass characteristics accurately. This method includes higher order statistics and handles drilling parameters intercorrelation [28]. This combined behavior is then related to the rock mass characteristics
In order for the calibration to be accurate, extensive measurement and testing campaigns are necessary [27, 29].

3.1. Analysis of Measurement While Drilling Data
The first rock mass characterization with drilling logs has been performed by Rziha [8] in 1888. He used the destructive work [m-kg] per volume of rock drilled [cm³] to characterize the rock mass. The work per hammer swing was calculated based on the weight of the sledgehammer and drill steel and the end velocity of the swing, multiplied by the number of blows, see Equation 1. The collected data was separated into different rock categories. The data collected showed a good correlation between the rock strength and mechanical work [8]. The same concept was used for the Specific Energy for Drilling (SED) in rotary drilling [kg/m³], equation 2 [30], and refined for percussive drilling, equation 3 [31]. These concepts were further developed into drillability classification methods. Examples of such methods are the Alternation Index based on relative feed pressure and penetration rate [24], the Drillability index based on an empirical study correlating drilling performance with the Brittleness Value and the Sievers’ J-Value [32], Rock Quality Index for blastability based on the variation of drilling parameters [24] and Drilling Energy Index [N/m³] for water driven In-the-Hole hammers by multiplying the water pressure by the number of blows per unit length drilled [6].

\[ W_{ork} = \frac{\text{Hammer weight} \times \text{swing velocity} \times \left(\text{hammer weight} + \text{drill weight}\right)}{2 \times g} \]  

\[ \text{SED} = \frac{\text{Thrust}}{\text{Hole Area}} + \frac{2\pi \times \text{Rotation Speed} \times \text{Torque}}{\text{Hole Area} \times \text{Penetration Rate}} \]  

\[ \text{SED}_{perc} = \frac{4 \times \text{Transfer coefficient} \times \text{Power output drill}}{\pi \times \text{hole diameter} \times \text{Penetration Rate}} \]

3.2. Hardness Indication
MWD data analysis packages have commonly used a “Hardness” parameter. This parameter portrays the drillability of the rock mass according to the filtered and normalized penetration rate [34], where a higher “Hardness” value indicates soft or fractured rock masses and a lower “Hardness” value indicates solid competent rock masses [29]. For uncalibrated data, a relative ranking is used, but in many
projects an absolute value of the hardness is preferred; mostly the Uniaxial Compressive strength. The calibration can be performed based on Schmidt hammer test, Point Load test, Brazilian Tensile Strength test or UCS test [35, 36].

3.3. Fracture Indication
A fracturing parameter is available in the different MWD software packages. This parameter is based on the variation of the normalized penetration rate and normalized rotation pressure or torque [15, 34]. This parameter reflects, in principal, the heterogeneity of the rock mass [37]. Open, clean fractures result in an increase penetration rate, rotation speed and reduced torque, thrust and water pressure; followed by an increase of thrust etc. when the fracture has been crossed. In poor, highly fractured, rock masses the drill holes will partly cave, will result in an increased rotary friction and therefore increased torque. In the worst case, this could cause jamming of the drilling rod [29]. This might result in a lower overall penetration rate. The calibration of the “Fracture Frequency” has been performed by correlating the measured data to rock mass observations, e.g. diamond core holes, borehole TV, Ground Penetrating Radar, and geological mapping of the tunnel. These methods have shown a reasonably good correlation between the “Fracture Frequency” and RQD or observed fractures [15, 29].

3.4. Water Indication
The water indicator is available on the MWD software packages and displays the normalized water flow. The algorithm detects sudden water losses or inflows in the drill hole, by incorporating the water pressure changes. This parameter indicates both water-bearing structures in the rock mass as well as dry fractures [34, 38-40].

4. Discussion
MWD data has been available for face drill rigs since the 1990s [10] and on roof bolters since 2002 [5]. Unfortunately, the data collected on rock masses have not been used to its full potential. The main purpose for MWD data is the recording for the following up process of the tunnel quality, operator performance and saving of data for a future investigation [4]. The MWD data can be used to assess drilling quality, e.g. drill hole collaring a drilling deviation, and improving the drill rigs overall drilling performance [23, 27]. Recent improvements in computing power and the need for increased excavation quality have driven the developments of MWD. The objective and detailed drilling data have a bright future. Although not implemented, the investigation for rock mass characterization, orebody boundaries, grades, blastability and the over-break investigation has been studied, see Table 1. In several studies, the application of MWD in rock mass characterization and rock quality assessment has been used with success [2, 6, 13-15, 24, 28, 31, 35, 41] as well as rock quality assessment during roof bolting [5, 14, 16, 33]. MWD in probing is often used to alter the excavation process, i.e. pilot tunnels, New Austrian Tunnel Method or the excavation method, Drill and Blast instead of Tunnel Boring Machine. Since MWD data has shown to be a good indicator for the rock mass quality and could be incorporated during the rock support design as part of the observational method [42]. Further, MWD data is nowadays used during grouting operations in tunnel construction [3, 7, 38, 40]. The MWD data is used for the decision whether or not to increase the number of grout holes in a single grouting fan. In the near future MWD data could be used as support in blast and fragmentation design [23], with the use of the MWD data the specific charge may be altered to minimize Blasting Damage and over-break [37], as well as optimize fragmentation, depending on fracturing, water inflow and drillability or blastability of the rock mass, see Table 1. Other applications of MWD can be found in mining where the technology is used to detect orebody boundaries and extinguish between grades [17, 29]. These methods are based on mechanical differences between ore and waste as well as an established correlation between drilling parameters, or drillability and ore grade. Unfortunately, these usages of MWD data are not commonly practiced. One major concern of the use of MWD data during excavation is the transfer and process the drilling data directly, the processed information, e.g. Fracture Indication or SED should be feedback to the drilling operator; directly inside the drill rig, on e.g. a tablet PC. Simple, indicative images, such as given by Alas Copco’s Underground Manager in Figure 1, or Sandvik’s iSure, could give the drill operator enough information to support the decision-making process during excavation.

Many underground operations for mining and tunneling are equipped with a WiFi (W-LAN) network; it could be utilized for information data transfer from and to the drill rig in order to give the drilling operator the best information available. These direct-information processes have been tested in Virginia State University, USA [5] and SKB HRL Åspö, Oskarshamn, Sweden. Examples of the utilization of WiFi networks in underground environments can be seen in the mining and tunneling industry. Where the information is transferred to a central database it can be used to optimize other activities of the excavation cycle, e.g. charging, rock support etc., as portrayed in Table 1. This information can be related back to the operators i.e. on the drill and charge crews. The operators could receive this information on e.g. tablet PC and use this to improve the excavation activities, i.e. rock fracturing for charge-ability, rock support or drilling of nearby faces or drill bit wear. Concerning the technological developments today, e.g. W-LAN, “internet of things” and “Big Data”, MWD data can be used to take mining and tunneling productivity to the next level.
5. Conclusion

Rock mass characterization using drilling parameters has a long history and is driven by a need to have a more detailed characterization of existing geological uncertainty. This paper shows to what extent the drilling is influenced by geological features of the rock mass and operational parameters. With correct normalization and processing of the data, models of the rock masses based on the drilling parameters as well as drilling performance can be made and used during excavation. This will result in an improved tunnel quality at a lower cost, due to the increased knowledge of the rock mass and actions which are taken based on objective data.

6. Acknowledgements

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7. References


Table 1-Status of MWD: P: Practiced, T: Tested, F: Future

<table>
<thead>
<tr>
<th>Percussive Drilling</th>
<th>Parameter</th>
<th>Application</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quality of drilling</td>
<td>Hole location/depth Penetration rate &amp; Drilling accuracy</td>
<td>Drilling [13, 43, 44] P</td>
</tr>
<tr>
<td></td>
<td>Drilling accuracy</td>
<td>Blasting design [43, 44] P</td>
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<tr>
<td>Operator quality</td>
<td>All parameters</td>
<td>Quality of drilling [23] P</td>
</tr>
<tr>
<td>Rock mass quality</td>
<td>Fracture frequency (Penetration rate &amp; torque)</td>
<td>Grouting [3, 7, 40, 45, 46] P</td>
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<tr>
<td></td>
<td></td>
<td>Blasting – Fragmentation [43, 44, 47] T</td>
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<td>Blasting – Damage [35, 37] F</td>
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<td></td>
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<td>Excavation [13, 17, 21, 22, 35-37, 43-48] P</td>
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<tr>
<td></td>
<td>Hardness (Penetration rate &amp; torque) Weathering</td>
<td>Support T</td>
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<td>Support T</td>
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<td></td>
<td>Hardness (Penetration rate &amp; torque) Rock contact</td>
<td>Minimize dilution &amp; ore losses [13, 17, 28, 29] P</td>
</tr>
<tr>
<td>Mechanical rock alteration</td>
<td>Hardness (Penetration rate &amp; torque) Variation in grades</td>
<td>Grade control [13, 28] T</td>
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</table>

Figure 1. Atlas Copco Underground Manager penetration rate and "Fracture Frequency"


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Towards real-time, wireless in-ground simultaneous monitoring of rock deformation and pore pressure in open pit mines

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Abstract—Understanding rock mass behaviour and especially how the ground reacts to changes induced by mining is critical to making the right decision for maintaining safe and efficient operations. Identifying when, where and how the ground moves is critical to identifying possible ground failures at an early stage before the slope stability is severely impacted. The earlier ground movement is detected, the better the chance of reducing the likelihood of a failure occurring or limiting its impact.

This paper describes the latest developments of the monitoring platform that would combine the measurement of deformation and pore pressure allowing for data collection at multitude locations without considerable increase in the cost of installation as compare to the currently used measurement techniques and for coupling the pore pressure measurement with rock mass deformation data. The laboratory testing of selected components and their integration within the monitoring platform are presented and the advances in data processing and communication algorithms to enable remote data gathering and provision of information in real-time to mine site personnel are summarized.

Keywords: open pit, slope stability, deformations, pore pressure, monitoring.

1. Introduction

Rock mass geomechanical properties are the most critical sources of uncertainty in the design of open pit mines that may affect the stability of slopes with possible catastrophic outcomes: design details can have a drastic influence on the mine production and its economic viability [1]. Novel technological developments coupled with scientific interpretation of acquired data and provision of information to mine site personnel in real time are fundamental and key components to manage the uncertainty and lower the risk of slope failure.

In open pit mining, comprehensive and rigorous surface and subsurface monitoring for determining slope displacement is one of the most important means for assessing overall slope performance, providing ways towards:

- Maintaining safe operating conditions to protect personnel and equipment;
- Providing advanced notice of zones of potentially unstable ground so that mine plans can be modified to minimize the impact of slope displacement;
- Providing geotechnical information for analyzing any slope instability mechanism that develops, designing appropriate remedial action plans and conducting future slope design; and
- Assessing the performance of the implemented slope design.

Today’s surface displacement monitoring instruments are sophisticated, including automated wireline extensometers, universal EDM total stations, 3D digital photogrammetry and laser scanning, and ground-based and satellite-based radar. Together, they can provide a real-time 3D record of any surface movements that may be taking place around the walls of the pit [2]. However, in-ground displacement monitoring instruments are less sophisticated. Typically, they include shear strips and/or time domain reflectometers (TDRs), extensometers, and inclinometers placed in boreholes to locate or examine the propagation of subsurface movement after evidence of subsurface deformation has been detected at the surface [3]. Less typically, they are placed where it is anticipated that movement may not be detected by surface instruments. Rarely, if ever, they are able to detect in real-time subsurface deformation as it develops and propagates to the surface. In addition, subsurface sensors are often adapted to mining conditions from the civil engineering applications; they often do not perform well in localized shear deformations areas due to the kinking of the casing which restricts the movement of the internal sensor or even causes the breakage of the communication cables, as is the case with inclinometers and Shape Accelerometer Array (SAA) [4].

To address the drawbacks of the current in-ground monitoring instrumentation, there is a need to move to wireless sensor technology (transfer of data wirelessly between sensors through ground) with multiple sensing capabilities, which would need to provide the following advantages:
• Be more robust than traditional monitoring solutions because it does not rely on cables for data transmission from the in-ground sensor to the data acquisition system on surface;
• Combine several sensors in a single unit;
• Be easy to install multiple sensors in a single hole; and
• Be easier to install the entire system than the cabled systems.

It has been envisioned that this new development in sensing would provide better understand of the rock mass behavior prior to and during excavation and give an advanced notice of potentially unstable ground allowing mine personnel to adjust mine and production plans to avoid or minimize disruptions to the mine operation, loss of ore, equipment and/or life or injury to personnel.

In this paper, the latest developments of the monitoring platform that would combine the measurement of deformation and pore pressure allowing for data collection at multitude locations without considerable increase in the cost of installation and for coupling the pore pressure measurement with rock mass deformation data are described.

2. New developments - ENSM sensing platform

The new sensing platform (Enhanced Networked Smart Markers. Figure 1 shows the Smart Marker device [5] with the additional components which form the Networked Smart Marker, developed by Elexon Electronics [6][7].

2.1. Inclinometer sensor selection and validation

Inclinometers are devices often used to monitor subsurface horizontal deformations and displacements in vertical boreholes by measuring changes in inclination with respect to the vertical axis through a gravity sensing transducer [8][9].

To include inclination measurement capabilities within the NSM platform, a magnetometer and a three axis MEMS accelerometer combined in a single package were integrated within the internal board of the device. In this configuration, the magnetometer provides three orthogonal channels of magnetic field measurement while each of those channels provides an orientation to movement detected by the accelerometer. The measurements taken by the accelerometer in the three axes can be combined to determine the markers attitude relative to vertical and the horizontal displacements that the device may be experiencing.

A series of tests are being undertaken at the Elexon laboratories to analyze the precision and accuracy of the tilt sensors. The first test uses a 3-meter beam with a sensor attached: one end of the beam is fixed while the other end is raised to different heights (1mm, 5mm, 20mm and 22mm). For each height, accelerometer and magnetometer readings are recorded. Figure 2 shows accelerometer readings in the Y axis (readings along the X and Z axes are not shown) for two beam positions: 1st state from 0 to 20 iterations and 2nd state from 20 to 40 iterations with the height increased of 5mm. The Y values of the graph are sensitivity units, which, after data processing are converted to degrees. The angle measured between the two states is 0.0014°. These tests are ongoing and the outcomes of additional testing will be reported in future publications.

![Image 1](https://via.placeholder.com/150)

**Figure 1. Prototype of new ENSM adapted to the NSM case**

**Figure 2. Accelerometer sensitivity testing (brown line represents noise with respect to each state, which is filtered to achieve the final signal – black line)**

During the preliminary testing, it was discovered that the signal strength can be affected by the magnetic properties of the rock and by the presence of saline water. Therefore, experiments are also being design for augmented radio frequencies to improve signal transition in these conditions and to increase signal range from 5m up to 10m.

2.2. Pore pressure sensor selection and validation

Pore pressure is closely related to rock mass strength since reductions in groundwater pressure increases the effective stress in the rock mass and, consequently, increases the shear strength. In some cases, dewatering the rock mass is the only alternative available to the mine personnel to increase the strength of the rock mass and, thus, decrease the probability of slope failure [10]. Accurate characterization of the vertical distribution of pore pressure requires taking in-situ pressure measurements at different depths and is particularly important if vertical gradients are relevant in the area, as open pit mines and in operations with active depressurizing programs.

At present, the instruments used in geotechnics to measure pore pressure are vibrating wire piezometers. The piezometers are installed in boreholes and grouted with cement and bentonite grout by the fully grouted method. The standard pore pressure sensor used in mines today consists of the transducer and a protective filter.

The selection of the pore pressure sensors took into consideration the pressures that the transducer has to measure once installed in the borehole and the pressure that the transducer receives during the installation process. During the grouting process, and before the grout hardens, the sensors are exposed to head pressures, which depend on the borehole depth
and grout density; the head-pressure exerted by the grout mix is higher than the head-pressure of an equivalent water column due to the higher density of grout. Therefore, the sensor must withstand large ranges of pressures while maintaining a measurement accuracy after the installation [11].

An examination of typical mining conditions led to the compilation of parameters, which the pore pressure sensors must conform to for the integration within the Electronic Smart Marker platform:

- Able to be used in ranges of 1 to 20 bar with 0.25% of full scale accuracy
- Able to withstand an overpressure of at least 30 bar.
- Conduct measurement of absolute pressure
- Small enough to fit into either end of the existing Marker casing.
- Electronically compatible with Electronic Marker System.

The sensor selected to be integrated within the ENSM device was the OEM TTF-1 pressure transducer manufactured by the German company Wika [12] as it fulfilled all the requirements (including precision and pressure resistance) and provided the electronic connections needed to facilitate integration to the Marker.

The sensors will be subjected to grouting conditions and water pressure while in operation, thus, a filter is needed to protect the sensor from clogging up or mechanical damage while allowing the flow of water. If the filter is significantly weakened due to grouting or initial water pressure, or does not allow passage of water, the sensor may record incorrect values.

The filter selection parameters for the mining installations were linked with the pore pressure sensor requirements and included:

- Temperature resistance from -40 to +100°C
- Resistant up to 30 bars of pressure
- Sufficiently porous to allow the passage of water through it into the Marker area where the sensor is located.
- Resistant to corrosion and tearing
- Adaptable to Markers current hardware configuration
- Easily attach to any sensor (including ENSM): highly moldable.

These considerations led to the selection of the steel mesh, which has a wide variety of aperture diameters (from 1 to 1670μ), allowing for a combination of several layers with different openings for greater protection of the sensor, and is available on the market.

The initial laboratory testing has been successfully conducted for the stainless-steel mesh with a 48μm screen; the screen allowed the passage of water but not the passage of grouting, thus, protected the sensor without altering the measurements of the water pressure, as long as the time required to stabilize the medium was accomplished. The stabilization time depends on the permeability of grouting being similar to permeability of the rock and, in the laboratory conditions it took six hours [11].

In addition to screen testing, the grout mixture was investigated as one of the key factors that could affect the correct measurements of pore pressure are the permeability of the grout and the time lag until the sensor is exposed to the water pressure present in the surrounding rock. The permeability of the grout could be up to two or three orders of magnitude higher than the rock in the range of 10^-6 to 10^-8 cm/s. This range is controlled by the water:cement ratio, which does not appear to be affected by the type of bentonite used according to [14]. The time lag depends on the type of sensor, the volume of water required to record the variation of the recorded pressures, the permeability of the grout, the formation of air bubbles during installation or by the degree of saturation of the grout. High time lags are not desirable.

A "Grout Processing and Behavior Observation" test was performed to verify whether the type of bentonite influences the behaviour of the grout. Two grout mixture in proportions suggested by [13] with different bentonites, sodium and calcium were tested. Despite the viscosity difference observed at the time of their elaboration, both grouts behaved similarly during the setting and curing process (Figure 3), confirming that the type of bentonite does not influence the grout behaviour.

The "Permeability of the grouting" test was also performed. Due to the size of the markers and the method of installation in the borehole, the grout was inserted into the borehole via 12mm pipes. The small diameter of these pipes called for the modification of the proportions of the grout and the ratio used for water:cement:bentonite was 2.5:1:0.25 (modified from [13]) to enable free flow of the grout in the 12mm tubes. Permeability tests were performed using three grout samples of the same composition, but for different curing time: tests were realized for the 7, 14 and 28 days of duration at pressures of 10, 32, 98 and 163 KPa. The permeability of 10^-6 cm/s and stabilization times of one minute with grout saturation indexes of 98% were obtained. The results agree with the literature showing that the Pozzolanic cement does not influence the permeability of the grout.

![Figure 3. Sodium and calcium grout at seven days.](Image)

3. Deformation and pore pressure sensors integration into the ENSM

Both the deformation and pore pressure sensors are currently being implemented within the Networked Smart Markers casing.

With the introduction of the pressure transducer inside the device, the redesign of the casing was necessary. The need to keep the sensor in contact with water while protecting the electronics meant that the pore pressure transducer had to be exposed to the outside conditions of the Marker. For this purpose, an adaptation to the current housing has been
undertaken with a threaded hole where the transducer is to be attached (Figure 4).

The electronic components of the pore pressure and deformation sensors are installed in the sealed area of the Marker. The interior of the marker is sealed with epoxy resin to improve its tightness and resistance to blasting.

The selected mesh is attached to a plastic adapter with apertures of 1 or 2 mm in diameter that allow the passage of water, but prevents grout or other large particles (Figure 5). In addition to allowing the filter to be attached to the Marker, the plastic adapter protects it from tearing and increases sensor protection.

4. Implementation of ENSM in the field

One of the most important aspects to consider in relation to implementation of the ENSM in the field is the manner in which the sensors will be anchored within the boreholes. After careful consideration, the “Fully Grouted” method was selected, which consists of introducing the sensors into the boreholes and coupling them to the surrounding rock by the injection of a cement-bentonite grout of similar characteristics to that of the surrounding rock (Figure 6). The method is suitable for medium to low permeability media ($10^{-6}$ to $10^{-9}$ cm/s), which are the conditions typically found in mines in Chile, where the field trials are planned to be conducted. The grout mixture for the use in the installation of the Markers is the one established by Mikkelsen and Green [13] for medium to hard soils (water: cement: bentonite, 2.5:1:0.3). According to [13], the cement commonly used in this mixture is Portland cement type I. However, in Chile, the most commonly used cement is the Pozzolanic cement, which is earthquake resistant and has higher void ratio than the Portland cement.

5. Data processing and representation

Additional consideration must be given to the data processing and representation; the objective is to develop algorithms that would acquire and process the sensors data in near- or real-time and provide this information to the mine site personnel in an intelligent, intuitive and decision-based visualization manner.

5.1. Gathering Data (Remote information)

The radio signal transferred between Markers is repeated until the signal reaches a modified NSM near the surface that functions as an antenna and is connected via cable to a Reader station, where measurement data is gathered at predefined intervals. The data acquired from the sensors is downloaded from the Reader via 3G modem by an online server, which makes it possible to obtain the data remotely.

5.2. Data processing

To-date, data processing algorithms were developed to allow near real-time evaluation of the gather data. The data obtained remotely from the Reader was received by a Rest Web Service and was processed as follows:

- Rest Web service checked if the current data was sent by the Reader and not from another site;
- data was cleaned;
- data from each sensor was transformed from the raw format to an understandable expression through an algorithm developed for the application.
- both raw and processed data were saved in the relational database; and
- processed data was sent to big data's micro service and the micro service saved the data in a Nosql database.

### 5.3. Data visualization

The RSSI processed data has been represented quantitatively in a graphical form (Figure 7) for an easy and fast interpretation. In Figure 7, the horizontal axis (X) indicates the Markers’ numbers, organized according to their order during installation, with the closest to the XYZ origin being the closest to the surface. The vertical axis (Y) refers to the signal strength between two consecutive markers and is called the ‘Received Signal Strength Indicator’ (RSSI). The RSSI values range from 0 to 64. Signal strength depends on the distance between Markers and the propagation media. When installed, the initial values of RSSI are not important as the measurement focuses on the difference in values after the installation. Laboratory tests indicate that short-term fluctuations of five to ten in the RSSI values may be the result of temporary changes in the surrounding conditions. If the distance between markers is static and the conditions stable, the maximum change in RSSI values is approximately three points. The Z-axis corresponds to the average weekly values obtained during consecutive weeks. Figure 7 shows the average value of one week compared with the average values obtained during the subsequent five weeks (six weeks are represented in total).

![Figure 7. Six weeks NSM 3D data representation](image)

The variation of RSSI values of the same marker over time represents the change in distance between markers, which enables the assessment of rock movement. The change in distance between markers is due to the movement of the rock in the vicinity of the marker’s initial installation.

The next version of the data processing algorithm for the new ENSM sensors is being developed currently to process the quantitative data, which will be gathered from the inclinometer and pore pressure sensors. This new algorithm will be capable of processing the RSSI signals, the data obtained from the magnetometer and the inclinometer, the X, Y and Z axes of both, the data from the pore pressure sensor and the Marker’s temperature.

This new data requires the development of a new data processing module, which is under development and testing in the laboratory conditions. The graphical interface is also under development to allow rapid assessment of in-ground conditions.

### 6. Conclusions

In open pit mining, the ability to identify when and where subsurface deformation develops and to transmit that information to the surface in real time before the deformation propagates to and is observed at the surface, is a game-changer in slope stability monitoring. The ability to couple the technology with real time measurements of any pore pressures associated with the deformation is also a game-changer. Together they will bring a step-change to the current methods of assessing overall slope performance.

As the ENSMs can transmit data wirelessly and are designed to be blast resistant, the system has the potential to perform measurements under conditions of significant rock deformation due to large strain where cable based systems fail.

The direct impact on profitability and safety of open pit operations will be significant through advanced notice of zones of potentially unstable ground and subsequent avoidance of loss of life or injury, loss of equipment, disruption of the operation, and the loss of ore.

It is envisioned that the developed system will greatly improve risk diagnosis and decision-making based on the information provided by the sensors, allowing:

- Improved mining design and planning
- Improved production / operating cost ratio
- Improved prediction / anticipation of potentially dangerous events.

The benefits are embedded within the:

- Novel wireless monitoring platform that could potentially replace several existing technologies used for monitoring various variables in open pit.
- Improved knowledge of rock mass behavior prior to and during excavation leading to improved risk assessment and prevention of geo-hazards.
- Advanced geomechanical and geotechnical analysis of rock mass behaviour based on the real time data of sub-surface deformation and pore pressure distribution.
- Real-time data processing and communication algorithms, using wireless communication between in-ground sensors, which will transmit the data and inform the mine site personnel of the state of the deformation before the deformation propagates to and is observed at the surface.

### 7. Future works

Prior to field testing of the new sensing platform, laboratory testing of fully integrated ENSM immersed in the grout mixture will be conducted investigating the time lag in the response of the Markers before and after stabilization of the grout.
Acknowledgement

The Authors would like to acknowledge the financial support of the Chilean Government through the project Conicyt FB0809 and through the Fundación CSIRO Chile, Smart Open Pit Slope Management Project, under the auspices of CORFO; the University of Chile, Advanced Mining Technology Center, Elexon Electronics for their continuous collaboration in the development of this novel technology and Dr. John Read for the important role played at the inception of this technology and the project.

References

Review of pre-conditioning practice in mechanized
depth to ultra-deep level gold mining

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Abstract — Mining in South Africa has reached ultra-
depth levels, approaching 4000m, requiring appropriate mining
practices which stabilize ground conditions. As mining depth
increases, the stress levels ahead of the mining faces increase
with high probability of experiencing face bursting. Face-
perpendicular pre-conditioning practice was employed in order
to fracture the rock mass ahead of the face, so as to transfer
stress further away from the face. Although the face-
perpendicular pre-conditioning practice has been employed on
most deep level gold mines in South Africa, rockbursts are still
being experienced. The rockburst related accidents result in
minor and major injuries to fatal injuries as well as machinery
damage. This paper aims to highlight the effectiveness of a
revised practice of four face-perpendicular pre-conditioning to
five face-perpendicular pre-conditioning blasting technique in
a mechanized ultra-depth level gold mine in South Africa. The
investigations involved the use of underground monitoring
instrumentations for the effectiveness of preconditioning
practice, these include; micro seismic monitoring, Ground
Penetrating Radar and borehole camera. Fragmentation of the
extracted ore, fracturing on the face, side and hangingwall were
taken into consideration. Vantage numerical modelling
software was used to generate sigma 1 and safety factor within
the section and ahead of the stope faces. It was found that most
of the faces where four face-perpendicular pre-
conditioning was practiced were not effectively pre-
conditioned. The panels had high seismic activities and many large magnitude events
compared to five face-perpendicular pre-conditioned faces. Fragmentation and face fracturing were found to have
improved during the implementation of five face-perpendicular preconditioning practice. The maximum principal stress (sigma
1) and Safety Factor were found to be higher during four face-
perpendicular preconditioning practice, while these were found
to be reduced during the five face-perpendicular
preconditioning practice.

Keywords: Face-perpendicular pre-conditioning, rockbursts,
face fracturing, borehole camera, ground penetration radar
and seismicity.

1. Introduction

Pre-conditioning or de-stress blasting, as was initially
called, started at East Rand Proprietary Mines (ERPM) with the
guidance of Council for Scientific and Industrial Research
(CSIR) in the early 1950’s [11]. The argument for this was
based on the concept that, if the holes drilled at right angles into
the face were blasted, they would advance the depth of
fracturing and in so doing transfer the high stress zone further
away from the face into the unfractured rock mass. Furthermore, should sudden failure occur in the high stress
zone, only limited damage would result because of the cushion
effect of the ‘distressed’ zone ahead of the face [15]. During
the testing period, incidences of rockburst per area mined
reduced by 36%, severe rockburst events by 73% and on-shift
events dropped to almost zero. This was however, not accepted
by the mines as a viable and safe mining method and

Safety in Mines Research Advisory Committee (SIMRAC)
re-investigated pre-conditioning in the late 80’s. Long hole,
face parallel pre-conditioning tests started at West Driefontein
and at Blyvooruitzicht GM in 1990. This was not successful
from a production viewpoint as it caused production delays.
Tests on face-perpendicular pre-conditioning in a Longwall
started at Mponeng (then Western Deep Levels South) in 1994.
Although successful, it was not pursued after the testing was
completed. Various attempts were made to do preconditioning
mine-wide but it was only in the beginning of the 2000’s that it
was rolled out and enforced mine-wide [8 and 15].

Pre-conditioning transfers the stresses away from the stope
face through remobilization of existing fractures in the rockmass by shearing through asperities that are responsible for
the ‘lock-ups’ on the fractures. This prevents the accumulation
of strain energy ahead of the working face through stress
redistribution, thus reducing the potential for faceburst damage
[8 and 15]. The stress redistribution away from the working
face from a well-executed pre-conditioning blast provides a
low stress / cushion ahead of the working face, which is able to
absorb energy from distant seismic events [15]

Two different preconditioning techniques have been
developed, namely four face-perpendicular preconditioning
and five face-perpendicular preconditioning. Both have prevented face bursting in areas to which they have been
applied, even though several large seismic events have
occurred close to the faces in some areas [15].

Pre-conditioning involves regularly setting off carefully
designed blasts in the fractured rock ahead of a mining face.
This is achieved by drilling holes into the face that are longer than the normal production holes, this is shown in Figure 3.

These holes are charged and blasted with the production holes. It is essential that the pre-conditioning holes are timed to go off prior to the production holes. The gas and shock generated by a blast within the fracture zone can be remobilized in the rock mass by shearing through rigidities that cause any ‘lock-ups’ on the fractures. In the process, the strain energy release is facilitated by stable sliding of rock mass past one another, which then reduces the risk of face bursting during production shift [15].

This study investigates the effectiveness of a revised five face-perpendicular pre-conditioning technique compared to the previous four face-perpendicular pre-conditioning technique.

2. Field Investigation
The investigation was carried out in six sections where de-stress mining is practiced. Underground observations (visual examination) and recordings (measurement of preconditioning hole diameter after and before blasting, face fracturing before and after blasting, number of preconditioning holes drilled) were conducted within the sections. Visual examination included; taking photographs of the faces, sidewall and hangingwall, locating the position of preconditioning holes after blasting, measuring the depth of the preconditioning holes. Observations on the fracturing of the stope face, sidewall, hangingwall, general ground conditions and fragmentation were done daily and monthly reports were generated. OSTRCH software was used to monitor micro seismicity of the mine. Ground Penetrating Radar (GPR) and Borehole camera were used for the identification of fracture frequency ahead the face. Vantage software was used for numerical modelling. Sigma 1 and Safety Factor were obtained from simulation using the Vintage software.

The material properties used in the models are shown in Table 1. The visual observation data, recordings data and instrumentation data (borehole camera data, GPR data and micro seismic data) were collected from March 2014 to December 2016. From the collected data, the following were determined: fracture frequency ahead of the face (using borehole camera and GPR), sidewalls and hangingwall conditions (using visual observations), the comparison of micro seismic events (using OSTRCH software) and sigma 1 and safety factors ahead of the face and along the regional pillars (using Vantage software).
Table 1: Material properties of the deep level gold mine

<table>
<thead>
<tr>
<th>Number</th>
<th>Domain</th>
<th>Code</th>
<th>Density (g/m³)</th>
<th>UCS (MPa)</th>
<th>GSI</th>
<th>Stage</th>
<th>Plastic Strain</th>
<th>E (GPa)</th>
<th>ν</th>
<th>m</th>
<th>d</th>
<th>ε</th>
<th>Φ</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Pretoria Group</td>
<td>PG</td>
<td>2.752</td>
<td>100</td>
<td>65</td>
<td>Peak</td>
<td>0.0%</td>
<td>20.9</td>
<td>0.25</td>
<td>25.6-E-03</td>
<td>1.77</td>
<td>0.5</td>
<td>0.25</td>
</tr>
<tr>
<td>2</td>
<td>Malman Dolomite</td>
<td>DOL</td>
<td>2.752</td>
<td>100</td>
<td>65</td>
<td>Peak</td>
<td>0.0%</td>
<td>20.9</td>
<td>0.25</td>
<td>25.6-E-03</td>
<td>1.77</td>
<td>0.5</td>
<td>0.25</td>
</tr>
<tr>
<td>3</td>
<td>Ventersdorp Lava</td>
<td>VENT</td>
<td>2.752</td>
<td>100</td>
<td>65</td>
<td>Peak</td>
<td>0.0%</td>
<td>20.9</td>
<td>0.25</td>
<td>25.6-E-03</td>
<td>1.77</td>
<td>0.5</td>
<td>0.25</td>
</tr>
<tr>
<td>4</td>
<td>Westenaria Formation</td>
<td>WAF</td>
<td>2,900</td>
<td>79</td>
<td>54</td>
<td>Peak</td>
<td>1.0%</td>
<td>18.5</td>
<td>0.25</td>
<td>1.80-E-01</td>
<td>1.19</td>
<td>0.5</td>
<td>0.20</td>
</tr>
<tr>
<td>5</td>
<td>Ventersdorp Contact Reef</td>
<td>VCR</td>
<td>2,680</td>
<td>56</td>
<td>57</td>
<td>Peak</td>
<td>0.0%</td>
<td>11.7</td>
<td>0.23</td>
<td>7.89-E-04</td>
<td>0.92</td>
<td>0.5</td>
<td>0.07</td>
</tr>
<tr>
<td>6</td>
<td>ELSburg Reef package</td>
<td>EREEF</td>
<td>2,780</td>
<td>200</td>
<td>75</td>
<td>Peak</td>
<td>0.0%</td>
<td>50.0</td>
<td>0.20</td>
<td>5.70-E-02</td>
<td>3.00</td>
<td>0.5</td>
<td>0.50</td>
</tr>
<tr>
<td>7</td>
<td>Footwall quartzite</td>
<td>FWQ</td>
<td>2,780</td>
<td>157</td>
<td>65</td>
<td>Peak</td>
<td>0.0%</td>
<td>32.8</td>
<td>0.25</td>
<td>1.29-E-02</td>
<td>3.12</td>
<td>0.5</td>
<td>0.39</td>
</tr>
<tr>
<td>8</td>
<td>Footwall shale</td>
<td>SHALE</td>
<td>2,700</td>
<td>157</td>
<td>65</td>
<td>Peak</td>
<td>0.0%</td>
<td>32.8</td>
<td>0.25</td>
<td>1.29-E-02</td>
<td>3.12</td>
<td>0.5</td>
<td>0.39</td>
</tr>
<tr>
<td>9</td>
<td>Dolomite dyke</td>
<td>DIYE</td>
<td>2,800</td>
<td>250</td>
<td>65</td>
<td>Peak</td>
<td>0.0%</td>
<td>45.0</td>
<td>0.30</td>
<td>1.59-E-01</td>
<td>7.39</td>
<td>0.5</td>
<td>0.63</td>
</tr>
<tr>
<td>10</td>
<td>Fault type 1</td>
<td>F1</td>
<td>2,700</td>
<td>50</td>
<td>60</td>
<td>Peak</td>
<td>0.0%</td>
<td>10.5</td>
<td>0.25</td>
<td>4.41-E-04</td>
<td>0.66</td>
<td>0.5</td>
<td>0.13</td>
</tr>
<tr>
<td>11</td>
<td>Fault type 2</td>
<td>F2</td>
<td>2,700</td>
<td>60</td>
<td>50</td>
<td>Peak</td>
<td>0.0%</td>
<td>12.5</td>
<td>0.25</td>
<td>7.22-E-04</td>
<td>0.91</td>
<td>0.5</td>
<td>0.15</td>
</tr>
</tbody>
</table>

The two methods were designed differently and in different patterns. Four face-perpendicular pre-conditioning practice involves the use of four drilled face-perpendicular pre-conditioning holes, drilled with the drill bit of 51mm diameter and the holes were 1.5m longer than the production hole (see Figure 1a and Figure 1b). The last 1.0m of each of the four holes is charged up with emulsion and 30cm gassing gap is created, the rest of the hole is tamped by appropriate methods and equipment.

Detonation of the production and four face-perpendicular pre-conditioning holes were sequenced with 1 millisecond delays in the following manner;
First – detonate face-perpendicular preconditioning holes below the grade line
Second – detonate face-perpendicular preconditioning holes above the grade line
Third – detonate the cut and then the rest of the production holes
Alternate the positions of the face-perpendicular pre-conditioning holes after each blast.

Figure 1a. Four face-perpendicular preconditioning hole pattern for the first blast (shift 1)
Five face-perpendicular preconditioning holes were designed in the following manner (see Figure 2):

- All five face-perpendicular pre-conditioning holes were drilled to a minimum of 1.5m more than the production drill lengths. Where the production round lengths were effectively 2.5m, each face-perpendicular pre-conditioning hole length was at least 4.0m.
- 51mm diameter drill bits were used for face-perpendicular preconditioning holes and 41mm diameter drill bits were used for production holes.
- The last 1.0m of each of the five face-perpendicular preconditioning holes was charged with emulsion and 30cm gassing gaps were left, the rest of the hole was tamped by appropriate methods and equipment.
- Detonation of the production and face-perpendicular preconditioning holes were sequenced with 1 millisecond delays in the following manner (see Figure 3);
  
  First – detonate face-perpendicular preconditioning below the grade line
  Second – detonate face-perpendicular preconditioning above the grade line
  Third – detonate the cut and then the rest of the production holes

- Alternate the position of the face-perpendicular pre-conditioning holes after each blast.

Figure 1b. Four face-perpendicular preconditioning hole pattern for the second blast (shift 2)

Figure 2. Five face-perpendicular preconditioning practice pattern.
3. Results and Discussion

3.1. Visual examination results

The investigation of the blast holes (sockets) showed that four face-perpendicular pre-conditioning produced sub-standard preconditioning compared to five face parallel pre-conditioning (Figures 4a and 4b). It can be observed from Figures 6a, and 6b that the five face pre-conditioning holes produced fragmented zone around the preconditioning sockets whereas, for four face-perpendicular sockets, it was only after the removal of the slabs that fragmented zone around the pre-conditioning socket could be observed. The conditions of the mining faces, the hangingwall, the sidewall and fragmentation also improved (Figures 5a, 5b and 5c) from four face-perpendicular preconditioning holes to five face-perpendicular preconditioning holes.
3.2. Borehole camera

Borehole camera is an effective and rapid way to observe dynamic morphology of boreholes and temporal and spatial distribution of mining induced fractures around the borehole [16]. A borehole camera was inserted in each of the pre-conditioned holes and the images taken were used to evaluate the fracturing extend in the face using a fracture frequency standard designed by the mine (Table 2). The table was designed to assess the number of fractures per meter of the blast hole length.

Table 2. Standard for fracture frequency analysis design. The method of assessment being used is fracture frequency per metre (ff/m) as a measure of potential strainburst risk. The different categories used are outlined below:

<table>
<thead>
<tr>
<th>Fractures/m</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 5</td>
<td>the rockmass has not fractured or yielded and it is considered to have a high strainburst risk</td>
</tr>
<tr>
<td>5 - 10</td>
<td>the rockmass is beginning to fracture and has begun to yield and is considered to have a medium strainburst risk</td>
</tr>
<tr>
<td>10 - 20</td>
<td>the rockmass is fractured and has yielded and is considered to have a low strainburst risk</td>
</tr>
<tr>
<td>&gt; 20</td>
<td>the rockmass is highly fractured and fully yielded and is considered to have a very low strainburst risk</td>
</tr>
</tbody>
</table>
The four face-perpendicular pre-conditioning analysis shows that the faces were only fractured for the first 1m beyond the face and the ground was solid up to the end of the hole. Previous work has shown that a face with such short depth of fracturing is prone to face bursting. In this study, although some of the faces did not experience face bursting, majority of the faces were prone to face bursting (Figure 6). After revising the standard to five face-perpendicular preconditioning, most of the holes were noted to have extensive fracturing between 0-3m and the fracturing decreased towards the end of the blast hole (Figure 7).

Figure 6. Borehole fracture frequency with less fracturing ahead of the face from four face-perpendicular preconditioning

Figure 7. Borehole fracturing with intensive fracturing from five face-perpendicular pre-conditioning.

The borehole fracture frequency data was used to determine the face burst risk per meter for each blast hole using the fracture frequency along the hole (FF/m) on 71 holes from five face-perpendicular blasting (Figures 8 and 9). It was found that 48.6% of the 71 five face-perpendicular pre-conditioned holes were of relatively low strain burst risk and only 5.4% of the 71 holes were of high strain burst risk, at the depth of between 3.5m to 4m (see Figure 9). However, 46% of the 71 holes were found to be of low strain burst risk between 2m to 3m depth of the holes. This indicates that the faces are less prone to face burst due to highly fractured ground ahead of the face (3.5m to 4 m) (Figures 8 and 9).
3.3. Ground penetration radar surveys

Ground Penetration Radar (GPR) is considered to be a rapid, non-destructive, high resolution electromagnetic reflection geophysical technique that can provide information about rock fracturing and geological discontinuities around underground excavations to distances of up to 30m [5]. The high frequency
electromagnetic pluses of GPR are transmitted into the rock, and the reflections are detected, via an antenna. An integrated computer records the strength and time required for the return of reflected signals. Any subsurface variations, metallic or non-metallic, will cause signals to bounce back. When this occurs, all detected items are revealed on the computer screen in real-time as the GPR equipment moves along. In data processing, detailed examination/interpretation of GPR sections may be able to identify soils, bedrock, fractures, groundwater, etc. The depth range of GPR is limited by the electrical conductivity of the ground, the transmitted center frequency and the radiated power. Although the instrumentation provides useful results, Gardner [5] studies highlighted some difficulties that are experienced with GPR. The difficulties includes low reflectivity of the structure in rock mass due to the presence of metallic objects (pipes, rockbolts, mesh and lacing).

Ground penetration radar was used to analyse the effectiveness of the two face-perpendicular pre-conditioning practices within different mined stopes by studying the fractures. Analysis was conducted after each blast. The electromagnetic pulses emitted by the GPR antenna are reflected strongly by fracture planes, particularly if the fractures are open and the sides are coated with residues from the blast gases [5; 15]. From the GPR images, it was possible to study the fracture depth and intensity of fracturing ahead of the face to define the zone of influence of individual preconditioning holes. The fracture patterns from the four face-perpendicular pre-conditioning blasting were compared to the five face-perpendicular pre-conditioning blasting.

GPR images from four face-perpendicular pre-conditioning produced weak reflections within 1m to 3m ahead the face, which indicate that there were less changes in material properties and less presence of discontinuities. Underlying that, the rock mass ahead of the face is not properly fractured and is therefore prone to face bursting (Figures 10). “Note that the bright red colour on the GPR image indicates stronger reflections within the material under test, the reflections are caused by discontinuities in the material and change in material properties.” Figure 11 shows intense bright red colour on the GPR image at greater depths (up to 4m) than Figure 10. The five face-perpendicular pre-conditioning practice produced a significant difference in the nature of fracturing ahead of pre-conditioned faces compared to four face-perpendicular pre-conditioning practice. This is evident from the GPR images (Figure 10 and Figure 11). In the four face-perpendicular pre-conditioning, the fracture ahead of the face extended to less than 3m in depth while the fracturing in the five face-perpendicular pre-conditioning blasting extended up to 4m. The density of open fractures was also much higher in the five face-perpendicular pre-conditioning. This can be observed from the intense red colour in the five face-perpendicular pre-conditioning blasting image.

Figure 10. Less deeper fracturing from four face-perpendicular pre-condition blasting (circled with red paint)
In four face-perpendicular pre-conditioning, the fracture ahead of the face extended to less than 3m in depth while the fracturing in five face-perpendicular pre-conditioning blasting extended up to 4m. The density of open fractures was also much higher in five face-perpendicular pre-conditioning as can be observed from the intense red colour in the five face-perpendicular pre-conditioning blasting image, Figure 11.

3.4. Seismicity in de-stress cuts in deep level gold mining (micro seismic monitoring)

According to Ryder and Jager [12], seismic monitoring in mines enables the quantification of the exposure to seismicity, and provides a tool to guide in the effort for prevention, control or prediction of stress accumulation that could result in a rock burst. Seismic monitoring can provide information for:

- Evaluating seismically hazardous situations related to mining (mining sequences, geometry, pillar design etc.). This assists in improving safety and in optimizing mine designs.
- Back analysis of large events (typically M ≥ 2.5) to assess the cause of the instability
- Evaluating the seismic hazard associated with geological structures

In this study, a comparison of seismic events between 2014 and early 2015 (when four face-perpendicular preconditioning was in practice) and mid-2015 to early 2017 (the current events when five face-perpendicular preconditioning is in practice) was conducted using ostrich and ticker seismic monitoring system of the mine within the vicinity of the panels under study.

Seismicity detection and analysis in mines has the potential to provide vital information for the planning and control of mining operations, including identification and quantification of a particular hazardous structure as well as analysis of mining situations [1, 4 and 8]. Ryder and Jager [14] pointed out that seismic events locations can be used to locate the potential locations of rockburst associated with the intermediate and large seismic events locations. This can have some bearing in analysing fundamental mechanisms of rockbursts in deep level gold mines.

The seismicity, as expected, was clustered around the mining panels. It was noted that high seismic activities with many large events occurred when four face-perpendicular pre-conditioning was in practice compared to five face perpendicular pre-conditioning. Most of the large magnitude events occurred along geological structures and within destress cuts (Figures 12 and 13). The difference in colour represent time in which the event occurred, so different colours were used to differentiate the time at which the event occurred and the size of the circle represent the magnitude of the event.
The slope of the linear portion of the frequency magnitude graph yields the b-value, which gives an indication of the relative proportion of larger and smaller seismic events (e.g., a lower b-value indicates that more seismic energy is released via a relatively large number of larger events) [1, 4, 8, 14 and 15]. The Gutenberg–Richter magnitude distribution were used to compare the number of seismic events which occurred during the use of four face-perpendicular pre-conditioning and five face-perpendicular preconditioning blasting. A b-value of 0.5 was obtained, a fairly typical b-value of deep level gold mines (see Figures 14 and 15). It is also noted that during implementation of four face-perpendicular pre-conditioning practice, many large events were experienced with magnitude 3.0 as the maximum event (see Figure 14). The number of large events dropped on the introduction of five face-perpendicular preconditioning, with magnitude 2.1 being the maximum size event (see Figure 15). It is also noted that there was a
decrease in the number of events on implementation of five face-perpendicular pre-conditioning practice. This indicates that the five face-perpendicular pre-conditioning blasting was effectively destressing the rock mass ahead of the panel face as compared to four face-perpendicular pre-conditioning blasting.

Figure 14. Gutenberg–Richter number of events magnitude distribution during the four face-perpendicular preconditioning practice

Figure 15. Gutenberg–Richter number of events magnitude distribution during the five face-perpendicular preconditioning practice
3.5. Incidents associated with rockbursting at deep level gold mining

Rockburst incidents were high during the practice of four face perpendicular pre-conditioning holes than five face parallel pre-conditioning, the number of rockbursts incidents decreased rapidly from 2014 to 2017, (see Figure 16). The rockburst incidents percentage dropped from 38 injuries to 8 injuries from year 2014 to mid-2016, that is a drop of about 79% (when five face-perpendicular pre-conditioning holes replaced the four face perpendicular blasting). This indicates that the five face-perpendicular pre-conditioning blasting was effectively destressing the rock mass ahead of the panel face and thus more effective in combating rockbursts compared to the four face-perpendicular pre-conditioning practice.

![Figure 16. Rockburst incidents](image)

3.6. Numerical modelling

In conventional mining, elasticity based numerical models are commonly applied in an attempt to quantify the expected stresses in a given mining layout. This is generally used for rapid turnaround for mine design and support requirements determination [6, 9, 10 and 14]. In this study, numerical modelling analysis was carried out to understand magnitudes of stress (sigma 1) and Safety Factors ahead of the face and at the regional pillar using Vantage software, a 3-D numerical modelling software. Sigma 1 and Safety Factor were determined from the four face-perpendicular preconditioning practice. It was found that the sigma 1 (σ₁) ahead of the face was between 132 MPa and 270 MPa and between 180 MPa to 270 MPa for the regional pillar (see Figure 17). Furthermore, Safety Factor analysis was done and it was noticed that the Safety factor ahead of the face was between 2.0 and 1.0 and 1.8 and 1.0 along the regional pillar (see Figure 18).

![Figure 17. Modelled induced stresses (σ₁) ahead of destressed faces and at the regional pillar](image)
The stress levels from numerical modelling results show that the stresses ahead of the face are about 270 MPa and the Safety Factor was found to be about 1.0. This is higher than the unconfined compressive strength (UCS) of Witwatersrand quartzites. UCS values of typical Witwatersrand quartzites are in the order of 180 MPa [11]. This type of stress is likely to generate hazardous rockbursts on daily basis, if the use of face-perpendicular pre-conditioning is not practiced.

Sigma 1 ($\sigma_1$) and Safety Factor were also generated for five face-perpendicular preconditioning practice. It was found that sigma 1 ($\sigma_1$) ahead the face was between 60 MPa to 180 MPa and 182 MPa to 252 MPa at the regional pillar (see Figure 19). Safety Factor values were determined and it was found that the Safety factor ahead of the face was between 1.50 and 0.75 and between 1.35 to 0.75 at the regional pillar (see Figure 20).

The stress was found to be lower than the unconfined compressive strength (UCS) of the mining rock for the practice of five face-perpendicular preconditioning holes. The deeper fracturing from the five face-perpendicular pre-conditioning holes were found to transfer stresses away from the production face and thus lower the stresses ahead of the face and regional pillar.

Although the five face-perpendicular pre-conditioning practice is time consuming, it has several benefits. It produces deep fracturing away from the face. This transfers stresses away from the production areas and thus reduce face bursting risk. This in turn improves the hangingwall and sidewall fracturing (less shallow dipping fractures), it also reduces fracturing over hangingwalls and improve face conditions (i.e. less sockets and face shape). The extended fracturing ahead of the face induced by five face-perpendicular pre-conditioning reduced the seismicity, rockburst incidences, accidents and injuries resulting from rockburst and falls of ground. Over the past nine months, since the inception of the five face-perpendicular practice, rockburst accidences have reduced, with injuries dropping from 38 injuries to 8 injuries, that is about 79% drop in rockburst related accidents.

4. Conclusions
Effective preconditioning goes a long way in stabilising ground conditions in ultra-deep mining operations prone to seismicity and rock bursts. This study has investigated two different preconditioning techniques (four face-perpendicular pre-conditioning and five face-perpendicular pre-conditioning) using practical observations from an ultradeep gold mine over three years. It was found that most of the faces where four face-perpendicular pre-conditioning was practiced were not effectively pre-conditioned. The panels had high seismic activities and more large magnitude events compared to five face-perpendicular pre-conditioned faces. The images from GPR and borehole camera showed intense fracturing of up to 4m from the face for five face-perpendicular pre-conditioning compared to less than 3m for four face-perpendicular pre-conditioning. The blast sockets were well fractured in the five face-perpendicular preconditioning case. Most of the injuries associated with rockbursts were within ineffectively pre-
conditioned faces (four face-perpendicular pre-conditioning), a few were within effective pre-conditioned faces (five face-perpendicular pre-conditioning). The five face-perpendicular pre-conditioned faces have low face burst risk compared to four face-perpendicular pre-conditioned faces. The extended fracturing ahead of the face induced by five face-perpendicular pre-conditioning reduced seismicity, rockburst incidences, accidents, injuries associated with rockburst and falls of ground. Over the past nine months, since the inception of the five face-perpendicular practice, rockburst accidents have reduced with the number of injuries dropping from 38 to 8, which is about 79% drop in rockburst related accidents. The practice of five face-perpendicular preconditioning is considered to be the most effective method to be employed in deep to ultra-deep gold mining.

References


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Development of Chain Conveyor Cutter Method and Its Application

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Abstract—Currently, ocean disposal sites can provide more space for landfill than other areas such as waste landfill in the mountains and in the coastal areas. The construction of disposal sites in the ocean, therefore, has received attention in Japan, which has limited space for landfill. In the case of ocean disposal sites, the leakage of pollutants from waste materials has to be prevented by constructing a vertical impermeable wall around the disposal site.

In the field of ground improvement works, mechanical soil mixing is an indispensable technique and is carried out using an auger type machine with mixing paddles attached to the drilling rods. This type of machine excavates the soil in situ horizontally and cannot mix the cut soil vertically. In this system, the strength of the constructed soil cement column is governed by the weakest strata and the impermeability of the constructed soil cement wall is not necessarily uniform. From this point of view, the Chain Conveyor Cutter (CCC) method has been developed as the technology for soft-ground stabilization in Japan. The ground is excavated with the construction of an improved wall by injecting a fixation agent in the method. It is expected that this method will be utilized for the construction of an impermeable wall in ocean disposal sites due to low cost and the simplicity of the construction process.

This paper presents a comparison of the performance of a vertical impermeable wall constructed using the CCC method and another using the conventional method through the simulation of the spread of contaminants from the waste storage area with two-dimensional advection-dispersion program, Dtransu-2D, with the aim of the application of the CCC method to the construction of ocean disposal sites. The results indicated that the CCC method can be applied to the construction of ocean disposal sites, enabling the construction of an impermeable wall in a simple way and at lower cost when compared with walls that were constructed using the conventional method. Additionally, the distance between the impermeable wall and the waste storage area in ocean disposal sites should be larger in order to minimize the effect of water flow near the impermeable wall on the spread of pollutants from the waste storage area.

Keywords: Chain Conveyor Cutter (CCC), Ocean Disposal Sites, Advection-dispersion, Dtransu-2D.

1. Introduction

Mitsui Miike Coal Mine was the largest mine in Japan and closed in March 1997. The mine extracted coal in the underground beneath the Ariake inland sea for more than 100 years and developed advanced technology in the field of coal winning, tunneling and so on. As shown in Figure 1, Mitsui Miike High Power Plant (MHP) was introduced in 1988 for realizing the average daily output of 6,000 tons by allowing a peak capacity of 10,000 tons of coal per day and contributed until the closure of the mine, maintaining the safety of the mine.

![Figure 1. Underground machinery MHP](image)

Based on the accumulated technology of coal mining in Miike, the authors have collaborated and developed new types of machines for improving soils in situ named Chain Conveyor Cutter (CCC) and the CCC method has enabled us to develop new types of effective working methods. The CCC was introduced into public works this year for constructing impermeable walls surrounding large settling ponds for preventing the water from flowing out through the bottom of
the ponds and successfully finished the works. This paper describes the development works of the CCC and the working method utilizing the CCC.

2. Characteristics of Underground Mining Machineries

Notable differences exist in the working conditions between the machineries in underground use and the earth moving machineries used on surface.

The spaces available for the underground machineries are limited, and the combustible gas emitted from the coal seams require extra safety considerations. The characteristics of underground machineries and the know-how employed for designing and manufacturing the CCC are summarized as follows:

1) The advanced mining machineries are systemized and require the harmony of each element.
2) In spite of the limited space, the machineries must be robust, heavy duty, powerful and flexible enough to follow the changeable mining conditions such as thickness, dip and hardness of coal, geological fault etc.
3) The power required to operate the electric motors is sometimes more than 200% of nominal output for a short duration of time like starting the conveyor of the working face due to fallen coal or rocks. The torque distribution curve of the machine is very characteristic and is useful for designing the CCC.
4) Methane gas is combustible and requires the electrical equipment be flameproof and is controlled by sensors.
5) In addition to the methane gas emission, the water sprayed for dust suppression prohibits the electrical and/or mechanical chambers to be opened to the atmosphere due to the loosening of packing bolts caused by the severe vibrations accompanied by the operation itself.
6) The sealing techniques are indispensable for the machineries that the techniques utilized are adopted.

3. Application of CCC method to Construction of Ocean Disposal Sites

An ocean disposal site is considered one of the best ways to treat a large volume of waste. It has received attention due to a rapid increase in the amount of waste produced in Japan. The performance of an impermeable wall to prevent the leakage of contaminants from the waste storage area is the most important parameter in ocean disposal sites. The vertical impermeable wall is usually constructed by the soil improvement; however, it is difficult to construct the wall using conventional methods under the condition that the ground is hard. In order to construct the impermeable wall via soil improvement, the CCC method was developed, and it is highly expected to be applied to many cases such as ground improvement and the construction of impermeable walls as shown in Figure 2 [1].

The CCC method utilizes a pile driver even if it is a chain saw type. This method was established in order to perform a variety of construction by taking the equipment body in and out in the same manner as the auger type. In this method, the soft ground is strengthened by mixing the soils and cementing materials which are injected into the ground from the top of the chain through the agitation and the mixing process. The advantages of this method are described as follows:

i) Vertically homogeneous ground is formed by mixing the soils and the cementing materials in a vertical direction.
ii) Continuous wall of equal thickness is constructed by the CCC method since the cross-section of improved area becomes a rectangular cross-section due to the cross-sectional shape of the equipment.
iii) Its versatility and high economic efficiency

When an impermeable wall is constructed in ocean disposal sites, the CCC method is useful for the construction since it allows us to construct the wall which strength and permeability are homogeneous in a vertical direction [1].

In the conventional method, an impermeable wall is constructed under the ground before the construction of the embankment, followed by the construction of an impermeable wall along the slope of the embankment as shown in Figure 3(a). Thus, caisson type, slope type, sheet pile type, and the combination of them in Figure 3(a) have been introduced to the embankment in ocean disposal sites [2]. On the other hand, when the CCC method is applied to the construction of ocean disposal sites, an impermeable wall is constructed through the revetment and the ground after the construction of the revetment as shown in Figure 3(b). The difference in the structure of the impermeable wall in Figures 3(a) and (b) may result in the change in performance of the impermeable wall. In order to apply the CCC method to the construction of the impermeable wall in ocean disposal sites, the performance of the impermeable wall constructed using CCC method has, therefore, to be compared with walls constructed using the conventional method.
In this paper, the leakage of contaminants from the waste storage area in ocean disposal sites was simulated with Dtransu-2D, which considers advection and diffusion of pollutants, aiming at comparing the performance of the impermeable wall constructed using the CCC method and the conventional model. Based on the results, the application of the CCC method to the construction of the impermeable wall in ocean disposal sites was discussed.

![Diagram](a) Constructed using the conventional method (an impermeable wall with impermeable sheets)

![Diagram](b) Constructed using the CCC method

Figure 3. Schematic view of the ocean disposal site

4. Analysis for application of construction of ocean disposal sites

4.1. Analysis model

Figure 4 shows a schematic view of the analysis model, and the input parameters are listed in Table 1. The model was set at 50 m in height and 110 m in width, and the riprap revetment with a 10 m height was set between the ocean area and the waste storage area as shown in Figure 4. The ground was set until 40 m depth, and the impermeable wall was embedded into the ground. Whereas the impermeable wall constructed using the CCC method was set as illustrated in Figure 4, the wall constructed by the conventional method was set along the slope of the embankment as shown in Figure 3(a). Additionally, the combination of the impermeable layer and an impermeable wall was set at the slope of the embankment in the disposal site constructed using the conventional method. The impermeable layer was composed of the protection mat, the impermeable sheets, and the backfilling materials as described in Figure 5. The layer composed of alluvial clay with 40 m in thickness was set up as the ground II under the layer composed of alluvial clay with 10 m in thickness which was set up as the ground I, as illustrated in Figure 4. The permeability of the ground I and II was set at 1.0×10⁻⁴ cm/sec and 1.0×10⁻⁵ cm/sec, respectively. The impermeable wall was embedded into the ground II until 2.5 m depth in the disposal sites constructed by the conventional method and the CCC method. The difference in hydraulic head between the ocean area and waste storage area was set at 1.0 m. The leakage of contaminants from the waste storage area was simulated for 50 years in the models. These input data was determined by reference to the past studies as summarized in Table 2 [2,3].

<table>
<thead>
<tr>
<th>Table 1 – Physical properties of the impermeable layer</th>
</tr>
</thead>
<tbody>
<tr>
<td>Permeability (cm/sec)</td>
</tr>
<tr>
<td>Impermeable wall</td>
</tr>
<tr>
<td>Impermeable sheet</td>
</tr>
<tr>
<td>Backfilling material Protection mat</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Table 2 – Input data</th>
</tr>
</thead>
<tbody>
<tr>
<td>Permeability (cm/sec)</td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td>Effective porosity (-)</td>
</tr>
<tr>
<td></td>
</tr>
<tr>
<td>Specific storage (1/m)</td>
</tr>
<tr>
<td>Longitudinal dispersion (m)</td>
</tr>
<tr>
<td>Transverse dispersion (m)</td>
</tr>
<tr>
<td>Coefficient of molecular diffusion (cm²/sec)</td>
</tr>
<tr>
<td>Tortuosity (-)</td>
</tr>
<tr>
<td>Retardation coefficient (-)</td>
</tr>
<tr>
<td>Adsorption coefficient (1/sec)</td>
</tr>
</tbody>
</table>
4.2. Boundary conditions
The boundary conditions of the analysis models are summarized in Table 3. The flow rate was set at zero at the bottom, right and left side of the analysis models and the top edge of the impermeable wall. Total hydraulic head was set at 10 m at the top edge of the ocean area and at 11 m at the top edge of the waste storage area as a fixed boundary: the difference in hydraulic head was set at 1 m. The concentration of contaminants was fixed at 1 in each location of the waste storage area.

4.3. Evaluation method
Some of the regulations in the environmental standard and the acceptable concentration of contaminants in ocean disposal sites are listed in Table 4. The environmental standard referred to the regulation enacted by the Ministry of the Environment in Japan [4], and the acceptable concentration referred to the Ordinance of Prime Minister’s Office [5]. The concentrations in the environmental standard are one-tenth of the acceptable standard. Therefore, it was considered that the contaminants leaked from the disposal site when the relative concentration of the contaminants in the ocean area became one-tenth of the fixed concentration in the waste storage area in this study.

Table 3 – Boundary conditions

<table>
<thead>
<tr>
<th>Elements</th>
<th>Environmental standard</th>
<th>Acceptable concentration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hydrargyrum</td>
<td>&lt; 0.01mg/L</td>
<td>&lt; 0.1mg/L</td>
</tr>
<tr>
<td>Hexavalent chromium</td>
<td>&lt; 0.05mg/L</td>
<td>&lt; 0.5mg/L</td>
</tr>
<tr>
<td>1-3-Dichloropropene</td>
<td>&lt; 0.002mg/L</td>
<td>&lt; 0.02mg/L</td>
</tr>
<tr>
<td>Tetrachloroethylene</td>
<td>&lt; 0.01mg/L</td>
<td>&lt; 0.1mg/L</td>
</tr>
</tbody>
</table>

Table 4 – Environmental standard and acceptability standard for waste

<table>
<thead>
<tr>
<th>Hydraulic head (m)</th>
<th>Top edge of ocean area</th>
<th>10</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fixed concentration (-)</td>
<td>Top edge of waste storage area</td>
<td>1</td>
</tr>
<tr>
<td>Flow rate (-)</td>
<td>Bottom, right and left side of the model, and top edge of the impermeable wall</td>
<td>0</td>
</tr>
</tbody>
</table>

5. Result and Discussion of Analysis for application of construction of ocean disposal sites

5.1. Comparison of the performance of an impermeable wall
The performance of an impermeable wall constructed using the CCC method were compared to that constructed using the conventional method in an ocean disposal site. Figure 4 shows the concentration of contaminants after 50 years in the analysis models. The area with red color indicates higher concentration of contaminants. The contaminants spread across a wide area in the disposal site constructed using the conventional method in Figure 4(a), in comparison with that by the CCC method in Figure 6(b). More than 0.1 of the concentration of contaminants was observed in the ocean area in Figure 6(a). On the other hand, the leakage of contaminants was not observed in the ocean area after 50 years in Figure 6(b). Compared to the result in the disposal site constructed using the CCC method in Figure 6(b), the contaminants, additionally, spread widely along the impermeable wall under the waste storage area in the site constructed using the conventional method in Figure 6(a). Thus, the difference in the structure of the impermeable wall in the disposal site constructed using the conventional method and the CCC method affected the performance of the impermeable wall for the prevention of the leakage of contaminants from the waste storage area. It can be said that the impermeable wall constructed using the CCC method is appropriate for the prevention of the leakage of contaminants based on the results.

Considering the spread of pollutants along the impermeable wall in Figures 6(a) and (b), the water flow near the wall has to be discussed to understand the transfer of pollutants. Figures 7(a) and (b) show the total hydraulic head in the analysis models, and Figure 7(c) illustrates the enlarged view of total...
hydraulic head near the impermeable layer in the site constructed using the conventional method. The equipotential lines between the points of same hydraulic head are, moreover, drawn in Figure 7. Since the lines represent the difference in hydraulic head, the large distance between the lines indicates that hydraulic gradient becomes smaller with the decrease in water flow. Meanwhile, hydraulic gradient becomes larger with the increase in water flow when the distance of the line is small [6]. In Figure 7(a), the distance of the lines is, especially, small near the impermeable wall, indicating that the flow rate is high with the increase in hydraulic gradient along the wall. Additionally, the distance of the lines became small near the impermeable layer along the slope of the revetment in the site constructed by the conventional method as shown in Figure 7(c). This area corresponded to the boundary of the waste storage area and the revetment in Figure 7(a), where the concentration gradient is high since the difference in the concentration of pollutants is maximum between the areas. According to the Fick’s laws of diffusion, the diffusive flux is proportionate to the concentration gradient [7]. Therefore, the spread of pollutants was accelerated by diffusion and advection due to the high hydraulic gradient and concentration gradient along the slope of the revetment in the site constructed using the conventional method. In the disposal site constructed using the CCC method in Figure 7(b), hydraulic gradient was high along the impermeable wall, while the concentration gradient was high at the boundary of the waste storage area and the revetment. Hence, the contaminants spread across a wide area in the site constructed using the conventional method due to the confluence of the areas along the slope of the revetment, where the concentration gradient and the hydraulic gradient were high. In the area under the disposal storage area, the distance of the equipotential lines is small near the impermeable wall in the site constructed using the conventional method compared to that constructed using the CCC method in Figures 7(a) and (b). It also caused an increase in flow rate near the impermeable wall owing to the increase of hydraulic gradient, resulting in the spread of pollutants in a wide area in Figure 7(a).

5.2. Effect of the location of an impermeable wall on the spread of pollutants

Considering that the difference in the structure of an impermeable wall affected the spread of pollutants as discussed in the previous section, the distance between an impermeable layer and the waste storage area may also affect the spread. For that reason, the effect of the distance of an impermeable wall and the waste storage area on the spread of pollutants was discussed in the three models as shown in Figure 6: an impermeable wall was set at ocean side of the revetment in Figure 8(a), it was set at the center of the revetment in Figure 8(b), and it was set at the waste storage area side of the revetment in Figure 8(c). The location of the impermeable wall differed by 5 m in each model. In this case, the permeability of the impermeable wall was set at 1.0 x 10^{-6} cm/sec and that of the ground was set at 1.0 x 10^{-3} cm/sec in each location of the ground. The length of the embedded impermeable wall into the ground was set at 10 m from the bottom of the sea, and the thickness was set at 100 cm. The difference in total hydraulic head between the ocean area and the waste storage area was set at 1.0 m.

The time for the pollutants to reach the ocean area under each condition is summarized in Table 5. Figure 7 shows the total hydraulic head with the equipotential lines in each model. The time decreased with the reduction of the distance between the permeable wall and the waste storage area as shown in Table 5. The distance of the equipotential lines became, furthermore, smaller in the area under the waste storage area with the decrease of the distance in Figure 9. This indicated that hydraulic gradient and flow rate were high near the impermeable wall under the waste storage area when the impermeable wall was constructed near the waste storage area [6]. The increase in flow rate caused the leakage of pollutants from the waste storage area earlier when the impermeable wall was constructed near the waste storage area. Therefore, the impermeable wall should be constructed on the ocean side for the prevention of the spread of pollutants.

6. Construction of impermeable wall by CCC

In January 2010, the CCC was adopted to the public works of restoring Miike Harbor in Omuta City, Japan. As a part of the entire works, two settling ponds were constructed for receiving dredged materials and the CCC took part in constructing impermeable walls of those settling ponds for preventing the contaminated water from flowing out, and from disturbing the environment of the surrounding area. The specifications of the first walls were 0.75 m in thickness, 7 to 8 m in depth, a total 1,128 m long and the total area was 8,470 m². The requested hydraulic conductivity value of the wall was less than 1.0 x 10^{-6} cm/sec. For ensuring the impermeability of the wall, soil samples were taken at the site and mix design tests were conducted. The obtained values were less than 2.0 x 10^{-7} cm/sec which is five times less than the designated value 1).

Based on the test, the mix design was decided as follows:

- Type of cement  Blast furnace cement
- Cement factor 150 kg/m² of soil in-situ
- Water/cement ratio 150%

The CCC started the wall works of the first settling pond on 23rd of January 2010 and finished spending 35 days. The core samples taken from the constructed wall showed good results as shown in Table 6.

The hydraulic conductivity of the constructed wall is much less than designated value that was obtained from the mix design test. The productivity of the CCC was 250 m²/day and the cost of wall production was more than 20% cheaper than those using the conventional method. Figure 10 shows the CCC working at the first site.
Figure 6. Concentration of contaminants after 50 years in the ocean disposal site: (a) constructed by the conventional method; (b) constructed by the CCC method.

Figure 7. Total hydraulic head after 50 years in the ocean disposal site: (a) constructed by the conventional method; (b) constructed by the CCC method; (c) enlarged view along the impermeable layer in the site constructed by the conventional method.

Figure 8. Analysis model with the impermeable wall: (a) at ocean side; (b) at central point; (c) at waste storage area side.
Figure 9. Total hydraulic head in each model with the impermeable wall: (a) at ocean side; (b) at central point; (c) at waste storage area side

### Table 5 – Time when the pollutants reach to the ocean area

<table>
<thead>
<tr>
<th>Location of impermeable wall</th>
<th>Ocean side</th>
<th>Center point</th>
<th>Waste storage area side</th>
</tr>
</thead>
<tbody>
<tr>
<td>Time (year)</td>
<td>42</td>
<td>13</td>
<td>5</td>
</tr>
</tbody>
</table>

### Table 6 – Hydraulic conductivity of the diaphragm wall

<table>
<thead>
<tr>
<th>Bore Hole No.</th>
<th>Hydraulic Conductivity</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Upper</td>
</tr>
<tr>
<td>1</td>
<td>4.83 x10^{-8}</td>
</tr>
<tr>
<td>2</td>
<td>2.63 x10^{-8}</td>
</tr>
</tbody>
</table>

The second work started in April, 2010 and finished in July in Arao city located south of Omuta city, as shown in Figure 11. The specifications of the walls were 0.75 m in thickness, 8 to 10 m in depth, a total 1,197 m long and the total area was 10,420 m². The requested hydraulic conductivity value of the wall was same as the first wall of less than 1.0 x 10^{-6} cm/sec. According to the mix design test, the cement factor and the water/cement ratio were changed from 150 to 200 kg/m³ of soil in situ and from 150 to 100%, respectively. This was due to the occurrence of thick layers of sand and gravel which easily pass the water and makes it difficult to establish an impermeable wall.

The actual performance of the CCC had interference caused by the hard rock filled in the ground and hard soils that were encountered. The N value of the hard ground was more than 50. In spite of the adverse conditions, the CCC finished the second work as scheduled and the samples taken from the constructed wall satisfied the target value of 2.0 x 10^{-7} cm/sec.

Figure 10. Scenery of high power CCC

Figure 11. Diaphragm wall surrounding 2nd pond

### 7. Conclusions

Since the beginning of the development works of the CCC in 2002, the CCC demonstrated the advantage of its chain saw type mechanism in the field of constructing soil/cement mixed walls. The advantages of the CCC method include not only cost and the days spent for completing given works, but also the uniformity and quality of the constructed structure which is impossible for the existing auger type machines.

Moreover, the spread of pollutants from ocean disposal sites was simulated with Dtransu-2D, aiming at comparing the performance of the impermeable wall constructed using the CCC method and the conventional method. The result indicated that the leakage and the spread of pollutants were prevented in the disposal site where the impermeable wall was constructed using the CCC method compared to the wall constructed using the conventional method. Moreover, the pollutants reached to the ocean area earlier with the decrease in the distance between the impermeable wall and the waste storage area due to the effect of the high flow rate near the impermeable wall. Therefore, the CCC method can be applied to the construction of an impermeable wall in ocean disposal sites. The distance between the impermeable wall and the waste storage area in ocean disposal sites should be larger in order to minimize the effect of water flow near the impermeable wall on the spread of pollutants from the waste storage area.
References


MPES2017

Session 3 – Rock Mechanics & Mine Safety

15:30-17:00

August 29, 2017
Utilizing production data to predict operational disturbances in sublevel caving
A data driven approach

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Abstract—The ability to successfully charge blastholes is of great importance in sublevel caving (SLC) operations. The amount and distribution of explosives along the length of a blasthole directly influences local specific charge and will therefore affect fragmentation and recovery accordingly. Partially or fully uncharged blastholes will result in poor fragmentation and insufficient mobilization of the blasted material. Blasthole instability has been found to affect the degree of which a blasthole can be charged. Blockages and/or blasthole collapses, due to poor rock mass quality, are not uncommon in some mining areas. Indications are given that, for some areas in LKAB’s Malmberget mine, the charged length is, on average, less than 70% of the drilled length. The costs and implications of this are not entirely investigated. Drill monitoring technique (e.g. MWD) could potentially be used to predict chargeability, scaling from individual holes to entire ore bodies, and therefore allow for an effective planning of various mining activities. Charging procedures, re-drilling operations and machine usage could all be optimized for the predicted conditions. Further, incorporating chargeability predictions into existing fragmentation models would potentially allow for more accurate fragmentation estimations.

Keywords: fragmentation, sublevel caving, SLC, MWD, chargeability

1. Introduction

MWD (measurement while drilling) technology has proven to be an effective tool for obtaining information on the rock mass quality in sublevel caving (SLC) rings [1]. The rock mass quality is normally regarded a controlling factor in terms of blasthole stability [1]. The rock mass conditions, provided by drill rigs equipped with MWD technology, may therefore be used to predict which holes in a SLC ring that, most likely, will be charged only partially or not at all [1]. A decrease in chargeability (charged vs. drilled length) will entail a lower local specific charge within a SLC ring. Furthermore, the specific charge is considered a dominant factor affecting fragmentation [2] and recovery [3].

Andersson [4] studied chargeability in LKAB’s Malmberget mine and indicated that chargeability in the mine, on average is around 90%. However, individual areas had chargeability rates of only 70% and individual rings, in those areas, suffered from chargeability rates of as low as 50%. However, the limited amount of data collected and analysed in this study entails large uncertainties in terms of actual numbers.

Generally, no predictions are made in terms of which mining areas that are likely to suffer from low chargeability rates. Hence, expected production disturbances related to poor fragmentation, due to charging and/or blasting issues, are not planned or accounted for on a local scale. There is a lot of potential in incorporating a probabilistic approach with actual predictions of chargeability, based on drill monitoring technique (e.g. MWD) [1]. A probabilistic approach based on input from drilling, could be utilized to more accurately predict fragmentation on a local scale to plan charging procedures, re-drilling operations and optimize machine usage.

1.1 Fragmentation in SLC

Fragmentation plays a key role in sublevel caving. The effect on downstream operations has been widely discussed over the years, where loading, transport, crushing and grinding all, to some extent are influenced by variations in fragmentation [5]. Also important, gravity flow and recovery in SLC is highly dependent on fragmentation of a blasted ring. Wimmer [6] indicated that fragmentation with a low characteristic size and a high uniformity index generates better ore recovery and a decreased dilution.

The efficiency of the LHD operation is also affected by variations in fragmentation. Excessive fines in combination with water entail suboptimal digging conditions which decreases overall LHD productivity [7]. Boulders may result in dangerous hang-up situations and needs to be removed separately for secondary blasting.

In LKAB’s Malmberget mine, oversized boulders (OSBs) are defined as anything larger than 1x1x1m. Grizzlies are commonly not utilized at the ore pass inlet and observations
have indicated that OSBs are sometimes dumped into ore passes entailing costly production stops [8].

Further, fragmentation has proven to correlate well to the energy requirement of the crusher. An increase in the median fragment size of a sample entails an increase in energy required to crush the corresponding material [9].

1.2 Assessing fragmentation

Most underground mines do not continuously monitor fragmentation in pre-crushing stages. This leaves significant uncertainties in terms of blast performance and impact on the efficiency of pre-crushing operations such as loading and material handling (incl. ore pass disturbances) [8]. In many cases it is not possible to evaluate the cause of occurring problems that negatively influence the productivity of the mine due to lack of observation and documentation.

The problems associated to continuous monitoring of fragmentation underground is commonly related to environmental factors such as dust, water and poor lighting conditions [5].

Comparing fragmentation from a set of production blasts in SLC is inherently difficult. Power [10] indicated that the primary recovery, i.e. material blasted in the current ring, constitutes on average 59% of the loaded material at the corresponding drawpoint. Secondary recovery, i.e. material originating from the level above, combined with primary recovery constitute on average 75% of the loaded material. Furthermore, dilution entry occurs from above at low extraction rates (<20%).

The particle size distribution of material measured at a drawpoint is therefore heavily influenced by material from the levels above and in certain cases from previous rings. Power [10] concludes that; “... it is unrealistic to expect that all material is recovered on the level from which it is fired”. Lith et al. [11] further discusses the possibility that poorly fragmented and/or compacted ore is left behind in the ring. This material will therefore not be recovered at the level of which it is blasted. The complex nature of gravity flow in SLC effectively inhibits a single-level local approach to evaluation of fragmentation in respect to design changes or known local disturbances. Marker trials and extensive fragmentation monitoring on several levels are required as the material origin has to be controlled to accurately assess and evaluate blast performance for a given ring.

1.3 Blasthole instability

Studies [12], [13], [14] conducted in LKAB’s Malmberget mine has identified a number of blasthole instability problems, namely:

- Shearing
- Caving
- Cavities
- Weak rock
- Fractured rock
- Crushed zones

The inability to charge a blasthole due to in-hole collapse and/or blockage is caused by shearing, weak rock, fractured rock and crushed zones. The standard procedure in Malmberget is to initially attempt to clean the hole with the charging hose. If not successful, re-drilling may be ordered depending on the severity of charging problems occurring in a ring [1]. If the problems are not severe or if re-drilling is not possible, the blasthole may be left partially or fully uncharged to avoid further delays and costs.

Cavities may disturb the charging operation, inhibiting charging of the full hole length. Excessive explosives may be pumped into a cavity instead of distributing it along the hole [13]. However, in case of cavities, charging of the full hole length may be possible although the risk of explosive leakage into the cavities is increased [1].

1.4 MWD

Drill monitoring techniques (e.g. MWD) is used in mining and petroleum industries to characterise the penetrated rock mass [17]. Information on the hardness, fracturing and weathering of the rock mass [15] can be obtained through analysis of the recorded parameters. Drill monitoring does not cause disturbances as it is conducted automatically during drilling. The results can be presented rapidly [16] and at an early stage in production. In surface mining operations, the technology has been utilized since the 1970s [17]. Information obtained from MWD can be used to improve blast design and fragmentation [17], [18].

The indicated stability of a blasthole can be obtained from MWD data and hence provide indications on the charging conditions [1]. The chargeability is in turn influencing the local specific charge which impacts on fragmentation and recovery [19].

This paper presents results from a test in LKAB’s Malmberget mine where the fragmentation of a number of rings, including roughly 10 000 tonnes of ore each, together with other registered data (MWD data, loading data, etc.) have been analysed, aiming at understanding and controlling the effect of fragmentation on production and productivity.

2. Material and Methods

2.1. Fragmentation measurement

Fragmentation has been monitored by mounting a surveillance camera (Axis Q-3505 VE) at the ore pass inlet. The camera was triggered by motion and videos of every passing bucket were stored locally in a NAS (Network Attached Storage). A program was developed to extract specific frames from the obtained video files and a GUI (Graphical User Interface) was programmed to allow for quick, yet careful, evaluation of fragmentation.

The reference system is similar, but not identical, to the QRS (Quick Rating System) developed by LKAB [20]. The system relies on visual classification where the user categorizes each image based on a reference system, see figure 1. The reference system is created by utilizing the
software split desktop to provide values of $x_{50}$, or the median passing fragment size, for various images. The observed images are then compared to the reference images and classified accordingly.

<table>
<thead>
<tr>
<th>Reference</th>
<th>Category</th>
<th>$x_{50}$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Class I</td>
<td>0-50mm</td>
<td></td>
</tr>
<tr>
<td>Class II</td>
<td>50-150mm</td>
<td></td>
</tr>
<tr>
<td>Class III</td>
<td>150-400mm</td>
<td></td>
</tr>
<tr>
<td>Class IV</td>
<td>400-1000mm</td>
<td></td>
</tr>
<tr>
<td>Class V</td>
<td>&gt;1000mm</td>
<td></td>
</tr>
</tbody>
</table>

Figure 1: Reference system

Analysis was conducted for every bucket up to 100% extraction ratio (~500 buckets per ring), see figure 2. Uniformity of the sample material was not analysed at this time. The frequency of each category during the course of extraction was used to give indications on overall ring performance and downstream operational impact.

### 2.2 MWD

MWD parameters are monitored on all 6 Atlas Copco W6C rigs used in the mine. The rigs are equipped with Wassaras hydraulic ITH machines and the recorded parameters are penetration rate, percussive (water) pressure, rotation pressure and feed pressure. From those parameters, the chargeability of the drilled hole can be predicted [1]. MWD data from drill rigs supplemented by borehole filming was analysed to identify and categorize blasthole stability issues, see figure 3. This was later correlated to chargeability issues emerging in production rings. The level of fracturing inside the blast hole gives a statistical measure of how likely that a hole will be charged properly or not. Blast holes drilled in solid rock will more likely remain stable until charging than a fractured blast hole [1].

Figure 2: Fragmentation vs. extraction rate

Figure 3: Calculated fracturing (Ghosh et al. 2017)

### 2.3 Wolis

In LKAB, WOLIS (Wireless Online Loading Information System) is used to wirelessly transfer data from the LHDs to the database [21]. An ore grade characterization system based on the LOADRITE scoop weighing system is used for all LHD machines [22]. The system measures the hydraulic pressure in the lift cylinders of the LHD’s arms connecting the machine to the bucket. Since the ore and waste density are significantly different in Malmberget the system is used to predict the amount of ore in the bucket. In this study this system has been used to assess the ore grade for each bucket.

### 3. Results and Discussion

Ring 9 and 11 were selected for the initial analysis. The rings are located only 6 meters apart but the operational impact of the material loaded at the corresponding drawpoints varies significantly. The fragmentation, ore grade and indicated charging problems, based on drill monitoring, for both rings was considered in the analysis.
3.1 Fragmentation measurements

Fragmentation for the two observed rings varies considerably, see figure 4. Low amount of fines and high amount of boulders was observed for material extracted at the drawpoint of ring 9. Oppositely, high amount of fines and low amount of boulders was observed for material extracted at the drawpoint of ring 11.

![Fragmentation Ring 9 & 11](image)

Figure 4: Fragmentation ring 9 & 11

The amount of boulders loaded up until 100% extraction rate was 8.5 boulders/kton (17%) for ring 9 and 0.5 boulders/kton (~1%) for ring 11.

3.2 Ore grade measurements

The ore grade for the two rings also varies considerably. Ring 9 has, on average, a significantly lower grade and a high number of waste rock peaks, see figure 5. Ring 11 has a high average ore grade and a low amount of waste rock inflow, see figure 4. Ring 9 was closed at 114% extraction rate whereas ring 11 was closed at 128%. Ring 11 has a very high Fe-grade except for a short section between 60% and 80% excavation rate. Ring 9 on the other hand, show relatively low Fe grade, from the beginning and throughout the excavation of the ring, with a cyclic pattern with alternating lower and higher Fe-grade with increasing extraction rate.

![Ore grade Ring 9 and 11](image)

Figure 5: Ore grade ring 9 and 11

3.3 MWD analysis

The MWD analysis, see figure 6, indicates that one of the blast holes in ring 9 has cave-in issues and a high probability of collapse or blockage [1]. Ring 11 is constituted by fairly intact rock with fracture zones in some of the holes.

![MWD assessment](image)

Figure 6: MWD assessment

3.4 Discussion

One of the largest production challenges for LKAB’s sublevel caving mines is the significant variation in each step of the underground production chain. For an efficient production planning it is important to have a reasonable long planning horizon, during which the prerequisite for the production is well known. One of these most important prerequisite is the fragmentation of the blasted material that often varies significantly within short time intervals. Being able to predict which mining areas that will suffer from poor fragmentation in advance and possibly control, or at least plan, for the effects on downstream operation would provide a unique opportunity for production improvement.

This paper discusses the different data sources available on the loading level at LKAB’s mine in Malmberget. MWD data, which continuously are monitored on all six production rigs, has a potential in providing useful information at an early stage of production, before charging, blasting and loading. This information could be used for optimizing charging procedures but also predicting whether fragmentation will be favourable or not in specific areas of the mine. Planning of re-drilling operations and optimizing machine usage for loading
and transport could also be incorporated into current production planning. The origin of the measured material at a drawpoint is, to a great extent, unknown. Based on findings in the literature, indication is given that a large part of the material observed at the drawpoint comes from above. The poor ore grade and coarse fragmentation observed for ring 9 could potentially be explained by a sub-optimal gravity flow where excessive dilution disturbs the measurement of the material actually originating from the ring. It might therefore be difficult to spot the actual impact of partially or fully uncharged blastholes in terms of fragmentation due to the excessive noise caused by dilution. However, it is fair to conclude that MWD data could be used to identify larger zones of poor quality rock and that utilization of that information could provide useful indicators on expected operational issues likely to occur in broader terms and on a larger scale.

Large scale studies are required to properly evaluate the effect of charging and blasting in areas of poor rock mass quality. The obvious problems associated to partially or fully uncharged blastholes discussed in this paper should be complemented by evaluation of problems occurring due to explosive leakage in cavities and blasting properties of heavily fractured rock (wave propagation etc.).

In this study a manual follow-up on the charging operation have been done to calibrate the MWD model, but a detailed, digital monitoring is missing and may complement the analysis. The on-line estimation of iron ore content, used in the mine, together with the predicted fragmentation, may also provide a unique tool for production planning to improve the selection of active production rings to meet the requirements on grades and tonnage from the processing plant.

4. Conclusions

The selected rings are located only 6 meters apart and the layout is close to identical. However, fragmentation and ore grade of the loaded material in the corresponding drawpoints differs substantially. The operational implication for the two scenarios are vastly different as the impact on the LHD operation, ore pass function, and crushing stage are sensitive to large variations in terms of fragmentation. The rock mass quality, indicated by MWD, provide some explanation why the rings are different. Ring 9 has one blasthole that is likely subjected to stability issues. However, at this point, no information is available if that, or any of the other holes, was charged or not. The uncertainties associated to evaluating fragmentation on a single level without monitoring the material flow inhibits conclusive explanations to why the loaded material from the observed drawpoints is greatly different in terms of size distribution and ore grade. However, potential methods of predicting where operational disturbances are likely to occur are discussed in the paper.

References


Fundamental Study on Quantitative Evaluation and Prediction Method of Cutter Bit Wear for Shield Machine in Gravel Ground

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Abstract—It is unavoidable to have cutter bit wear during shield tunnel excavation. In recent years, because of a variety of ground conditions being excavated, problems caused by excessive cutter bit wear sometimes happen in projects. Many studies on cutter bit wear have been done in past but quantitative evaluation method of cutter bit wear in advance of excavation has not been established. Especially, there is few research on the characteristics of bit wear in gravel ground. This paper discussed the effects of characteristics of gravel and gravel contents on the bit wear based on the results of a series of laboratory tests in order to understand the mechanism of bit wear in gravel ground and develop the prediction method of bit wear in it.

Keywords: Bit wear, gravel ground, characteristics of gravel, gravel contents, mechanism of bit wear

1. Introduction
A shield method is now widely applied to construct tunnels for infrastructures. Because this method can be applied for various geological conditions and has small impacts on road traffics and surrounding environment. This method is that the shield is pushed into ground with cutting and maintaining the stability of the cutting face [1]. The shield machine has the cutter head with bits and the ground is drivaged by rotating the cutter head and pushing it by thrust. The bit wear during cutting operation is inevitable as shown in Figure 1. Nowadays, as a closed type shield machine is mainly adopted and the conditions of tunnel construction become to be wide, the operating issues due to the bit wear often occurs and becomes seriously. As the bit wear has an obvious impact on the construction cost and constrains, such as lowering of drivage efficiency, increasing the frequency of bit replacement, etc [2]. Therefore, the prediction of cutter bit wear in advance is very important in order to make adequate construction plan and calculate estimated cost. However, there is few research on the characteristics of bit wear in gravel ground and the prediction method of the bit wear in theoretically and quantitatively. Because the mechanism of bit wear in gravel ground seems to be very complicated compared with rock mass ground as shown in Figure 2.
From the results of previous researches [3], the bit wear when a gravel ground is excavated can be evaluated and predicted based on the abrasiveness of gravel itself. So, it can be expected that the characteristics of gravel itself and the gravel contents have an obvious impact on the characteristics of bit wear when the gravel ground is excavated by shield machine.

From above points of view, this paper discussed the effects of characteristics of gravel and the gravel content on the characteristics of bit wear in gravel ground based on the results of a series of laboratory tests for the simulated sample of the gravel ground in order to develop the prediction method of bit wear for excavation in gravel ground.

2. Laboratory Tests

2.1. Sample preparation

The specimen of gravel ground was made of gravel and cement in this research as shown in Figure 3. Three different types of rocks: sandstone, andesite and granite were used as the gravel. The size of gravel was from 10mm to 20mm. Portland cement was used as the binding material of the specimen. The gravel contents of specimens were changed as 0%, 25%, 50% and 75%. The water cement ratio of binding material was also changed as 1:1, 1:1.5 and 1:2 in order to discuss the effect of strength of binding material on the characteristics of bit wear in gravel ground. The curing time of specimen before testing were 7days and 28days.

2.2. Mechanical and fundamental properties of gravel and gravel ground

As the mechanical properties of gravel seems to affect the characteristics of gravel ground, the uniaxial compressive test and the shore hardness test were conducted by using three types of rocks used as gravel of specimen. The XRD analysis was also conducted in order to measure quartz content because the quartz is high harness mineral and has a significant impact of bit wear [4].

2.3. Bit wear tests

2.3.1. CERCHAR Bit Abrasive Test

It is necessary to evaluate the abrasively of gravel and gravel ground in order to predict the amount of bit wear in quantity. The CERCHAR bit abrasive test was conducted by using specimen of gravel and that of gravel ground. The CERCHAR abrasive test is a laboratory method to quantify the
rock abrasively. It allows to determine an index called CERCHAR Abrasivity Index (CAI) for the rock's abrasivity which can be used for evaluate the wear of excavation equipment in different application such as mining, tunneling and drilling. The method was initially developed in 70's by the Laboratoire du Centre d'Etude et Recherches des Charbonnages (CERCHAR) de France for coal mining purposes [5].

Figure 4 shows the equipment of The CERCHAR bit abrasive test. The testing principle is based on a steel pin with defined geometry and hardness that scratches the surface of a rough rock sample over a distance of 10mm under static load of 10N. The pin and the dead load are moved across the rock surface. The pin is made of a standard steel and has a 90° conical tip.

![Figure 4. Equipment for CERCHAR bit abrasive test.](image)

The specimen was placed in the equipment and firmly clamped using a rigid vice. The device is rigid and also fixed to avoid any lateral movement. The dead load was then placed on the top of the pin and the pin is carefully lowered to the rock surface. The test was then carried out by relative displacement of the pin on the rock surface across 10mm at given time intervals. After the test, the pin is carefully removed and the tip flat wear is measured.

The CERCHAR - Abrasivity-Index (CAI) was then calculated from the measured diameter of the resulting wear flat on the pin:

\[ \text{CAI} = 10 \times d \]  

(1)

Where, CAI: CERCHAR Abrasivity Index (1/10mm), d: diameter of wear flat measured in mm to an accuracy of 0.01mm.

2.3.2. Lathe Bit Wear Test

In order to simulate the cutting process by bit, the Lathe bit wear test was conducted [6]. Figure 5 shows the equipment of the Lathe bit wear test.

This is an abrasion test and the basic principle is to press a test bit against a rotating specimen in a horizontal lathe. The specimen then is moved along the log surface for some time, and finally removed and measured the mass decrement of the bit.

![Figure 5. Equipment for Lathe bit wear test.](image)

In this test, the size of specimen was 50mm in diameter and 100mm in length, respectively. The material of the test bit was SKC24. The rotation rate of specimen and the static load applied to test bit were determined based on the actual conditions (rotation rate and thrust). The static load of test bit was fixed as 1N and rotation rate of specimen was changed from 72rpm to 144rpm. Moreover, the temperature of the top of the bit was measured by infrared thermometer.

Lathe Bit Wear Index was defined and calculated as follows:

\[ \text{Lathe Bit Wear Index} = \frac{\text{Mass decrement of bit}}{\text{Travel distance of bit}} \]  

(2)

Lathe Bit Wear Index = Mass decrement of bit / Travel distance of bit (mg/m)
3. Results and Discussions

3.1. Mechanical and fundamental properties of gravel

Table 1 shows the results of a series of laboratory tests for three types of rocks used as gravel. The CERCHAR Abrasivity Index of rock increases with increasing its compressive strength and/or shore hardness. It seems that the abrassivity of rock increase with increasing its strength and hardness of rock. However, no correlation between the Lathe Bit Wear Index and the compressive strength of rock can be found. It can be thought because several mechanisms and factors during cutting process affected on the bit wear in the Lathe bit wear test.

Table 1 - Results of a series of laboratory tests for gravel rock

<table>
<thead>
<tr>
<th>Type of rock</th>
<th>UCS (MPa)</th>
<th>Shore hardness</th>
<th>Quartz content (%)</th>
<th>CAI (1/10mm)</th>
<th>Lathe bit wear index (mg/m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone</td>
<td>27.6</td>
<td>44.3</td>
<td>21.4</td>
<td>0.42</td>
<td>0.22</td>
</tr>
<tr>
<td>Andesite</td>
<td>48.1</td>
<td>60.3</td>
<td>6.29</td>
<td>2.16</td>
<td>0.09</td>
</tr>
<tr>
<td>Granite</td>
<td>62.9</td>
<td>85.2</td>
<td>46.9</td>
<td>3.12</td>
<td>0.13</td>
</tr>
</tbody>
</table>

3.2. Mechanical properties of gravel ground

3.2.1. Mechanical Properties of Gravel Ground

Figures 6 (a) and (b) show the relationship between UCS and gravel content under different strengths of binding material and rock types of gravel at different curing time. From these figures, it can be seen that the strength of gravel ground decreases with increasing the gravel contents and it also increases with increasing the strength of binding material. The failure or fracture was found along the interface between binding material and gravel or inside of binding material not in gravel. So, it can be said that the strength of gravel ground is defined by the strength of binding material and/or the adhesion between the gravel and binding material not by the strength of gravel itself. On the other hand, these results also show that the specimens with different strength of binding material can be made by controlling the water cement ratio of binding material and/or its curing time.

3.2.2. CERCHAR Abrasivity Index of Gravel Ground

Figures 7 (a) and (b) show the relationship between the CERCHAR Abrasivity Index and the gravel content of specimen at different curing times. It can be found that the CERCHAR Abrasivity Index increases with increasing the gravel content. In other words, the mass decrement of bit increases with increasing the gravel content. It can also be seen clearly that the CERCHAR Abrasivity Index of gravel ground become to be large in the order of the specimen with sandstone gravel, that with andesite one and that with granite one, and this trend is the same as the CERCHAR Abrasivity Index of each type of rocks used as gravel. On the other hand, no effect of strength of binding material on the CERCHAR Abrasivity Index of gravel ground can be found in this result. Hence, it can be said that the mechanism of bit wear of gravel ground was different with that of uniform ground (soil and/or rock mass) and the characteristics of abrasivity of gravel ground is dependent on the properties of gravel itself.
3.2.3. Lathe Bit Wear Index

Figures 8 (a) and (b) show the relationship between the Lathe Bit Wear Index and the gravel content of specimen at different elapsed times. As no effect of strength of binding material on the Lathe Bit Wear Index of gravel ground can be found in Figures 8 (a) and (b), the average values were used in this discussion as shown in Figure 9. It can be seen obviously that the amount of bit wear increases with increasing the gravel contents. The amount of bit wear become to be large in the order of the specimen with andesite gravel, that of sandstone one and that of granite one, and this trend is different with the result of the CERCHAR bit abrasive test and the strength of gravel itself. The increasing ratio of the amount of bit wear associated with increasing gravel content were almost constant and same among three gravel types of specimens in case that the gravel content of specimen is lower than 50%. On the other hand, in case that the gravel content of the specimen is larger than 50%, it can be found that the increasing ratio of the amount of bit wear associated with increasing gravel content three types of specimens were changed and different gravel type of specimen shows the different trend as shown in Table 2. For example, the increasing ratio of the specimen with granite gravel is changed from 0.04 (gravel content: 0-50%) to 0.07 (gravel content > 50%), and that with andesite gravel is done from 0.03 to 0.04. Based on these results, it can be considered that the characteristics of bit wear of gravel ground is dependent on the characteristics of gravel. In other words, the mechanism of bit wear is different with different rock types of gravels and gravel content.
Both the CERCHAR bit abrasive test and the Lathe bit wear tests results show that the amount of bit wear increases with increasing the gravel content of specimen. However, the trend of the amount of bit wear in different types of gravels was different in two test results. This may be because of the different mechanisms of bit wear in two tests. In the CERCHAR bit abrasive test, the bit is worn by mechanical wear. On the other hand, several mechanisms affected on bit wear such as quartz content, thermal wear and chipping caused by impact in the Lathe bit wear test. Therefore, the effects of these major factors on the amount of bit wear were discussed in detailed as follows:

**Effect of quartz content of gravel**

Table 3 shows the quartz content of each types of gravel rocks. Compared with Figure 9, it can be seen that the quart content has an obvious impact on the amount of bit wear and the higher quartz content is the larger the amount of bit wear is.

**Table 3 - Quartz content of rocks used as gravel**

<table>
<thead>
<tr>
<th>Type of gravel</th>
<th>Quartz content (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone</td>
<td>21.4</td>
</tr>
<tr>
<td>Andesite</td>
<td>6.3</td>
</tr>
<tr>
<td>Granite</td>
<td>46.9</td>
</tr>
</tbody>
</table>

**Effect of chipping caused by impact**

When the gravel ground is excavated, the chipping of bit caused by impact due to collision with boulder is serious problem [3]. Figure 10 shows the tip of bits after Lathe bit wear test in different gravel types of specimen. From these figures, it can be seen that the tip of a bit which cut the specimen with sandstone gravel was flat and smooth surface. On the other hand, those with andesite and granite gravels were uneven and irregularities comparing with that with sandstone gravel. From these results, it can be considered that the bit chipping caused by impact due to collision with gravel have a significant impact on the total amount of bit wear when the gravel ground with andesite gravel and/or that with granite gravel are excavated. Therefore, the tip of the bit was observed by magnifying glass and the volume of bit loss was predicted as shown in Figure 11. Then the ratio of the mass decrement of bit due to the chipping was calculated as follows:

\[
\text{Ratio of bit mass decrement by chipping} = \frac{W - W_t}{W} \times 100 \, \% \quad (3)
\]

Where, \(W\): the total mass decrement of bit (mg), \(W_t\): the mass decrement of bit at the tip of bit (mg)

**Table 2 - Increasing ratio of the amount of bit wear**

<table>
<thead>
<tr>
<th>Type of gravel</th>
<th>Gravel content</th>
<th>Gravel content (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>25%-50%</td>
<td>50%-75%</td>
</tr>
<tr>
<td>Granite</td>
<td>0.04</td>
<td>0.07</td>
</tr>
<tr>
<td>Andesite</td>
<td>0.02 &gt;</td>
<td>0.01</td>
</tr>
<tr>
<td>Sandstone</td>
<td>0.03 &lt;</td>
<td>0.04</td>
</tr>
</tbody>
</table>

**Figure 9. Results of the Lathe bit wear test (average).**

**Figure 10 Shape of the tip of the bit after Lathe bit wear test.**

**Figure 11 Calculation of mass decrement of bit by chipping.**
Effect of Thermal Wear

Figure 12 shows the relationship between the temperature of the tip of the bit and travel distance in different three gravel types of specimen. As the travel distance was 33.92m in the Lathe bit wear test, the effect of thermal wear seems to have an obvious impact on the bit wear in case of specimens with sandstone gravel and that with granite one. Table 4 summarizes the effects of three factors mentioned above. The ratio of mass decrement of bit by chipping represented in Table 4 was obtained by cutting the specimen with 75% gravel content.

On the other hand, when the specimen with sandstone gravel and that with granite one are excavated, the amount of bit wear increases with increasing gravel content and, especially, the increasing ratio of the specimen with granite gravel is also increased in case that the gravel content is larger than 50%. As the temperature of the top of the bit is almost same in both gravel types of specimens as shown in Table 4, it can be considered that the effect of thermal wear on the amount of total bit wear seems to be the same degree and the effect of quartz content of gravel on the bit wear increases with increasing gravel content.

![Graph showing temperature vs. travel distance for different types of gravel]

**Figure 12 Relationship between temperature of the tip of the bit and travel distance of bit.**

Table 4 - Quartz content of gravel, ratio of the mass decrement of bi by chipping, temperature of tip of the bit.

<table>
<thead>
<tr>
<th>Type of gravel</th>
<th>Quartz content (%)</th>
<th>Ratio of bit decrement by chipping (%)</th>
<th>Temperature of the tip of bit (°C)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone</td>
<td>21.4</td>
<td>24.8</td>
<td>150</td>
</tr>
<tr>
<td>Andesite</td>
<td>6.29</td>
<td>60.8</td>
<td>50</td>
</tr>
<tr>
<td>Granite</td>
<td>46.9</td>
<td>56.4</td>
<td>150</td>
</tr>
</tbody>
</table>

Comparing Figure 9 with Table 4, it can be said that the amount of bit wear increases with increasing gravel content of specimen. Besides, the higher the quartz content of gravel is the larger the amount of bit wear is in case that the gravel content of ground is the same. Moreover, it can also be found that the trend of increasing bit wear due to the increase of gravel content are different with different types of gravels in case that the gravel content is larger than 50%.

From the results of the specimen with andesite gravel, no obvious increment of the amount of bit wear can be observed in case the gravel content is larger than 50%. As the quartz content of andesite is low and the temperature of the tip of the bit is relatively low as shown in Table 4, the chipping of the bit has a significant impact on the total amount of bit wear. Besides, the bit wear due to the chipping is not increased obviously even though the gravel content is increased in case that the gravel content is larger than 50%.

4. Conclusions

From the results of a series of laboratory tests, the following conclusion can be made:

1. The amount of bit wear increases with increasing gravel content of the ground.
2. The amount of bit wear increases with increasing quartz content of gravel in case that the gravel content is the same.
3. The effect of mass decrement of bit by chipping on the total bit wear has no obvious change even though the gravel content is increased in case the gravel content of ground is larger than 50%.
4. The effect of quartz content of gravel on the amount of bit wear becomes to be large in case that the gravel content of ground is larger than 50%.

From the above results, it can be concluded that the mechanism of bit wear in the gravel ground is changed due to the characteristics of gravel and gravel content of ground. In order to develop the prediction method for bit wear in gravel ground, further studies have to be conducted.

References

Factors Affecting the Selection and Performance of Raise Boring Machines (RBMs) and Case Studies from Turkey

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Abstract — Vertical or inclined shaft construction is now safe and speedy using most modern methods. Raise boring is one of the mechanized excavation methods and generally consist of two stages: pilot drilling and reaming. Raise boring machine (RBM) uses a small diameter drill rod, around 230-350 mm, to drill a pilot hole down to the required depth. Then, the pilot drill bit is removed and replaced with a large diameter reamerhead. The reamerhead is then pulled back up to the upper level by enlarging the hole diameter. Proper selection of RBMs is one of the important factors to improve the efficiency of the drilling / excavation and decrease the costs in mining and tunnelling projects. Some design and performance parameters of RBMs are reamerhead diameter, breakout and reaming torque, power of machine, thrust, cutter type and size, rotational speed, penetration rate and specific energy. Rock mass properties and the geological conditions are the other important parameters in the selection of RBMs. The focus of this paper is to itemize the required information for the proper selection of RBMs and predicting their performance. In order to reach this goal, different mines and tunnel projects were visited in Turkey to determine the geological conditions, specifications of the RBMs used in those sites and operational / performance parameters of the RBMs. It is believed that the paper would serve as a guide for efficient selection and design of the RBMs and performance prediction.

Keywords: Raise Bore Machine, RBM selection, Drilling / Excavation performance.

1. Introduction

Raise Boring method is a full face excavation method which means whole cross section of shaft is drilled to the desired diameter (max~8 m) with no use of explosive. Raise Boring Machines (RBMs) have been used for the construction of various shafts such as ventilation, personnel access, ore production, penstocks and surge chambers, and equipment access (pipes, cables) for mining, tunneling or other infrastructural projects. There is an increasing interest on utilizing RBMs for mining and tunnelling projects in Turkey.

The two important advantages of using RBMs are safer and faster operation. Product quality is another benefit of RBM over the drill and blast model. RBMs create a shaft with smooth walls, which usually does not require lining. The hole is more stable than a hole opened by drilling and blasting method and has better airflow, making it ideal for ventilation shafts. However, unfavorable ground conditions can cause the loss of drill strings (rod) and reaming tools; in addition, initial and capital cost of RBM are high.

RBM uses a small diameter drill rod, around 230-350 mm, to drill a pilot hole down to the required depth. Once the pilot hole has been drilled to the desired depth, the pilot drill bit is removed and replaced with a large diameter reaming head. The reaming head is then pulled back up to the upper level by enlarging the hole diameter. In the pilot drilling, the cuttings are removed from the hole with the aid of flushing. However, in the reaming stage, excavated material (muck) falls by gravity to the bottom of the hole from where is continuously removed. These machines can apply tension forces of about 23,000 kN and torques of about 1,050 kNm. Figure 1 shows typical raise boring operation in mines.

Figure 1. Typical raise boring operation in mines (courtesy of Sandvik).
Access to both top and bottom locations of the hole is a prerequisite for conventional raise boring method. The raise bore operations are commenced with site preparation. In this stage, a flat concrete foundation is made for the RBM in which the base plate is anchored with rock bolts. After transportation of power and hydraulic units to the project site, the RBM is aligned for pilot hole drilling. In the pilot drilling, the stabilizers are used for supporting the drill string and reducing the oscillation and bending stresses. Reaming is started with a low rotational speed and force until the collaring is completed [1]. Generally, a small water pool is prepared for the flushing water. The water pool size mainly depends on the hole size and density of the cut material being lifted.

Appropriate selection of RBMs is an important factor to increase the efficiency of the excavation and save costs in mining and tunnelling projects. In the light of this fact, the main objective of this paper is to itemize the required parameters for proper design and selection of the RBMs. Within the scope of this study, some construction projects and mines were visited in Turkey to determine the geological conditions, specifications of the RBMs used in those sites and operational parameters / performance of the RBMs. This study summarizes the experiences obtained about the selection and performance of RBMs as functions of mechanical, geological / geotechnical, and operational parameters.

2. Main parameters affecting selection and performance of RBMs

The parameters affecting the selection, design and predicting performance of a raise boring machine can be classified into three general categories: mechanical (machine related) parameters, geological / geotechnical parameters and operational parameters. A suitable RBM should be selected / designed for the geological conditions to be encountered during the shaft excavation. A detailed definition of each parameter is summarized below.

2.1. Mechanical parameters

The parameters related to the machine could be categorized as mechanical parameters [2]. These parameters are generally listed as:

- Machine dimension
- Machine weight
- Tricone bit and reamerhead type
- Tricone bit and reamerhead diameter
- Break out and reaming torque
- Power demand
- Thrust capacity
- Rotational speed

Utilizing the highest rotational speed and correct bit load for specific rock formation could lead to optimum rate of penetration and bit service life [3]. It is also worth pointing out that the cutters with larger row spacing are suitable for reaming in brittle or soft rocks. In addition, cutters with narrow spacing are suitable for reaming in very hard and tough rocks where the thrust forces cannot be fully utilized.

2.2. Geological / geotechnical parameters

An appropriate level of geological / geotechnical investigation should be performed prior to the consideration of the use of RBMs. This investigation should be covered information about rock mass properties and physical-mechanical properties of intact rocks as listed below [2, 4]:

- Rock Quality Designation (RQD)
- Identification of main faults zones, broken ground, layered ground
- Hydrogeology (ground water, water content, water ingress)
- Strength (uniaxial compressive strength, Brazilian tensile strength, elasticity modulus, cohesion, etc.)
- Abrasivity of drilled / excavated ground
- Mineralogical and petrographical properties
- Cuttability properties

Raise boring is convenient method for shaft excavation in competent rock; however, several problems are faced in geological formations where geological discontinuities are dominant. Identification of fault zones and broken ground along the shaft alignment are the key requirements for successful shaft excavation with RBMs. However, this is not easy task to confirm these features with conventional methods. High-resolution airborne electric-magnetic is one the new methods that is used in geotechnical investigations during excavation of rocks with Tunnel Boring Machines (TBMs). This method could also be tested during excavation of the shafts with the RBMs.

It is possible to find a lot of different raise bore machines in the market. A comprehensive rock-testing program is the most important requirement for the assessment of RBMs; and by use of the obtained data from the rock-testing program the best-suited raise bore machine for each project could be selected. Samples of all different rock types in the excavation area should be included in the laboratory studies. The laboratory testing program should include uniaxial compressive and Brazilian tensile strength, cutting tests, abrasivity of ground (with Cerchar abrasivity index) and petrographic analysis. The cutter life is a function of the determined abrasiveness value of rock samples. Petrographical thin section gives some important information about the mineral constituents and percentage of hard minerals that could have an impact on RBM penetration.

It should also be noted here that additional rock testing such as indentation test should be carried out to predict the accurate advance rates of the RBMs. In the literature, there are many paper related to indentation tests and its application for performance estimation of the RBMs [2, 5, 6, 7].
2.3. Operational parameters

The main items in the operational parameters that affect the selection and performance of the RBMs could be generally listed as:

- Site preparation
- Shaft dimension
- Shaft inclination
- Bailing system
- Utility facility (power, water, air)
- Applied torque, pulling / pushing force, penetration rate
- Labor availability and quality

Site preparation plays an important role in the elimination of delay and increasing the efficiency of the raise boring operation. Survey drawing and geological cross-section are the main information used by the site planners to select the best layout for RBM.

Shaft dimension and inclination are important subjects in raise boring operation. Generally, the excavation of vertical shafts is easier than that of the inclined shafts. In addition, the largest (8-meter diameter) RBM was used in the excavation of a new access shaft at “Jim Walter Resources” number 7 coal mine in Alabama [8]. Load per cutter, reamerhead diameter, geological formation and inclination of shaft are some important factors affecting the torque requirements. Increasing shaft diameter causes additional torque effort on drill string and reamer stem. In addition, with the increase in diameter, there is a greater potential for instability of the shaft walls and the advancing face of the shaft.

The bailing system directly affects the hole deviation and penetration rate parameters in the pilot drilling. In addition, insufficient bailing (especially in water) has the main impact on the pilot bit life. The water pool size mainly depends on the hole size and density of the cut materials being lifted.

At the beginning and the end of the reaming operation, the operational parameters of the RBM show lower values. At the beginning of reaming (collaring) when the cutters contacted with the roof rocks of bottom level due to the concerns on the over wear and failure of the drill string (rod), the rotational speed should be lower than that of the other rods. Whereas at the ending of reaming, when the cutters are near to floor of upper level, the rotational speed should be lower than that of the other rods. Otherwise, rock blocks could fall in the reamerhead and cause damage to the cutters. Moreover, experienced operator plays an important role in increasing the machine utilization time. The rod pulls and tears down could be decreased by a well-trained and experienced operator. In addition, appropriate schedule for various operation (such as blasting) in the mine could decrease the downtime of raise boring operation.

Instantaneous cutting rate is a function of mechanical and geological parameters, as well as technical and operational parameters. Cutter consumption rate is generally a function of geological (especially mineral content) and mechanical parameters. Machine utilization time depends on the delays (stoppages) caused by operational features of the projects.

3. Problems of incorrect RBM selection and performance estimation

Many qualitative and quantitative factors influence the RBM selection, so that these factors are often in conflicting with each other. Selection of the optimum RBM among many alternatives is a multicriteria decision making problem.

Geotechnical risks (hazards) play an important role on the selection of RBMs. In order to prevent some geological and geotechnical hazards during shaft excavation, selecting the most appropriate RBM is very consequential. The risk assessments should take into consideration on the operational conditions of the RBMs, as well as the geological conditions.

Unfavorable ground conditions cause the need for better and more reliable selection and performance estimation of RBMs. Various machine performance factors such as drillability/excavability of the rock, penetration rate and tool wear are directly influenced by rock mass conditions. When the geological faults encountered in the shaft excavation, the hole could be reinforced by grouting and plastic packers. In soft rock excavation, with rapid progress, attention needs to be paid to adequate clearing of the cuttings. In addition, the geometry of cutters in the reamerhead could be modified by adding a different number of button cutters to the cutter arrangement. It is also worth pointing out that the cutters with larger row spacing are suitable for reaming in brittle or soft rocks. In addition, cutters with narrow spacing are suitable for reaming in very hard and tough rocks where thrust forces cannot be fully utilized. The applied load on cutters in different formation is one of the important factors that directly affects the insert wear during the drilling / excavation operation. When the inadequate load applied to the cutters, the insert bits mounted on the cutters will wear quickly and this leads to the delays and loss of money in the projects. However, if the applied load on the cutters is higher than the required value, this situation also could cause jamming and wearing of the drill string.

There are some important problems in the raise boring operation including shaft wall stability, removal of cuttings, rock fragmentation and deviation. Among these problems, deviation is an important factor that directly affected from the mechanical and operational parameters. Generally, it is easier to drill a pilot hole and control the hole deviation by upward reaming than by downward reaming. The reference [9] stated that the high penetration rates and minimum deviations are very seldom achieved at the same time. In the downward reaming method, the drill string is under compression during pilot drilling and easily could bent. Deviation of the pilot hole is generally caused by anisotropy of strata, steeply dipping stratification with differing rock hardness, flushing system, machining accuracy of drilling devices. The basic condition
for effective cuttings removal is that the speed of flushing media should be higher than the sedimentation speed of cuttings in the flushing media. Tests have shown that the cuttings with a thickness of 0.5 to 5 mm can reduce boring performance by 40% [10]. However, mechanical parameters of different strata, including drilling pressure, rotational speed, and torque, could cause deviation of the pilot hole. The study of the influences of geological/geotechnical parameters on the excavated rock and interaction of drill string with rock mass are the vital subjects on controlling of the pilot hole deviation [11].

It is possible to find some cases in the literature about the failure of raise boring operation that resulted due to the incorrect selection or performance estimation of the RBMs. Stacey and Harte [12] mentioned some failures during raise boring operation in the gold mines of South Africa. They stated that all cases of failure were preceded by considerable machine vibration and intermittent high torque demand that was particularly noticeable when starting up after the relaxation of the drill string for removal of a drill pipe. James [13] reported another failure of raise boring operation that related to the operational parameters. The catastrophic failure of the raise boring machine occurred during the reaming of a 3.66 m diameter by 266 m long hole at a dip angle of 88° to the horizontal that the failure was due to the fracture of the 32 drive head bolts.

4. Case studies from Turkey

Recently in Turkey, raise boring machines are used for shaft excavation in mines and infrastructure projects. Due to advantage of these machines, it is believed that the RBMs will be used extensively in the mining industry in Turkey especially in the metal mines and tunnel-construction sectors. Two project sites were visited to determine the geological conditions, specifications and operational / performance parameters of the RBMs. These projects were Yusufeli Dam-HEPP, and Balya lead-zinc mine. The details of each project are mentioned below.

4.1. Yusufeli Dam and HEPP

The Yusufeli Dam is an arch dam under construction on the Coruh River, near Yusufeli in Artvin Province, Turkey. The rockfill dam has a height of 270 meters from the foundation and the total capacity of reservoir is about 2.2 billion cubic meters. About 1.8 billion kWh of electricity will be generated annually in the plant, which has an installed capacity of 540 MW [14]. Yusufeli Dam and Hydroelectric Power Plant that will be the biggest dam to be constructed in Coruh Basin, and will be the world’s third highest double curvature concrete arch dam. Figure 2 shows the general view of Yusufeli Dam area. The construction of this project was awarded to Limak-Cengiz-Kolin (LCK) joint venture by the State Hydraulic Works (DSI) of Turkey.

The geological map of the dam site is given in Figure 3. Basement rocks of the Yusufeli Dam site are Ikizdere magmatic rocks. This magmatic formation is extremely durable and includes jointed granodiorite, granite and diabase. Alluvium is found along the riverbed.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Thrust capacity</td>
<td>4000 kN</td>
</tr>
<tr>
<td>Reaming torque</td>
<td>160 kNm</td>
</tr>
<tr>
<td>Break out torque</td>
<td>300 kNm</td>
</tr>
<tr>
<td>Power demand</td>
<td>400 kVA</td>
</tr>
<tr>
<td>Rotational speed of pilot drill</td>
<td>0-60 rpm</td>
</tr>
<tr>
<td>Rotational speed of reamerhead</td>
<td>0-21 rpm</td>
</tr>
<tr>
<td>Weight (without crawler)</td>
<td>16500 kg</td>
</tr>
</tbody>
</table>

Figure 2. General view of Yusufeli Dam project site.

Figure 3. Geological map of the dam site in Yusufeli Dam and HEPP project.

4.1.1. RBM selection and performance

In this project, dam shutter control and power tunnel were excavated with a RBM. The main reason for use of a RBM instead of drill and blast method was the problem related to transportation of excavated material from the bottom of the hole. As a subcontractor, Sargin Construction and Machinery Industry excavated these shafts with a RBM manufactured by Sandvik (Rhino 1088 DC). The main specifications of the RBM are summarized in Table 1. By use of this machine, the shutter control and power tunnel were excavated in 2.44 m diameter, then these shafts will be enlarged with drill and blast method to the final diameters of 14.0 and 10.8 m, respectively. Figure 4 shows enlargement work in the dam shutter control and power tunnel.
In the dam shutter control, machine commenced excavation of the pilot hole on 28 December 2015 and completed on 3 January 2016. Then, the reaming operation started on 6 January 2016 and enlargement of the 63.50 m length shaft finished on 18 January 2016. However, in power tunnel, the RBM excavated 130 m of diabase and granodiorite rocks between the depth of 637 m (surface level) and 507 m (underground level). The machine commenced excavation of the pilot hole on 31 January 2016 and completed on 13 February 2016. In addition, the reaming operation started on 15 February and enlargement of the 130 m length shaft finished on 22 March 2016. The average daily advance rates in pilot drilling and reaming operation for both shafts are given in Table 2. For excavation of both shafts, one operator, one chief and two workers were used for pilot drilling and reaming operations. Daily working schedule was one shift of 12 hours per day for the pilot drilling and reaming operations.

Table 2 – Average daily advance rates in Yusufeli Dam and HEPP project.

<table>
<thead>
<tr>
<th>Shaft</th>
<th>Pilot drilling (m/day)</th>
<th>Reaming (m/day)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dam shutter control</td>
<td>10.58</td>
<td>5.77</td>
</tr>
<tr>
<td>Power tunnel</td>
<td>8.69</td>
<td>3.83</td>
</tr>
</tbody>
</table>

The RBM performance data such as torque, pulling force and rotational speed were recorded during the pilot drilling and reaming. The average values of recorded performance parameters of the RBM are summarized in Table 3 for both pilot drilling and reaming stages in two shafts. In Table 3, penetration rate was recorded in the field for each rod. Net penetration was estimated by dividing the measured field penetration rate by the measured rotational speed. Field specific energy ($SE_{\text{field}}$) is calculated using Equation (1) with the aid of data obtained from data acquisition system of the RBM.

$$SE_{\text{field}} = \frac{2 \times \pi \times N \times T}{NPR}$$

Where $N$ is the rotational speed of the reamerhead (rpm) and $T$ is the RBM torque in (kNm). The part $(2\pi NT)$ of the Equation (1) is the power spent during excavation for a given torque and rotational speed. NPR is the net production rate (m$^3$/h). Net production rate is calculated by multiplying the shaft cross-section area (m$^2$) and penetration rate (m/h).

Table 3 – Average values of the measured RBM performance parameters in pilot drilling and reaming in Yusufeli Dam and HEPP project.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Dam shutter control</th>
<th>Power tunnel</th>
</tr>
</thead>
<tbody>
<tr>
<td>Penetration rate (m/h)</td>
<td>1.55</td>
<td>1.25</td>
</tr>
<tr>
<td>pen (mm/rev)</td>
<td>1.46</td>
<td>1.17</td>
</tr>
<tr>
<td>Rotational speed (rpm)</td>
<td>17.81</td>
<td>11.94</td>
</tr>
<tr>
<td>Push / pull force (kN)</td>
<td>177</td>
<td>191</td>
</tr>
<tr>
<td>Torque (kNm)</td>
<td>5.70</td>
<td>147.15</td>
</tr>
<tr>
<td>Power (kW)</td>
<td>10.89</td>
<td>15.12</td>
</tr>
<tr>
<td>Specific Energy (kWh/m$^3$)</td>
<td>98.12</td>
<td>147.15</td>
</tr>
</tbody>
</table>


In the dam shutter control excavation, the last rod (42$^{\text{th}}$ rod) was pulled out after 48 hour operation; and in the power tunnel the last rod (86$^{\text{th}}$ rod) was pulled out after 23 hour operation; both values indicated that the reamerhead had to be located well in the lower level to obtain effective contact with the bottom area of the shaft.

Due to availability of access to the upper and lower levels of the shaft, geological / geotechnical conditions and the dimensions of the operating site, using a RBM was the best option for shaft excavation in this project. Based on the laboratory studies of the obtained rock samples from the nearest borehole to the proposed raise boring site, the uniaxial compressive strength varied between 12 and 116 MPa. However, the RBM mostly excavated medium to hard rock formations and due to this, multi-row carbide cutters were used on the reamerhead. As seen in Table 3, the penetration rate in the power tunnel was less than in that of the dam shutter control. In the power tunnel raise boring operation, the majority of the excavated rocks were the fractured and faulted diabase and competent granodiorite with low weathering and joint frequency. In addition, operation delays (especially power supply) were the main factors that caused the low
performance in some days. In this project, drill and blast method was used to excavate the right and left coast of the dam. During the blasting operation, the site had to be leaved and after the blasting operation, it could be possible to continue RBM operation. This loose of time caused the low performance in excavation of the shafts. Figure 5 shows the time distribution of different activities in excavation of the power tunnel. As seen, the machine utilization time for reaming involved 62% of all activities. However, the main downtime was the delays due to electric maintenance (5%) and delays due to blasting operation (4%). As a conclusion, geological / geotechnical parameters and operational parameters were most dominated factors on the performance of the RBM in the Yusufeli Dam project.

4.2. Balya lead-zinc mine

The Balya lead-zinc underground mine located in the 50 km northwest of Balikesir city in western Turkey. Figure 6 shows the general view of Balya mine. This mine was first mined by a French company at the end of 19th century. Eczacibasi Esan industrial group has been operating the Balya mine since 2009. The ore deposit generally formed in contact area of limestone-dacite and limestone. Figure 7 shows geological map of Balya mine and surroundings. The sublevel stoping method is used for extraction. An inclined ramp with a slope of 8.5% is used as the mine main gallery to minimize the transportation distance from the ore production area to the main gallery. In addition, the main gallery has been used to transportation of the workers, waste and underground mine machines.

Figure 6. General view of Balya lead-zinc mine.

Table 4 – Minimum, average and maximum daily advance rates in Balya mine (m/day)

<table>
<thead>
<tr>
<th>Value</th>
<th>Pilot drilling</th>
<th>Reaming</th>
</tr>
</thead>
<tbody>
<tr>
<td>Minimum</td>
<td>2.61</td>
<td>4.56</td>
</tr>
<tr>
<td>Average</td>
<td>12.13</td>
<td>8.77</td>
</tr>
<tr>
<td>Maximum</td>
<td>16.72</td>
<td>15.20</td>
</tr>
</tbody>
</table>
The RBM performance data were recorded during the pilot drilling and reaming. The average values of recorded performance parameters of the RBM are summarized in Table 5 for both pilot drilling and reaming stages.

### Table 5 – Average values of the measured RBM performance parameters in pilot drilling and reaming in Balya mine.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Pilot drilling</th>
<th>Reaming</th>
</tr>
</thead>
<tbody>
<tr>
<td>Penetration rate (m/h)</td>
<td>1.89</td>
<td>1.05</td>
</tr>
<tr>
<td>Penetration (mm/rev)</td>
<td>1.71</td>
<td>4.25</td>
</tr>
<tr>
<td>Rotational speed (rpm)</td>
<td>19.0</td>
<td>4.4</td>
</tr>
<tr>
<td>Push / pull force (kN)</td>
<td>138</td>
<td>1116</td>
</tr>
<tr>
<td>Torque (kNm)</td>
<td>5</td>
<td>68</td>
</tr>
<tr>
<td>Power (kW)</td>
<td>9.75</td>
<td>33.11</td>
</tr>
<tr>
<td>Specific Energy (kWh/m³)</td>
<td>74.11</td>
<td>8.76</td>
</tr>
</tbody>
</table>

As seen in Table 5, the penetration rates in both pilot drilling and reaming are higher than that of the penetration rate in the Yusufeli Dam project. One of the main reasons for this difference is related to the excavated rock formation. In the Balya raise bore project, the main excavated rocks were limestone with average 43.6 MPa uniaxial compressive strength and dacite with average 72.0 MPa uniaxial compressive strength. However, andesites were also encountered in some parts of the shaft (with average 114.1 MPa uniaxial compressive strength). These values indicated that the excavation of this shaft was easier than excavation of the Yusufeli shafts. However, in some places the presence of unfavorable ground was one of the delay source that reduced the overall performance. In addition, the site preparation (operational parameters) caused more delays in this project. It is well known that the proper site arrangement eliminates delays and increases the efficiency. Moreover, it was the first time that the operator used the machine in field. These two parameters could be main reasons for a low performance especially in pilot drilling.

### 5. Conclusions

Raise boring is an attractive method of constructing shafts, being safe, fast and comparatively cheap. The optimum raise boring system is designed or selected based on drilling / excavation performance analysis. In addition, the project owner and contractor need some general information about raise boring system such as electric power demand, the dimensions of the machine, bailing system (air or water), transportation of the equipment (crawler, sled) and etc. The parameters affecting selection and performance of RBMs could be classified into three categories as mechanical (machine related), geological / geotechnical and operational parameters.

This study aimed to itemize the required parameters for the design and proper selection of RBMs, as well as performance estimation. Two different project sites using the same RBM in Turkey were visited to collect general field and machine data. Results of the field studies are summarized as follow:

- In the Yusufeli Dam project, penetration rate of reaming were 0.66 and 0.46 m/h for dam shutter control and power tunnel, respectively. However, this value in the Balya mine was 1.05 m/h.
- In the Yusufeli project, RBM mostly excavated medium to hard rock formations; however, in the Balya mine RBM mostly excavated soft to medium strength rock formation.
- Operation delays (especially power supply) were the main factors that caused the low performance in the Yusufeli Dam project. In the Balya mine project, the site preparation and inexperienced personnel were the two main parameters causing low performance of raise boring operation.

As a conclusion remark, it is important to know that the rock test records and field performance data are the main parts to optimum select and performance estimation of RBMs. However, the difficulty involved in the taking representative samples of the excavated site may seriously limit the precision of performance estimation. In addition, the issue should be raised here that performance parameters of this study should be extended for development not only performance estimation models but also for selection of raise boring machines.

### Acknowledgement

This paper summarizes some of the results of PhD research work carried out by the first author. The authors are grateful to the support of DSI (the State Water Authority of Turkey), Joint-venture Limak-Cengiz-Kolin, Eczacibasi Esan Lead-Zinc Mine and SARGIN Construction and Machinery Industry Trade Inc.; this work would be impossible without their support.

### References


Studies on the effect of high power microwave irradiation as a means of inducing damage to very hard rock to reduce the cutting resistance during mechanical excavation

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Abstract — This paper covers recent learnings from tests with the linear cutting test machine of TU Bergakademie Freiberg while cutting Neuhauser Granite specimen that have been irradiated with high power microwave radiation. The granite samples have been irradiated by MU Leoben with 24 kW at 2450 MHz with an open-ended waveguide in a chessboard-like spot pattern. Four specimen have been investigated, one block with a radiation time of 30 s, one with 45 s, one block where only 50% of the rock face was radiated with 30 s, and one uniradiated reference sample. For the cutting tests, regular point attack picks on the linear cutting test machine have been used. Tests in clearly defined surroundings, as applicable with a linear single pick cutting rig, provide the possibilities to qualify and quantify the weakening effect, which the microwave irradiation had on the rock specimen.

Clearly visible to the eye, the irradiation led to a dense network of micro cracks in an area with a radius of ca. 2.5 cm around the radiation spot. Furthermore, a network of macro cracks formed between the individual radiation spots. The effects of this radiation could be quantified through its impact on cutting forces, specific energy consumption and grain size distribution of the cut material.

It could be shown that irradiation has a significant effect on the efficiency as well as resulting cutting forces during cutting. The cutting force could be lowered by up to 29% comparing the 45 s block to the uniradiated one.

Keywords: mechanical rock excavation, rock cutting, cutting forces, rock weakening, microwave radiation

1. Introduction

Mechanical excavation technologies are common and widespread. They show a number of advantages over drilling and blasting technologies. Major amongst them is the continuous excavation instead of the blasting cycle. Extraction and loading take place simultaneously, haulage can also be conducted while excavating. Furthermore, the surrounding rock mass is affected less, increasing safety as well as rendering scaling redundant. Blasting gases do not occur and the influence by blasting noise to adjacent communities does not apply. Furthermore, the excavated material appears in a relative narrow particle size distribution compared to the one generated by drilling and blasting. Possible economic advantages in this regard are at hand – neither does the material be pre-sized, nor do boulders affect the haulage chain’s effectivity. However, the rock strength and abrasivity are limiting an economic application of cutting technologies [1–3].

The aim of numerous research and innovation activities therefore is to overcome these barriers and enable the technology to work efficiently also in tough conditions. These activities can be grouped in three main groups (Figure 1):

a) Optimizing the cutting parameters; spacings, cutting speed, angles but also the very direction of cutting as in undercutting technologies
b) Cutting materials; introducing cutting materials other than tungsten carbide, e.g. polycrystalline diamond or silicon carbide composites or even pulsed water jets
c) Weakening or altering the rock mass, inducing cracks and by such reducing the cutting resistance of the rock face

This research work follows the path of point c) – the use of microwave radiation being an advanced mean with a similar aim as the setting of fire that was applied by our forefathers in excavation.
2. Fundamentals of Microwave radiation

The treatment with microwaves causes the radiated rock mass to absorb a portion of the electromagnetic energy and to converse it into heat energy. In comparison to most other heating methods like convectional or laser driven heating, the process is not limited to the surface but driven by a continuous absorption within the material itself.

Due to the expansion of the radiated zones, they expand and this results in the induction of stress within the rock mass. Such stress can exceed the strength of the rock, leading to the occurrence and propagation of cracks.

The ability to transfer or absorb microwaves in a dielectric material can be described by the complex dielectric permittivity \( \varepsilon \) (Formula 2.1):

\[
\varepsilon = \varepsilon_r - i\varepsilon_i = \varepsilon_0 (\kappa_r - i\kappa_i)
\] (2.1)

Whereas \( \varepsilon_0 \) defines the permittivity of vacuum, \( \kappa_r \) the real part of the relative permittivity and \( \kappa_i \) the imaginary part. The absorption of microwave energy within the material is mainly governed by \( \kappa_i \). The relative permittivity is to a high extend influenced by a number of parameters, mainly the frequency of the electromagnetic field and temperature of the radiated material, to some extent by the relative orientation of crystal structures towards the field. Depending on the individual parameters of the rock forming minerals, typical values for hard rocks range from \( 10^{-3} \) – 50 for \( \kappa_i \) and 2 – 10 for \( \kappa_r \), based on various parameters (rock type, mineral distribution, microwave frequency, temperature, water content, …) [5].

The individual minerals that appear in various distributions, show individual values for different \( \kappa_i \) values as shown in Table 1.

<table>
<thead>
<tr>
<th>Mineral</th>
<th>( \kappa_i )</th>
</tr>
</thead>
<tbody>
<tr>
<td>Plagioclase</td>
<td>0.004 - 0.32</td>
</tr>
<tr>
<td>Pyroxene</td>
<td>1.62</td>
</tr>
<tr>
<td>Ilmenite</td>
<td>32.58</td>
</tr>
</tbody>
</table>

Hereby the minerals plagioclase, pyroxene and ilmenite can be considered to be higher absorbing, whereas quartz, orthoclase and muscovite are rather poorly absorbing.

This also indicates that alongside the general heating stresses, thermal stresses between individual mineral grains can be expected, which could already be proven experimentally for comminution purposes [6, 13] as well as on a simulation basis for a massive granite model [10].

3. Material and Methods

3.1. Rock specimen

For the tests, four granite sample blocks from the kind of “Neuhauser Granite” were used. The blocks were prepared with the dimensions of 50 x 50 x 30 cm³. The granite is rather uniform and consists of 27% quartz, 53% feldspar and 20% micas. The uniaxial compression strength was measured with 198 – 202.7 MPa. The Cerchar-Index was measured to be 4.2 [14].

3.2. Microwave irradiation

The microwave irradiation was conducted by Montanuniversität Leoben, Austria, in the testing facilities of Sandvik in Zeltweg. A 24 kW industrial scale, modified microwave using a frequency of 2450 MHz was applied. The radiation was guided to the specimen with an open end rectangular waveguide. By doing so, the available power could be focused to a spot-like area of ca. 5 cm in diameter.

From the four blocks, one was left untreated (referred to as B0) to serve as a reference block, two were fully radiated in a chessboard like pattern with a spacing between the radiation spots of 10 cm and 10 cm between the spot lines. One of this two was irradiated with 30 s radiation time for each spot (B30), the other for 45 s (B45). To visualize the crack distribution, a regular penetration fluid was used as can be seen in Figure 2. Visible to the eye, two different kinds of cracking occur:

a) Microcracking on the radiation spot
b) Radial macrocracks in a sun ray like pattern that can create a dense cracking network

In B45, the cracks were bigger and deeper (up to 10 cm) and more densely. Some cracks were also visible on the sides of the block.

For the last block, only half of the face was irradiated with 45 s (B1/2). The radiation took place in 3 lines. Also the distance between the lines was chosen to be varied. The spacing between the first line to the middle was 10 cm, the spacing between the second and the last line was reduced to 5 cm.
During the irradiation, the surface temperature of the specimen was measured with an infrared measuring gun. B30 showed surface temperatures of up to 300 °C; B45 and B1/2 of up to 500°C.

3.3. Cutting tests

Cutting tests were carried out using the linear cutting test rig HSX 1000-50. The rig was developed for the determination of cutting forces and volumes using various cutting tools (Figure 3) due to a modular cutter head design.

![Figure 3: Linear cutting test rig HSX 1000-50 of TU Bergakademie Freiberg](image)

The machine regularly operates under peak cutting forces of up to 50 kN. Without exceeding that limit, it was possible to cut with a cutting depth of 4 mm. Regular point attack picks have been used for these tests. Two different spacings were chosen: 8 mm and 12 mm. The cutting speed was chosen to be 0.1 m/s to minimize frictional heating between pick and specimen and the increased wear that goes alongside this [15].

While cutting, the three vector components $F_x$, $F_y$, $F_z$ of the resulting cutting force $F_i$ were measured. The measuring frequency was 10 kHz. $F_x$ hereby is the force in cutting direction, $F_z$ the normal force and $F_y$ the side force (Figure 4).

For the measurement of the outbroken volumes, a laser based surface scanner was used. By subtracting the surface scans of the face before and after cutting, the volume difference can be calculated.

For evaluation of the results, a calculation of mean forces and variance coefficient for $F_x$, $F_y$, $F_z$, furthermore a peak analysis for $F_x$, $F_y$, $F_z$ were conducted.

The calculation of the specific energy consumption $E_{sp}$ follows Formula 3.1:

$$E_{sp} = \frac{\overline{F_x} \cdot l_{cut} \cdot n_{cut}}{V_{cut}}$$

(3.1)

With $\overline{F_x}$ being the mean cutting force for the calculated area, $l_{cut}$ the total way that the pick moved in cutting direction for one cut, $n_{cut}$ the number of cuts used to excavate the volume and $V_{cut}$ the total volume that was outbroken. For better comparison to other results within the field of mechanical excavation, the results are shown in kWh/m³.

A sieving analysis was additionally conducted to confirm the other findings as a higher $E_{sp}$ usually correlates with a grain size distribution that shows higher amounts of fines.

![Figure 4: Directional definition of the vectorial dimensions of the cutting force in relation to the cutting direction.](image)

In cutting, B45 failed while being cut although the blocks were framed with steel reinforced concrete (Figure 5). Note the additional outbreak on major cracks, marked (a) and the still visible purple penetration fluid (b); the block failed as would be expected alongside a major crack.

![Figure 5: destroyed B45 after cutting the 5th layer (photo with highly increased contrast to increase visibility of penetration fluids)](image)
• The average increase of the forces between each layer is ~ 3%, this can be accounted to the wear of the picks and the reduced microwave influence
• The cutting forces of the tests with 12 mm spacing have in average been 13 % higher than with 8 mm
• Between B45 and B0 a clear reduction of the cutting forces could be monitored; for $F_x$ the reduction is 22.5 % in mean (16.5-29 %); for $F_z$ the reduction lies at 36.9 % (29.4-43.1 %); (see Figure 6)
• Between B0 and B30 no relevant change of the cutting forces could be observed

For the half treated block the spatial analysis of the cutting force distribution illustrates the effect of the microwave irradiation in Figure 8. The image has been created using a moving average algorithm based on a grid set with a spacing of 1 mm. The search distance to calculate the value of each point was set to 25 mm. Between the radiation points, $F_x$ drops as low as 5.1 kN where the minimum in the unirradiated zone lies at 6.4 kN. In average, $F_x$ was lower 10% in the radiated area (7.06 kN untreated side vs. 6.39 kN on treated side) [16]

Figure 8: Cutting force map of half radiated block [16]

Figure 9 illustrates the force distribution for the two fully radiated blocks and the unirradiated block. Visible are both the higher force levels on the sides that have been cut with a spacing of 12 mm, as well as the “force valleys” that are caused by the larger cracks and the overbreak that is also visible in Figure 5.

Figure 9: Cutting force maps of B0, B30 and B45, 2nd layer; red dots mark radiation points

Verifying the results, the sieving analysis in Figure 10 shows a similar picture. B45 presents a particle size distribution with a higher amount of coarse grains whereas B0 and B30 present similar sieving curves.

Figure 10: Sieving analysis for B0, B30 and B45
5. Conclusions

It was shown that high power microwave radiation can be used to induce thermostress crack patterns into massive granite. Directly on the radiation spots an area of small cracks is visible to the eye. Furthermore, radial macro cracks that overlap could be observed. They form a network between the radiation spots that effectively weakens the whole radiated area.

However this seems to show its effect only after a “critical treatment duration” where the crack network propagated dense enough and the cracks are deep enough to have an influence on the structural integrity of the rock. This is indicated by the fact that the weakening of B30 was not measurable on a block-wide scale.

For B45 however, the results show clearly that the irradiation has a considerable effect as the reduction of the cutting forces by 22.5% and the lowering of the specific energy consumption by 19.4% imply.

The presented results show that a pre-weakening of the rock mass with the aim of increasing the cuttability has a lowering effect on the cutting forces as well as on the specific energy consumption of the excavation. As such, in an industrial environment, a pre-weakening could extend the limitations during the excavation of hard rock or increase the net cut rate.

Following this thought into future, it can be stated that pre-weakening of the rock mass could allow lighter machine designs, increase the advance rate or bring a rock to a condition where it is cuttable economically in the first place - high power microwave radiation could be one of the methods to achieve this.

However research in this field is at a very early stage, further fields of work would be to deepen the understanding of what happens on a microscopic scale within the rock and further a more precise understanding of the “critical treatment” with reference to treatment times and treatment power as well as pattern geometry.

Acknowledgement

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6. References


Ranking of influencing factors on rock mass fragmentation in block caving

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b University of Alberta, Department of Civil & Environmental Engineering, Edmonton, Canada

Abstract—Assessment of rock mass fragmentation is one of the most significant engineering design issues in geomechanics and mining industry. There are several influencing factors on the fragmentation which can be grouped into geometrical, geomechanical, environmental and operational. These factors need to be identified and quantified for fragmentation assessment. The aim of this study is to identify and rank the influencing factors on rock mass fragmentation using Rock Engineering System (RES). In first step, the influencing factors are identified. Thereafter, the interaction matrix based on the RES is used to study the effect of influencing factors on rock mass fragmentation. The interaction matrix is established and coded by Expert Semi Quantitative (ESQ) approach. The interaction matrix analyzes the interrelationship between factors. The results of the paper show that the most dominant factors are fracture frequency and stress. The most subordinate factors are stress, drawpoint geometry and Hydraulic radius respectively. The most and least interactive factors in system are stress and discontinuity filling. Generally speaking, geomechanical factors are more dominant on fragmentation, in comparison with other group of factors.

Keywords: Fragmentation, Rock mass, Rock engineering system, Block caving

1. Introduction

The block caving method has several advantages such as high production, lower costs, and high productivity; furthermore, the process can be automatized [1]. However, the use of block caving at increasing depths and scales has introduced a number of serious technological and environmental challenges [2, 3].

Block caving fragmentation takes place when the rock mass breaks into smaller fragments after it has been undercut. This primary fragmentation further reduces block size through secondary fragmentation within the draw column [4, 5]. Poor fragmentation may have serious consequences on caving operation. If the fragmentation is oversized, large blocks will severely impact operations by impeding material handling and causing costly delays to clear hang-ups at the drawpoints. If fragmentation is too fine, narrower draw columns might develop, limiting an interactive broken ore flow between columns [6]. Regarding production losses due to hang-ups, Dessureault [7], show a notable difference between the design draw rate and the historical operational draw rate (0.60 and 0.42 t/m2/day, respectively) at El teniente mine (Chile). Considerable amount of fines, in the presence of water, can potentially be saturated, triggering mud rush events [4, 8].

In block caving, the rock mass fragmentation process is decisive in the design and success of the operation [1], which is influenced by different factors. However, more knowledge and study are required in the area of rock mass fragmentation and influencing factors on fragmentation. Therefore, it is of great importance to identify the most influencing factors and their interactions on the fragmentation process and also to identify, which factors (or interactions) are beneficial (and hence should be enhanced) and, conversely, which ones are detrimental (and hence should be minimized).

Several researchers attempted to assess fragmentation process in block caving operations. The Block Caving Fragmentation (BCF) software developed by SRK Consulting [9], represents one of the few established tools, to assess fragmentation and hang-up potential. Pierce [10], proposed a more rigorous methodology to evaluate secondary fragmentation based on the gravity flow simulator REBOP [11], and calibrated using empirical relationships [12]. Most of these techniques consider a limited number of affecting factors.

Rock engineering system (RES) is one of the most powerful system approach which not only can examine the problem in its totality with a complete list of the components, but also can take the interactions between the factors into account. RES first introduced by Hudson [13], to deal with complex engineering problems. The method combines adaptability, comprehensiveness, repeatability, efficiency and effectiveness [14], in order to study the interrelationship between various factors in an engineering project, (for more application of RES see [10, 15-28]).

The aim of this study is to identify and rank the influencing factors on rock mass fragmentation using Rock Engineering System (RES). In first step, the influencing factors need to be identified. Thereafter, the interaction matrix can be used to study the effect of influencing factors on rock mass fragmentation. The interaction matrix analyzes the interrelationship between the influencing factors on rock mass fragmentation.
The rest of this paper is organized as follow. In section 2, the concept of RES is presented. Then in section 3, influencing factors on rock fragmentation are identified and discussed. The application of RES for ranking of influencing factors on fragmentation is presented in section 4. Section 5 is discussion and conclusion.

2. Rock Engineering Systems (RES)
In order to design a structure to be built on or in a rock mass for mining or civil purposes, it is necessary to consider all relevant influencing factors and their interactions [14]. RES is a powerful system approach to study the effect of influencing factors on the performance of an engineering system. The first step of the method is to establish interaction matrix and coding of the matrix. The second step is to calculate and plot the cause and effect of influencing factors on the performance of the engineering system (for example in our case is rock mass fragmentation).

2.1. Establishing of Interaction Matrices
A systematic method for thinking about all the interactions is to list them in a matrix. The principal factors considered relevant to the problem are listed along the leading diagonal of a square matrix (top left to bottom right) and the interactions between pairs of principal factors form the off-diagonal terms. Figure 1, shows the basic concept of interaction matrix, with two influencing factors A and B, which are located in the top left and low right respectively. The top right location indicates the dominance of A on B, whereas the low left is dominance of B on A [13]. Then values should be assigned to off-diagonal terms in order to quantify the interaction of influencing factors. The assignment of these values is called Coding the interaction matrix. Several coding methods have been developed for characterizing the significance of the off-diagonal cells in the interaction matrix, such as Binary method, Expert semi-quantitative (ESQ), via solutions to partial differential equations (PDE), via complete analysis of the mechanisms and etc.

Most of the parameters influencing fragmentation of rock mass in block caving and their interactions are very difficult to quantify. Therefore, the ESQ method is used in this research, which includes five level of coding from 0 to 4 (Table 1). The level of coding is based on the expert opinion. For example, the value of 4 will be assign if the expert believes there is a critical interaction between two factors. Figure 2 shows an example of a 4×4 interaction matrix with leading diagonal terms of Rock Structure, Rock Stress, Water Flow and Construction. In each of the off-diagonal terms, an example of the potential interactions is shown. With N leading diagonal terms the matrix will have N(N-1) off-diagonal mechanisms [14].

2.2. Cause-Effect Plot
The figure (3a) shows an example of an i × j interaction matrix. After coding the matrix, for the i-th factor (P), the sum of its row values is the “cause value” (C) and the sum of its column values is called the “effect value” (E). Such information can be summarized as coordinates (C, E) on a cause–effect plot, where each point in the graph represents a particular factor P. In other words, C represents the way in which P affects the rest of the system and E represents the effect that the rest of the system has on P. It is related to the factor being “dominant” (lower right region of the (C, E) plot) or to the system being “dominant” (upper left region). C and Ei can be employed to compute the “level of interactivity” of P, which is C+Ei [13].

![Figure 1: The principle of the interaction matrix [13]](image)

<table>
<thead>
<tr>
<th>Table 1 - Concept of codes in ESQ [13]</th>
</tr>
</thead>
<tbody>
<tr>
<td>Code value</td>
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<td>0</td>
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<tr>
<td>1</td>
</tr>
<tr>
<td>2</td>
</tr>
<tr>
<td>3</td>
</tr>
<tr>
<td>4</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Rock structure</th>
<th>Fractures affect the values and orientation of the stress</th>
<th>fracture network governs the secondary permeability</th>
<th>Fracture can influence the size and orientation of excavations</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fij</td>
<td>Rock Stress</td>
<td>In general, the higher the normal stress, the lower permeability</td>
<td>High rock stresses can cause construction failures</td>
</tr>
<tr>
<td>Stresses can open or close fractures, and also create them</td>
<td>σij</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Continual water flow in fractures affects their properties</td>
<td>Normal stresses reduced by water pressure</td>
<td>Water Flow</td>
<td>Grouting and drainage may be required during construction</td>
</tr>
<tr>
<td>Kij</td>
<td>Water Flow</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Blasting can damage old fractures and create new fractures</td>
<td>In the vicinity of excavations the principal stresses are reduced</td>
<td>An excavation will always become a sink for the water flow</td>
<td></td>
</tr>
<tr>
<td>Cij</td>
<td>Construction</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

![Figure 1: 4×4 interaction matrix for mechanical factors [14]](image)
The cause-effect graph in figure (3b) has been generalized to \( N \) influencing factors. For using RES, it is very important to note carefully, the location of each point which represents the main factors of the system. The cause-effect plot is helpful in understanding the behavior of each factor individually as well as studying the whole system. For example, the points tend to distribute perpendicularly to the \( C=E \) diagonal, this shows low level of interactivity between factors, whereas a high interactivity will result in the points distributed along the main diagonal [27].

![Figure 3: Cause and Effect](image)

3. Influencing factors on rock mass fragmentation

According to the literature, there are several influencing factors on rock mass fragmentation. For the purpose of this study, those factors are divided into four category including geomechanical, environmental, geometrical and operational factors.

3.1. Geomechanical Factors

**Uniaxial Compressional Strength (UCS):** Rocks with low level of UCS have higher caveability and fragmentation. Since the UCS in the mining zones in most cases are different, it is necessary to consider different operational zones based on the value of UCS and the analysis should be made for each operational zone separately [1].

**Elastic Modulus:** Elastic modulus of rock indicates the deformability of the rock and similar to UCS, it is a key factor in caveability and fragmentation.

**Frequency of Discontinuities in Rock Mass:** Rocks having higher fractures frequency have more caveability. The recent studies indicate the sensitivity of fragmentation to the volume intensity of joints \( \left( P_{32} \right) \) and the importance of critical intensity value in which the in-situ rock is converted to rock mass, which can remove [38].

**Discontinuity Aperture:** Aperture is the distance between the two walls of discontinuities that is measured perpendicular to main plane, which can be filled with materials. Aperture can have effects on shear strength of discontinuities, water transmissivity and therefore can have effect on fragmentation of the rock mass.

**Discontinuity Persistence:** Persistence is the areal extent or size of a discontinuity. The persistence of joint sets controls large scale sliding or ‘down-stepping’ failure of slope, dam foundation and tunnel excavation. It has the highest effect on the rock mass strength among other specification of the discontinuity [39].

**Roughness of Discontinuity Planes:** Joint surface roughness is a measure of the inherent surface unevenness and waviness of the discontinuity relative to its mean plane. Discontinuities having lower shear strength are suitable for caving because they have more intensity for opening and caving [40].

**Discontinuity Filling:** The wide range of physical behaviour of discontinuities depends on the properties of the filling material. In general, filling affects the shear strength, deformability and permeability of the discontinuities. Filling is influencing on rock mass caveability because this factor has a significant role on rock mass strength [40].
Weathering of Discontinuity Plane: Weathering and alteration of joint plane can decrease the roughness of joint plane and therefore can decrease the shear strength of discontinuity which can increase the potential movement of blocks on each other and increase caving and fragmentation.

3.2. Environmental Factors
The rock mass fragmentation is also affected considerably by environmental factors. The most important influencing environmental factors on fragmentation are Stress field, ground water and fine material proportion.

Stress field: The ratio of horizontal to vertical in-situ stresses has a considerable effect on the intensity of induced stresses in cave back, cave propagation and caving rate. Despite the existence of suitable structures and Geomechanical characteristics of these structures for caving of rock mass, high stresses may limit the initial caving and its propagation in the deposit. The high values of these stresses can lock the blocks and create the rock mass stability against caving propagation, if low angle discontinuities in the rock mass do not exist [41].

Ground Water: The presence of water in the area can reduce the friction between joints and increase of water pressure cause increase in the caveability. The origin of water can be from ground water resources or seasonal precipitation

Fine material proportion: The ratio of fines to medium/coarse fragmentation needs to be noted, as a high percentage of fines will cushion the larger blocks and prevents further attrition of these blocks and reduce the secondary breaking effect [3].

3.3. Geometrical Factors
Geometrical factors such as Hydraulic radius of undercut, undercut height and block height can cause variation of induced stresses in the cave back and its primary fragmentation.

Geometry of Undercut: A successful undercutting cause breaking and downward vertical movement of rock mass. A weak undercutting, leads to deformation of pillars, formation of large dimension blocks and eventually leads to lack of initial caving [42]. Narrow rectangular undercuts with respect to other undercuts can lead to forming a stable arch and halt the caving propagation compared to the undercuts with the same hydraulic radius.

Undercut height: Undercut height affects the amount of induced stresses, caving propagation, the amount of caved ore to be drawn out, time to the primary production, fragmentation and initial costs [43].

Draw column height: Secondary fragmentation occurs through downward movement of ore toward the draw point in draw column. The caving stress is likely to be significant if the height of the cave column is appreciable and there is irregular draw [3]. The blocks undergo abrasion and breakage (i.e., secondary fragmentation) which increases with draw column height.

Draw point geometry: The degree of fragmentation determines the size of the draw zone and hence the draw point spacing. It also influences the height of the draw point, the need for access for secondary breaking, the shape of the major apex, the LHD size and crushing requirements.

3.4. Operational Factors
The orientation and speeds of undercutting and draw rate from draw points are among the influencing operational factors on fragmentation [1].

Undercut direction: The shape of ore body, distribution of grade in the ore body, in-situ stresses, difference between the ore strengths in different zones, main structures and their orientations in the deposit and also the existence of previous cave zones adjacent to the block, are influencing on the selection of starting point and advance direction of the undercut. If the deposit is long and narrow in horizontal plan, the direction of undercutting will be in longitudinal direction [43]. The direction of undercutting in relation to the direction of main stresses is influencing on the intensity of boundary stresses. To reduce the surrounding stresses in the cave back undercuts are usually excavated in the direction of main stresses [43]. Advancement of caving in the direction of maximum stress can cause caving to be easier, however, only the rock masses in which support systems are installed, can bear high boundary stresses [44].

Draw rate: Control of extraction rate considerably affects caving and fragmentation behavior of ore [41, 42]. A low draw rate will result in time dependent failure of the blocks as they are subjected to the caving and arching stresses. Draw rate should not be high so that it causes to create an air gap and will increase probability of airburst. Weak control of the extraction rate, leads to leaving some caved rocks adjacent to walls. This can support walls and reduce the effect of undercut level.

Multiple draw interaction: Different draw rate from adjacent draw points is often the result of having zones of well fragmented material available, allowing for high productivity from those draw points at the expense of the draw points with coarse material. Draw control management is required not only to maximize the recovery but also to improve the fragmentation. A uniform draw over the whole mining area means little relative movement between rock blocks, so that high and low pressure areas will not set up to promote differential movement [3].

Air gap: Single blocks released from the cave back can align to form numerous block arrangements. In the case of a negligible air gap, the blocks released from the Cave back will have less chance to rotate and thus will retain their contact with adjacent blocks. This would lead to a tighter packing and smaller initial swell factor. In contrast, the presence of a sizeable air gap would facilitate a more disordered block arrangement, increasing the initial swell factor [3, 6].

Broken ore density: The broken ore density (BOD), commonly related to the swell or bulking factor, is an important factor for block caving design. It is well known that the ore column density decreases (and swell factor increases) at the draw point due to the development of a loosening zone generated by ore extraction. However, the broken ore in the
draw column also potentially experiences stress and density heterogeneities throughout, depending on the block properties (e.g., shape, aspect ratio and size distribution). This generates rounder block shapes and smaller particles, enabling different block shape configurations and finer broken ore size distributions. These smaller particles migrate downwards into the draw column increasing the BOD [3, 35].

4. Ranking of influencing factors using RES

After investigation and definition of influencing factors on Fragmentation, the Interaction Matrix (IM) is established as shown in table 2.

Twenty-one influencing factors on fragmentation and fragmentation potential are set on the main diagonal terms of the interaction matrix. Then the other cells of the matrix are coded using ESQ, in which the view of eight experts in mass caving method is expressed. Experts are selected among persons who have knowledge and experience about block caving process. Thereafter the cause and effect plot of all influencing factors have depicted as figure 4.

As can be seen in figure 4, filling has the lowest interaction in system and in-situ stresses, caveability and drawpoint geometry are the most interactive factors respectively. The fracture frequency, in-situ stresses and UCS are the most influencing (dominant) factors on system respectively while filling and air gap are the most subordinate factors in system (have minimal impact on system). A large data scatter along the main diagonal is observed, which mean a high level of interactivity between factors. Considering the column chart of figure 5, the following results have been obtained:

- In the class of geo-mechanical factors, fracture frequency and UCS have the highest effect on the fragmentation. Moreover, among the factors in this group roughness and alteration of joints are the most dominated by system.
- In the class of environmental factors, in-situ stress has the highest cause and effect, which is the most interactive factor in the system as well.
- In the class of geometric factors, the draw point geometry has the highest cause, effect and interaction intensity in the system.
- Among operational factors, the factor of draw rate has the highest cause and effect and also is the most interactive factor in this class.

<table>
<thead>
<tr>
<th>Table 2: Interaction matrix of dominating factors over caveability and fragmentation in block caving</th>
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<td>$P_1$</td>
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Ranking of the influencing factors according to their dominant, subordinate and interaction intensity are presented in table 3. It shows that the most dominant factors, affecting fragmentation are: fracture frequency, stress, UCS and persistence, respectively. The most subordinate factors are ranked as fragmentation, stress, drawpoint geometry and hydraulic radius. The most interactive parameters in the system are stress, fragmentation and caveability respectively. As noted, three out of four factors that have the highest cause on the system are among geo-mechanical factors. It means that geo-mechanical factors and specially factors related to discontinuities specifications have the highest cause on the system. Geometrical and operational factors not only have the highest effect compared to the other classes but also have the highest interaction intensity in the system.
5. Discussion and Conclusions

Ranking of influencing factors will help to identify the most important factors for improvement of the system. In order to improve the condition of the system, the first step is to consider the factors, which are more dominant on system. Subsequently one should consider the interaction intensity of each factor with the rest of system, especially those with high level of interactivity to achieve the demanded goals. The final precedence of the factors is to be done considering technological and economic aspects.

The most dominating factors on caveability and fragmentation, which can be controlled and changed by human are: fracture frequency, in-situ stresses, the geometry of draw points and undercut, draw rate and undercut direction.

The ranking of the factors based on the interaction intensity are: in-situ stress, drawpoint geometry, hydraulic radius, draw rate, undercut direction and fracture frequency, in order to improve fragmentation and caveability, some changes should be done in those factors, which have more dominancy, beside more interaction intensity in system. If the dominance and interaction intensity of factors be greater, it’s possible to make more improvement in system with smaller change in that factor.

Based on the result of ranking of influencing factors, the following suggestion can be considered for improvement of the system performance:

- The change in the draw rate and anisotropic draw from the adjacent draw points also help to control the fragmentation.
- The improvement in fragmentation related to the direction of undercutting is also doable, by advance in the direction of maximum main stress.
- The fracture frequency could be changed by using some artificial methods; like hydraulic fracturing and blasting in order to increase primary fragmentation.

The result of the paper shows that the RES is a very powerful tool for ranking of influencing factors on rock mass fragmentation. This will help improving of system performance through making a proper decision.

References


MPES2017

Session 4 – Rock Breakage & Blasting Techniques

15:30-17:00

August 29, 2017
How sustainable is Botswana’s base metal mining industry in the face of the recent closure of its only four base metal mining operations

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Abstract

Over the period from February 2015 to October, 2016, the four base metal mining operations in Botswana suspended their operations or closed due to persistent low nickel and copper prices. For instance, nickel prices over this period were in the approximate range of US$4.00/lb to US$5.00/lb which were about half of the prices seen in mid-2014 and they continue to be low. The closure of all the four base metal mines exposes the country to economic, socio-economic and environmental risk. This paper focusses on the economic, social and environmental risks that are likely to follow as a result of the suspension or closure of some of the country’s base metal mines. The paper makes conclusions regarding the long term economic sustainability of the base metal mines and projects taking into account their reported cost competitiveness in the industry.

Keywords: sustainability, mine closure, mining value added, Selebi-Phikwe, base metals

1.0 Introduction

There is no doubt as to the many benefits that large scale mining projects hold for a country and, especially, developing countries, where such projects, in addition to providing direct economic benefits in the form of mineral royalties, profit taxes, and employment and indirect benefits through economic linkages that may be up, side or downstream of the mineral project, have served as catalyst for economic development. This role is also recognized in the African Mining Vision 2050, of a “transparent, equitable and optimal exploitation of mineral resources to underpin broad-based sustainable growth and socio-economic development”[1].

The Africa Mining Vision 2050 foresees a role for the mining sector in Africa as being that of leveraging the goals of poverty reduction, industrialization, economic growth and diversification. These also align with the United Nation’s 2030 Agenda for Sustainable Development, such as, goal 1-end poverty in all its forms everywhere, goal 8 – promote sustained, inclusive and sustainable economic growth, full and productive employment, and decent work for all, goal 9 – build resilient infrastructure, promote inclusive and sustainable industrialization and foster innovation, and goal 12- ensure sustainable consumption and production patterns [2].

Sustainability in general is defined as “living to the future the option or the capacity to be as well off as we are.” [3]. Sustainable development in the mining industry has been discussed by many researchers (e.g.[4], [5],[6]. In essence, it is the integration of environmental, economic and social considerations to ensure that both the present and future generations meet their needs without compromising the needs of the latter. Since mining involves the extraction and or processing of non-renewable resources as well as a large and diverse range of stakeholders with potential to severely harm the environment, pursuing sustainability at mines entails “continuously strategizing to improve environmental protection and promote socioeconomic growth”[7].

While it is understood that economic performance, social and environmental objectives are equally important as separate pillars that drive the sustainability of a mining project [8], in practice it is the profitable mining project, and hence good economic performance, that provides the ability to fulfill the social and environmental sustainability objectives. The argument therefore is that with a long term real decline in commodity prices, technological innovation in the mining and processing of ores holds the key to sustainable economic performance and hence the whole sustainability agenda [9].

The debate about sustainability in mining projects recognizes that while it is not feasible to maintain constant mineral production over the life of mine due to factors such as declining ore grades and volumes, the benefits from mining can be utilized in a manner that takes into account the issues about intergenerational equity. This requires the existence of policies at the national level that would provide for a portion of benefits from mining to be invested in any of the human or physical capital or other forms of investments that would generate future revenue streams for the benefit of future generations [3]. At the mining project level, a sustainable mining business is about achieving the triple bottom line with regards to issues about
people, planet and profits. This means that the success of a mining project and hence the livelihoods of those in the host communities can be achieved through the balancing of the financial, social and environmental objectives of the project [10].

Sustainable development challenges in the mining industry have led to strategies to address them[11]. Initiatives to improve sustainability in the industry from the inception of sustainable development concepts in 1987 to 2007 have been tabulated [12]. The majority of the major mining companies who are members of the International Council on Mining and Metals (ICMM) use the sustainable mining business perspective or triple bottom line in their sustainability reports [13]. Some useful insights can be gained if the people or social pillar in the triple bottom line objective is unpacked to reveal both the employee and wider stakeholders. The latter includes host communities, traditional land owners, local authorities, non-governmental organizations and anybody with a link to the mining project [9]. To address the interests of the community stakeholders, mining companies should leave behind institutions and infrastructure that will support the community beyond the life cycle of the mine. This can be achieved by setting up sites with significant local economic development and interest to stakeholders, invest in institutions and infrastructure and drawing goods, materials, and services from local communities [6]

The whole issue of sustainability of a mining business is one that can be addressed along the full mining value chain. To facilitate this effort, the ICMM has come up with an 8 module toolkit that provides a menu from which the mining company, host government, community leaders and other stakeholders can identify the relevant issues and choose the appropriate module to apply in order to quantify and assess the economic, environmental and social impacts of a mining project at the community or national level in mineral dependent countries [14].

Mining companies that are members of the ICMM produce, voluntarily, sustainability reports that are prepared in accordance with the Global Reporting Initiative (GRI) and Mining and Metals Sector Supplement (MMSS). [15]. The GRI’s reporting framework consists of sector supplements and standard disclosures that guide companies on what to report and principles and guidance, and protocols on how to report it. The MMSS lists a total of 9 economic, 30 environmental and 40 social indicators. Companies have the flexibility to start with fewer indicators, a low declaration level denoted by the letter C, and to increase, overtime, their declaration levels to achieve an A+'.[14].

Sustainability spans the life cycle of a mine from exploration to mine closure and beyond. Sustainability means that, as a mine closes, the community does not become a ghost town or become severely limited with regard to socio-economic opportunities[5]. Terminal closure of a mine occurs when an ore body is exhausted. Abandonment closure is associated with inoperative mines that are under care and maintenance for several reasons. The reasons include economic, geological, technical, regulatory, changes in policy, social or community pressure, closure of downstream industry or markets and even flooding or mud inrushes [16, 17].

A closure plan is designed and implemented to promote sustainable development at mine closure and beyond. Closure plans describe the steps a mining company should take to minimize the adverse impacts of mining, and to maximize social and economic prospects after mining[18]. It is thus much more challenging to achieve sustainability for a terminal closure than an abandonment closure. Efforts have been made to account for environmental practices throughout the mine supply chain by encouraging mine suppliers to use eco-efficient and sustainable products, to recycling and to avoid the use of toxic products wherever possible[6].

Examples on the approaches to mine closure in the Southern African region are few. Two studies cited and commissioned by Coaltech examined the closure dynamics of 36 mines in Mpumalanga and KwaZulu-Natal in 2006 cited problematic social aspects of closure [18] such as confusion in the management of social risks; inappropriate training for self-employment, the failure of job creation schemes; the illegal occupation of houses; and vandalism of infrastructure and facilities. In addition, miscommunication over issues related to closure such as community expectations, ability of the company to deliver the expectations, company plans, psychological stressors on individual and groups, government policy and expectations, were some of the closure problems identified by [18] in South African mines.

A study that explored the management of social and environmental liabilities after mine closure using the Zambian copperbelt as an example was done by [19]. The study confirmed that mining is a major contributor to the national revenues, it also has significant negative impact on a country’s social and environmental status regardless of the cause of closure, and that both social and environmental impacts require proper and sustainable management approaches. In addition, the study suggested that consultations should not only begin at the inception of the project, but also that the approaches to managing these impacts should include consultations among the community, government and the operating company as key stakeholders. Furthermore, due to poverty, high illiteracy levels and lack of access to information, mining communities in developing countries were more of spectators and at the mercy of decisions made by the operating company and or the government. Finally, although policy and legal frameworks for closure were in place in most developing countries, implementation of these regulatory instruments were often not carried out due to financial and human resource constraints as well as the operational constraints brought about by the development agreements signed between the mining operator and the government.
Selebi-Phikwe mine contributed to local employment generation and attracted other economic activities such as industries, commercial businesses, social services and small-scale employment activities that expanded employment opportunities for the people in the area. The growth of the private sector since the establishment of Selebi-Phikwe as a mining town has been remarkable. For instance, infrastructure such as roads, telecommunications, rail and airstrip have improved remarkably while manufacturing and engineering industries, commercial businesses such as banks, restaurants and wholesale and retail shops in the town have increased.

The study cited the emissions of sulphur-dioxide gases into the atmosphere from the processing activities of the mine which led to the destruction of vegetation and contamination of soil in the direction of the prevailing winds as the major environmental problem in Selebi-Phikwe and its immediate environs. The air pollution rendered a vast area of residential land unsuitable for settlement resulting in acute land scarcity. The other risk cited was that of soil and water pollution from acid mine drainage from the tailings and slag dumps. In addition, the discharge of wastewater from the mine in most cases polluted recipient river systems with heavy metals, suspended and dissolved solids. However, the study cites that the mine outlined a number of environmental policy objectives to address these impacts.

The social impacts of copper-nickel mining in Botswana from exploration, through mine development and finally to mine closure for three different mining companies in three different socio-cultural and ethnic regions of Botswana were the subject of a recent study by [21]. The companies in the study were Mowana mine project operated by African Copper in the rural areas of Dukwi and Mosetse, which represented a mine at the stage of mine development phase, Tati-Nickel Mining Company’s Phoenix mine in the peri-urban areas of Matselagabedi and Matsiloje areas, which represented a mine in the production and expansion stages, and finally, the BCL mine in Selebi-Phikwe representing mine in its closure stage.

The impacts ranged from economic benefits such as foreign exchange earnings, employment, the optimal use of available mineral resources and the possible development of Dukwi and Mosetse villages, to deleterious social impacts in the form conflict between the mine and the local people.

The main impacts cited in the study were relocation of people during mine production and expansion. The Tati-Nickel Mining Company had elaborate policies of good corporate governance and best practices that promoted sustainable development. For instance, the mine had a social responsibility programme designed to benefit communities within a fifty kilometer radius of the mine.

1.1 Minerals sector contribution to the economy

The major diamond and base metals mines have been in operation for just over four decades and as expected, the production level will continue to decline as these mines get deeper and the ore diminishes. Figure 1 depicts diamond and copper-nickel matte production for the period 2006 to 2015. Production at the four Debswana Diamond Company’s mines peaked in 2006 at some 34.3 million carats. The global economic recession in the last quarter of 2008 that lasted through 2009 affected the diamond industry and led to an approximately 50% reduction in diamond production from the peak production (34.3 million carats) in 2006 to 17.7 million carats in 2009. Since 2009, Debswana’s diamond production has remained in the range from 20.5 million to 24.7 million carats, which translates into 60% to 72% of the peak production level of 2006. In 2015, diamond production was affected again by low diamond prices and this led to one of the Debswana mines, Damtshaa, near Orapa mine, being placed under care and maintenance. Two diamond mines operated by junior miners, Lucara Diamonds’ Boteti Diamond mine near Debswana’s Lethakane mine and Gem Diamond’s Ghabhoo mine at Gope, were commissioned in 2012 and 2014 respectively. The latter was placed under care and maintenance in 2017 due to low profitability. Total carat production from these two mines have been less than 0.5 M carats per annum. As for the base metals, there has been a drastic decline in the production of copper-nickel matte over the period 2006-2015 from about 64,000 tonnes in 2006 to a low of about 30,000 tonnes in both 2014 and 2015. The country also produces soda ash and salt, and gold for the export market. These contribute a mere fraction of less than 4% in export revenues. Coal is mined for power generation at the government owned Morupule Power Station.
On average diamonds and copper-nickel matte contributed 75% and 14% to export revenues respectively over the period 2006 to 2015. As diamonds are the main economic contributor, the decline in both the percent contribution to GDP and government mineral revenues from 32% and 48% respectively in 2006, when diamond production was at its peak, to an average of 19% and 30% for the period 2010-2015 is largely attributed to the decline in diamond production to about two thirds of the peak level in 2006. Figure 2 depicts the decline in the mineral contribution to both GDP and government revenue.

Mining is still the single largest contributor to GDP and regarding the issue of economic diversification, the fact that in recent times its contribution to GDP has been about 19% demonstrates that the economy does not rely on only one economic activity and is therefore diversified. Figure 2 also demonstrates that there is still a high level of concentration in exports, which are dominated by diamonds.

Mining is capital intensive and therefore the contribution to employment has remained relatively low at about 3% of formal employment.

1.2 Base metal mines and projects

The major base metal mines up until 2016 were the 100% government owned BCL limited copper, nickel and cobalt mining and smelting complex at Selebi Phikwe and the Selkirk and Phoenix mines some 45 km east of Francistown. The copper and silver projects owned by junior mining companies during the years 2015 and 2016 were Discovery Metal’s Boseto copper project at Toteng, near Maun, African Copper’s Mowana mine, near Dukwi. Cupric Canyon Capital are currently developing a copper-silver mine in the Kalahari Copper Belt, near their newly acquired Boseto copper project. By the end of 2016, the copper-silver projects that had been placed under care and maintenance were acquired by new owners and plans to resume operations were well under way.

The BCL Limited’s copper, nickel and cobalt operations at both Selebi-Phikwe and Phoenix mine were placed under care and maintenance in October 2016 due to financial difficulties brought on by depressed base metal prices. These too have attracted interest from possible buyers. (see Table 1 and accompanying notes).

In the rest of this section, we provide the history of the development of the two major base metal mines, the BCL and Tati Nickel Mining Company mines to provide some insights into the impact that these project have had on the Botswana economy.

Table 1 Base metal mines in 2015

<table>
<thead>
<tr>
<th>Mine metal / Production rate / capacity</th>
<th>Projected end life of mine</th>
</tr>
</thead>
<tbody>
<tr>
<td>BCL mine / (Cu, Ni, Co) RoM ore up to 3.0 Mtpa grading about 0.71% Ni and 0.65% Cu to produce 450 Kt/year of conc feed for the smelter grading 3.0% Ni and 3.2% Cu. Combined matte production in 2009 of 54K tonnes containing 29,616 K tonnes Ni and 13.6K tonnes Cu.</td>
<td>2022</td>
</tr>
<tr>
<td>Phoenix mine / (Cu, Ni, Co and PGMs) RoM ore up to 12 Mtpa and pre-concentrated to produce mill feed of 5.0 Mtpa to produce about 320 Ktpa of concentrate grading 5.5% Ni.</td>
<td>Dec 2015</td>
</tr>
<tr>
<td>Alecto Minerals’ 2008-2015: RoM ore 775.41 Ktpa at an average grade of 1.72% Cu. Alecto Minerals acquired Mowana in Q4 2016.</td>
<td>2033</td>
</tr>
</tbody>
</table>
1.2.1 BCL mine

The BCL mine project was originally based on ore reserves of some 43.7 million tonnes of ore grading 1.23% Copper and 1.18% nickel and a mine life of 22 years to December 1996. The discovery of additional ore resources at Selebi North extended the planned life of mine to beyond the year 2000 [22]. At full production including production from Selebi mine in 1979, planned production from the mine was 45,000 tonnes of concentrates to produce about 900,000 tonnes per year of concentrates to produce about 22.0 Ktpa of saleable copper.4

The BCL mine project required supporting infrastructure, largely from the Phoenix mine, for a total smelting capacity of about 34% was common stock held by US individuals through Parent Company BRST and effectively owned 33% of BCL [25]; disposal by AAC of its interests in both BRST and BCL to LionOre Mining International of Canada in 20021. This effectively left the government with 94% equity in BCL while 6% was held by LionOre. In 2007, Norilsk Nickel International acquired the Tati Nickel Mining Company and became a 6% shareholder in the BCL mine while government retained its 94% shareholding, and finally, in 2014, Norilsk Nickel International divested out of its interests in Botswana thus leaving the government as a 100% shareholder in both BCL and Tati.

At the time the company was placed under provisional liquidation it had a production capacity of up to 3.0 Mtpa of run-of-mine ore, grading 0.71% Ni and 0.65% Cu to produce some 450,000 tonnes conc grading about 3.0% Ni and 3.2% Cu [26]. It also had capacity to toll treat custom concentrates, largely from the Phoenix mine, for a total smelting capacity of 900,000 tonnes per year of concentrates to produce about 65,000 metric tonnes of matte per year [27].

1.2.2 Selkirk and Phoenix mining projects

The Selkirk and Phoenix mines were operated by Tati Nickel Mining Company, which was owned by AAC (43%), Francistown Mining and Exploration (42%) and the Government (15%). The Selkirk mine was commissioned in 2003.

Notes:
5. Cupric Canyon Capital acquired the nearby closed Boseto copper mine and is currently developing the Khoemacau copper project which is planned to start production in 2019 with a life of mine of 27 years. The project will process its ores through the existing Boseto plant. Accessed on 22/3/2017 from: https://www.cupriccanyon.com/development-exploration/zone-5-development

1 LionOre became an 85% shareholder in Tati with Government owning 15%. Downloaded on 21/3/2017 from: http://www.fin24.com/Economy/Anglo-sells-Botswana-nickel-interests-20020507
1989 at a production capacity of about 60,000 tonnes per year of direct feed ore grading 2.4% Nickel and 2.0 % Copper. The ore was transported some 200km by truck to the BCL smelter in Selebi Phikwe. In 1995, the Phoenix mine was commissioned as a dry concentration plant using magnetic separation with a production capacity of 80,000 tonnes of concentrate for direct smelting at the BCL smelter [25]. The Selkirk mine had an eleven-year life of mine to the year 2002 at which time it was placed under care and maintenance. In the same year, a hydrometallurgical treatment plant with a treatment capacity of up to 3.6 million tonnes of ore per annum was commissioned at the Phoenix open pit mine and it reached its full production in 2004. The plant was again expanded to 5.0 Mtpa in 2006. A dense media separation plant for pre-concentration of the run of mine ore was commissioned in 2008, which resulted in an increase in production capacity to some 12 million tonnes per annum of run of mine ore which gave rise to 5.0 Mtpa of feed to the plant [28]. The life of mine for the pit was up to December 2015.

The cyclicality of mineral commodity prices and their impact on mining projects, host communities and the wider economy in host countries can be drastic during periods of low metal prices. This is the case with the base metal mining sector in Botswana, which, in 2015 and 2016, experienced a total cessation of production due to low prices for nickel and copper. This raises the issue of sustainability of the base metal sector in Botswana as to whether the mines will restart once commodity prices improve; innovate to achieve better levels of cost competitiveness in the industry, or if the mines do not resume operations, what measures would policy makers take to reduce the impact of foregone economic and social benefits and the environmental risks from the unplanned closure of the mines.

The rest of this paper proceeds as follows: section 2 discusses the methodology and data analysis to determine the likely magnitude of the direct economic impact from the closure of the base metal mines and highlights the socio-economic and environmental risks that are likely to be suffered; the results are presented and discussed in section 3 and finally, section 4 concludes.

2.0 Material and Methods

Feasibility study reports provide information on the economic, social and environmental impacts of a project. Without the benefit of access to such feasibility study reports for both the Khoemacau and Mowana mine copper projects, the economic robustness of these two projects can be inferred from company reports about their position on the industry cash cost curve for copper. For the major two copper-nickel mines, BCL and Tati, there has been widespread reporting on the strategies the companies had developed in order to become sustainable, especially, in the face of a prolonged downturn in the prices of nickel and copper. Recent studies and articles in the media were also used as sources of information. These grey sources will be relied upon to piece together their journey towards sustainability and inference will be drawn regarding possible long term sustainability for these two mines.

Mining companies in Botswana have not yet adopted the Global Reporting Initiative (GRI) system of sustainability reporting and therefore do not produce such reports nor are their annual financial statements publicly available. At the time of this study, the main base metal mining operations of BCL and Tati were under provisional liquidation, which further complicated any efforts at data collection. The paper therefore analyzed the national accounts to derive an estimate of the direct impact on GDP that would be felt if the two mines were to move from care and maintenance and into permanent closure.

The national accounts state direct value added by economic activity. Mining activity includes all mines and the approach adopted involves the decomposition of mining industry value added into its major sources, diamonds and copper-nickel matte. Other minerals account for about 2% of exports and are therefore negligible. The following equations are used to decompose the mining industry value added into a mineral value added.

\[
V_{AX_{sector}} = \frac{V_{sector}}{X_{sector}}
\]  
\[
V_{AX_{mineral}} = \frac{X_{mineral}}{X_{sector}} \ast V_{AX_{sector}}
\]  
\[
V_{A_{mineral}} = \frac{V_{AX_{mineral}}}{V_{AX_{sector}}} \ast V_{A_{sector}}
\]

Where:

- \( V_{AX_{sector}} \) = computed value added to export value ratio for the minerals sector;
- \( V_{A_{sector}} \) = value added by the minerals sector from national accounts;
- \( V_{AX_{mineral}} \) = computed value added to export value ratio for the mineral;
- \( X_{sector} \) = Value of minerals sector exports from national accounts;
- \( X_{mineral} \) = Value of exports of given mineral or minerals from national accounts;
- \( V_{A_{mineral}} \) = computed value added by mineral.

The above calculations were evaluated for the period 2007 to 2015. The approach used to decompose the value added to the level of mineral oversimplifies the reality that exists between diamond and base metal mines, with the latter being characterized by low profitability and hence low levels of value added. The approach did not account for value added from coal, which is used for power generation at the mine mouth power plant operated by the Botswana Power Corporation. The approach does not compute indirect economic impacts that would arise due to the economic linkages that existed between the BCL and Tati mines with both the local and national.
economy, especially in the state-run utilities of water and electricity.

3.0 Results and Discussion

Sustainability risks of mine closure are classified according to the three main pillars of sustainability as environmental, social and economic. Low commodity prices or high costs have led to the closure or placement under care and maintenance of the base metal mining operations in Botswana. In this section, we discuss the direct economic, socio-economic and environmental risks from the closure or placement of the base metal mines under care and maintenance.

3.1 Economic risks

The economic risks from the closure of the base metal mines include the direct loss in value added from the mining operation and the indirect economic impacts from the linkages between the economy and the mining project. In this paper, our focus was only on the direct impacts.

The Khoemacau copper project is projected to have C1 cash costs, defined simply as all cash costs to produce a unit of metal less any by-product credit, of US$1.05 per lb of copper thus placing it in the lower quartile of the industry C1 projects[30].

Khoemacau and Mowana copper-silver projects are based on mine gate values that are realized after the concentrates have travelled long distances to seaports in Durban and eventually to overseas smelters.

Table 2 presents risks that are typical to mining projects and these may impact on the sustainability of the two copper projects discussed in this paper. For metal specific risks in copper projects, the top three risks are: price and currency volatility, access to energy and water and lastly resource nationalism. For nickel, the top three risks are: price and currency volatility, resource nationalism and capital projects which captures the schedule risk associated with large nickel projects)[30].

Table 2 Top five risks to the mining business

<table>
<thead>
<tr>
<th>Risk and ranking</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Cash optimization (price and currency volatility risk)</td>
<td></td>
</tr>
<tr>
<td>2. Capital access (not easy for Junior miners to raise debt financing due to lack of security)</td>
<td></td>
</tr>
<tr>
<td>3. Productivity – improve asset utilization efficiencies</td>
<td></td>
</tr>
</tbody>
</table>


As can be seen from Table 2, technological innovation, which is a driver for sustainability, fails to make it into the top 5 but it is in the 10th position in the full list of top ten risks for 2016/17 by Ernest and Young. Generally, it is accepted that mining operations are characterized by innovations that improve productivity as opposed to big shifts in technological benefits that characterize disruptive technological innovation [30].

The economic performance of the Khoemacau project would be affected by whether or not the planned grid power would be available when the mine goes into production in mid-2018. Any delay on the national power grid extension to the project would impact it negatively. The project will produce a copper concentrate grading some 42% Cu and this would be sold to Transamine Trading SA, a commodities trading company, and shipped to overseas copper smelters[29].

The risk of resource nationalism features in both copper and nickel projects but this cannot be considered a risk to sustainability in Botswana for the following reasons. Any government participation in mining projects is on a working interest participation basis and the fiscal regime is competitive and predictable. Regarding the push for further local processing to produce higher value mineral products, the government took the initiative to facilitate a cluster development approach for new mining projects from the Ghanzi Copper Belt and commissioned a pre-feasibility study in 2014 to investigate the viability of a base metal smelter and refinery for projects located on the Ghanzi Copper Belt. This was a project that included all stakeholders. Unfortunately, the depressed metal prices led to a reprioritization of the study.

Regarding the strategies to turn BCL and Tati into sustainable businesses, local news sources scanned reveal that there was engagement between officials of the two mining companies and the general public in which the latter were briefed about major strategic objectives such as Polaris II, a strategy under which the company aimed at diversifying its mineral portfolio to include other mining businesses along the full mining value chain from exploration to refining and related businesses. The
message that comes out regarding the journey that the BCL Group took towards sustainability is that:

- since about 2004, BCL has been carrying out exploration within the mining licence area in order to extend its life of mine beyond 2013;
- In June 2010, BCL completed a scoping study aimed at identifying sufficient resources to extend its life of mine and this was subsequently extended to 2022;
- In 2014, the company cleaned its balance sheet by repaying its P3.3bn or about US$330m, to its sole shareholder, the government made up of P1.0 bn in cash and P2.3 bn in shares;
- Operating costs were on the rise as the mines were getting deeper and ore reserves were becoming less and less;
- With a clean balance sheet, BCL entered into agreements to acquire Tati and 50% of the Nkomati nickel mine in South Africa for a total of cost of just over P3.1bn or about US$300m and in addition invested P58m in Pula Steel, a scrap smelting and fabrication plant. In 2015, BCL refurbished its smelter at a cost of P800m (US$80m); 4;
- The BCL Group developed a reorganization plan in the first quarter of 2016 under which the mine was to shed some 1,800 – 2000 of its 6000 employees at both BCL and Tati. The Group was reeling under a creditor debts of about P900m (US$90m) and continued to make losses as its cash operating costs were US$7.00/lb while the nickel price was about US$4.00/lb 3;
- In June 2016, Selkirk mine was reported to have added six years to the life of Tati, which, without these new open pit reserves, was bound to close by December 2016; 5
- On Friday October 7th, 2016, Cabinet took a decision to place BCL under provisional liquidation. The government could not afford the P8.0bn that was needed to keep BCL going. This amount included the P1.0bn that BCL needed to cover its operating losses in 2016 in addition to the P1.0bn that was advanced by Barclays Bank and guaranteed by the government, the P3.0bn or so for the acquisition of Nkomati and further losses forecast over 2017.

It is obvious from the foregoing news coverage about the journey that BCL had embarked on towards being a sustainable mining enterprise that in the event the government is successful in finding a buyer for the two mines, the concern about long term sustainability will remain due to the projected short life of mine to the year 2022.

Table 3 presents the results of the computed percent value added by base metal mines, which depicts a declining trend from a high of 6.31% in 2007 to a low 1.97% in 2015. In current terms, a 2% decline in GDP due to the permanent closure of BCL represents a direct decline of about P2.874 million. One can therefore interpret the results to represent a best-case level of direct economic contribution by the two copper-nickel mines to the country’s GDP. This will therefore represent the maximum direct economic risk to GDP that would be suffered from a permanent closure of the four base metal mines.

Table 3 Results table

<table>
<thead>
<tr>
<th>Mining value added (Pm)</th>
<th>2007</th>
<th>2011</th>
<th>2015</th>
</tr>
</thead>
<tbody>
<tr>
<td>% of GDP at current prices</td>
<td>29.14</td>
<td>23.37</td>
<td>17.58</td>
</tr>
<tr>
<td>Total GDP at Current prices (Pm)</td>
<td>67 153</td>
<td>104 980</td>
<td>145 923</td>
</tr>
<tr>
<td>Mineral sector exports</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rough Diamonds</td>
<td>19 967</td>
<td>28 851</td>
<td>25 809</td>
</tr>
<tr>
<td>Polished Diamonds</td>
<td>0</td>
<td>4 857</td>
<td>4 444</td>
</tr>
<tr>
<td>Copper-Nickel matte</td>
<td>5 522</td>
<td>3 398</td>
<td>3 790</td>
</tr>
<tr>
<td>Ratios</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>VAX sector</td>
<td>0.77</td>
<td>0.66</td>
<td>0.76</td>
</tr>
</tbody>
</table>

3 Sources for the first 5 bulleted points are:
BCL sees no alternative to Polaris II as it fights to keep Selebi- Phikwe alive


from the smelter at Selebi-Phikwe into the atmosphere, of the land affected by the discharge of sulphur-dioxide gases remain to be addressed. There is a need to address rehabilitation.

Typical socio-economic benefits include creation of employment, services and markets for local businesses, as well as subsistence livelihoods of community members [18]. At mine closure these socio-economic benefits are put at risk due to the dependence of the host communities on the mine for employment, services and markets for local businesses, as well as subsistence livelihoods of community members [18].

There is a general consensus that both Selebi-Phikwe and Francistown will be affected by the closure of the BCL and Tati mines respectively. While there is a lowered risk of turning into a ghost town for Francistown, the same cannot be said about Selebi Phikwe, where the economic activities are in one way or the other linked to that of the mine.

3.3 Environmental risks

The main environmental risks from mining are contamination of water, acid mine drainage, air pollution from releases of toxic and greenhouse gases, emission of heavy metals, disruption of animal habitats, biodiversity and natural landscape [6, 11].

While the mine had outlined a number of environmental policy objectives to address the negative impacts of its operations, the financial difficulties that the mine faces means that the risks remain to be addressed. There is a need to address rehabilitation of the land affected by the discharge of sulphur-dioxide gases from the smelter at Selebi-Phikwe into the atmosphere, discharge of wastewater from the mine into recipient river systems with heavy metals, suspended and dissolved solids as well as acid mine drainage from the tailings and slag dumping sites [31].

4.0 Conclusions

The impact of low base metal prices were felt the most in the years 2015 and 2016 in the base metal mining sector in Botswana, which over this period experienced a total cessation of production of copper-nickel matte from the BCL and Tati mines as well as copper and silver in concentrates from the Boseto and Mowana copper mines. There is hope that the Boseto and Mowana copper mines will resume operations as they have been bought by new owners and they reportedly have robust business cases, which other things equal, means that they have the potential to be sustainable.

The fate of the two major copper-nickel mines, BCL and Tati, depends on whether or not the government finds a buyer. We estimate that the direct impact on GDP from the permanent closure of these two mines will be 2% of GDP or about P2.874 million (USD287 million at current exchange rates) currently. The closure of the two mines raises socioeconomic risks as unemployment will rise in the Selebi Phikwe and Francistown regions. Due to the higher number of employees that were employed at BCL and the general reliance of the town on the BCL mine, the socio-economic impacts will be felt more at Selebi-Phikwe than in Francistown. It is also very likely that as a result of the financial difficulties at the two copper-nickel mines, no funds were set aside for environmental rehabilitation, which means that any future rehabilitation work would have to be funded by the government, which owned the mines.

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Economic Benefits and Technical Complexities of Grade Engineering® in Strategic Mine Planning of Metalliferous Projects

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Abstract — Grade Engineering® (GE) is a concept created by CRC ORE (Cooperative Research Centre for Optimising Resource Extraction) to describe a range of pre-concentration technologies and operating methods to improve head grades and throughput (TPH) to processing destinations. This is achieved through the early rejection of low value material prior to energy and cost intensive activities (e.g. milling and flotation). Grade Engineering pre-concentration technologies include; particle and bulk sensor based sorting, differential blasting to induce deportment of grade to finer size fractions to be separated by screening and screening for the preferential deportment of grade by size.

It is estimated that if the entire mining system is optimised incorporating Grade Engineering, the global financials of a project could be significantly improved by 3 to 15% in Net Present Value (NPV). Furthermore, total ore reserves could be increased and the geological resource could be utilised in a more efficient and sustainable manner. Although Grade Engineering offers flexibility and added value to a mining asset, it also comes with associated technical and operational complexities that must be addressed. One of the main technical complexities is the characterisation of the geological resource by pre-concentration (separation) properties. A new dimensionless geometallurgical attribute has been created to describe separation potential named Response Ranking (RR).

The separation behavior of each GE technique and rock type as Response Ranking (RR) attributes must be incorporated in the fundamental input data (Block Model) to facilitate assessment in all optimisation stages for strategic mine planning, including; final pit and production schedule/cut-off grade optimisation. This is required to assess the delta in economics achieved by Grade Engineering, and to use it as a strategic tool to reach objective indicators set by the mining project such as; net present value, internal rate of return, metal production, and life of mine.

To incorporate Grade Engineering economic benefits and complexities into the main optimisation stages of mine planning, Lane’s value and cut-off grade equations have been revised and updated for Grade Engineering and pre-concentration. This paper provides a discussion of GE economic block modelling, GE final pit optimisation, GE value and cut-off grade equations and GE production schedule optimisation at a metalliferous project. The optimisation of the case study incorporating Grade Engineering improved the NPV from developing the mineralised deposit through open pit mining.

Keywords: Mine Planning Optimisation for NPV, Grade Engineering, Sensor Based Sorting, Mining Economics, Strategic Mine Planning.

1. Grade Engineering®

Grade Engineering is a concept created by CRC ORE (Cooperative Research Centre for Optimising Resource Extraction) to describe a range of pre-concentration technologies and methods to improve ore feed grades and throughput to process plants. Grade Engineering can be used to remove uneconomic material from ore, recover economic material from waste and improve material allocation decisions by separating high and low value material prior to energy and cost intensive processing activities [6], [2].

It is understood through sampling, testing and simulation that besides having the property of upgrading average grades, overall mill throughput (TPH) of an operation could also be improved through Grade Engineering. Some rock types show the property of having higher grade in finer size fractions (smaller granulometry) after blasting and coarse breakage. When GE separates the higher value (finer fraction) material through screening, this material can show improvements in mill TPH as smaller granulometry will reduce time and energy required for comminution [4]. This TPH improvement is a function of screen aperture, hardness and P20 [4].
The four course separation pre-concentration techniques that encompass Grade Engineering are as follows:

- Preferential grade by size deportment (natural deportment).
- Differential blasting (induced deportment).
- Sensor based sorting (particle and bulk).
- Coarse gravity separation (dense media, or in-line pressure jigs).

Preferential grade by size deportment (natural deportment) describes the natural phenomenon of the metal to preferentially deport into specific size fractions of material after coarse breakage. This natural characteristic is a function of rock mechanical properties, mineralisation style, and energy to condition the size fractions of the rock [2].

Differential Blasting (induced deportment), is a methodology for drilling and blasting where each blast hole is charged with different energy according to the grade it intersects. Low grade regions within a blast block are charged with lower energy, and high grade regions are subjected to higher energy. The purpose of this charging strategy is to create a bimodal ROM size distribution that can then be separated through screening to create a high-grade stream (accepted fines) and a low-grade divert stream (divert coarse) [8].

Sensor based sorting (particle and bulk) are pre-concentration techniques which achieve coarse separation of valuable material from non-value material through sensor measurements of grade or value (gamma activation, laser, optical, MR, XRF, etc.) and separation systems which can be in the form of mechanical, hydraulic or pneumatic arrangements [6].

Coarse gravity separation applies dense media separation and in-line pressure jigs at coarse scale (>10 mm), achieving particle separation based on density properties of feed. Compared to other Grade Engineering techniques coarse gravity separation requires secondary crushing and screening of feed to create a specified particle size distribution [5].

The separation or yield response achieved by Grade Engineering techniques are a function of intrinsic material characteristics and the grade heterogeneity of material [7].

Grade Engineering techniques use yield response parameters to describe the separation potential of material by a given GE technique. These can be modelled through dimensionless dynamic parameters called Response Rankings (RR). Separation response parameters are also used to calculate the grade and quantity (mass) of the upgraded and downgraded products created by the GE plant. The percentage of mass separated as upgraded (high value) material is denominated ‘Mass Pull’ factor. Figure 1 shows the change in grades in upgraded and downgraded material achieved by the different mass pull factors [3].

![Figure 1. Response curves for ore types and separation devices][1]

Response Ranking (RR) parameters are used to model preferential grade by size deportment and describe the mathematical relation between metal upgrade and cumulative mass to screen underside (Figure 1) [3]. Other GE techniques such as sensor based sorting use different yield-response curves, however these curves can be translated into an effective RR to simplify comparisons and modelling.

Grade Engineering techniques deliver an additional level of strategic planning and operational flexibility which can be exploited in long-term and short-term planning. For short-term planning, GE provides flexibility to dynamically improve production, while at the strategic long-term level, GE improves the NPV of the project [6]. This paper provides an overview of the successful application of Grade Engineering in long-term mine planning optimisation for a copper porphyry open pit mine.

2. **GE Optimisation Methodology**

The work presented here is part of a UQ PhD project in partnership with CRC ORE and COMET Strategy, focused on developing an innovative methodology, models, algorithms and value equations around the application of GE technologies in long-term mine planning optimisation. With the purpose of maximising and improving the NPV of the mining project.

This work also presents a case study in which the long-term mine plan was completely re-optimised incorporating Grade Engineering concepts and techniques. The result of this case study is a Grade Engineered optimised mine plan and business case.

The case study focused on the assessment of three Grade Engineering techniques; preferential grade by size deportment, differential blasting, and sensor based bulk sorting. These were incorporated into the 3D resource block...
model and long-term mine planning optimisation steps, specifically the determination of reserves (final pit) and mine production scheduling [10].

The Grade Engineering benefits examined in this case study included improvements in processing head grades and comminution throughput (TPH) from GE material.

The case study is a large copper porphyry open pit operation located in South America. This deposit also contains molybdenum as a by-product, and arsenic as deleterious element. The operation has two mineral processing methods: comminution and floatation (mill) for sulphide minerals and dump leaching (leach) for sulphide and oxide minerals.

To successfully create a Grade Engineered mine plan and business case that evaluates all Grade Engineering strategies and scenarios, the following methodology was followed (Figure 2).

3. Incorporating GE Attributes into Mine Planning

The concept of Grade Engineering coarse separation can be embodied in separation response attributes for modelling as Response Rankings (RR). These are dynamic attributes (curves) representing a series of possible grade/mass outcomes dependent on the retained mass for individual block model ‘blocks’ and mining units as a ‘grade bin’.

This challenges conventional practice based on fixed in-situ average grades and tonnages. Essentially, instead of having one average grade per block or mining unit, now a series of grade and mass relationship scenarios are available for planning and development.

This is illustrated in Figure 3 where a block with an average grade of 0.5% Cu and a Response Ranking (RR) of 100 has different grade/mass modalities.

![Figure 3](image3.png)

Figure 3. Example of the dynamic interplay between retained mass and Cu upgrade for a set Response Ranking.

The bottom graph presents several separation options with their respective mass pulls, upgraded (red) and downgraded (black) material products for the given mining unit (block). In this example, a mass pull of 70% generates an upgrade of 1.2 time the initial feed grade, whereas a mass pull of 30% generates an upgrade of 1.6 times the initial feed grade in the higher-value separated fraction.

This dynamic interplay provides an additional degree of flexibility to mine planning, where instead of having a fixed total unit value ($/t) and one destination per block or mining unit, there are now flexible options of having two or more new streams of coarse separated material with different grades/mass, unit value ($/t) and economic destinations.

These dynamic characteristics also provide the capability and flexibility of dynamic outcomes that can be used to improve fulfillment of targets (production, quality restrictions), and management of capacity constraints (mining, processing).
These new properties require a more sophisticated mine planning approach to determine optimal mining policies and strategies that maximise value (NPV) and consider variable operating costs and capital costs related to coarse separation.

Therefore, separation response attributes as Response Rankings (RR) across the three pre-concentration separation technologies are the key Grade Engineering attributes that need to be used and incorporated into all main mine planning steps including: 3D block model, final pit, and production schedule optimisation for NPV.

4. Grade Engineered 3D Block Model

Grade Engineering separation response parameters need to be imbedded into one of the most important data sets for mine planning optimisation, the resource block model (RBM). Separation response parameters also form the basis for new GE financial, geological and technical parameters. The embedment of these into the RBM will create the Grade Engineered 3D block model (GE3B), which will be the foundation for the GE final pit optimisation and subsequent GE production schedule optimisation.

Incorporation of these dynamic GE attributes into the block model start with the creation and flagging of RR domains through geostatistical methods. RR variability could be modeled and attributed to rock types, minerology or other geological parameters [5]. This must be defined through a combination of physical laboratory testing at drill core or bulk scale and simulation using geometallurgical concepts [1].

5. Grade Engineering Economics Mine Planning

5.1 Traditional Value and Cut-Off Grade Equations

Traditional mine planning economics is based on the calculation of ‘fixed’ total unit values ($/t) to a resource block (3D Block Model), bench or any other mining unit throughout the resource and the calculation of breakeven and operational cut-off grades as explained by Lane [9].

Total unit value ($/t) is usually in the form of ‘Total Value’ or ‘Net Smelter Return (NSR), which is used to calculate the value ($/t) of a mining unit when treated by a specific metallurgical process or sent to the waste dump.

Unit values ($/t) are used to decide the optimum economic destination of mining units to maximise the total value (NPV) of a mining project. Typical ‘direct feed’ economic destinations in copper porphyry deposits are: mill (comminution and floatation plant), leach (lixiviation process) and waste dump as seen in Figure 5.

Figure 5. Typical ‘Direct Feed’ economic destinations in copper porphyry open pits.

Traditional total value and Cut-off grade (COG) equations as described by Ken Lane have the following form;

\[ V = (P - K)xy\bar{g} - xh - m - (f + F)\tau \]  
(1)

\[ COG_m = \frac{h}{(P - K)y} \]  
(2)

Where:

- \( V \) = Increment in present value per unit of resource utilised
- \( COG_m \) = Cuoff Grade in a mining constraint operation.
- \( P \) = Metal Price
- \( K \) = Selling Cost
- \( x \) = Ore to mineralised material ratio
- \( y \) = Metallurgical Recovery
- \( \bar{g} \) = Average grade of \( x \)
- \( h \) = Processing cost
- \( m \) = Mining cost
- \( f \) = Annual fixed Cost
- \( F \) = Opportunity Cost
- \( \tau \) = Time per unit of resource

The above COG equation is for an operation constrained by mining capacity. To see all possible value and cut-off grade equations please refer to Lane’s book ‘The Economic Definition of ORE’ [9].

5.2 Grade Engineering Value and Cut-Off Grades Equations

Grade Engineering on the other hand has the capacity to create a dynamic range of unit values ($/t) per mining unit, that are made up of all possible combinations achievable in by a Grade Engineering technique.
These combinations are a function of grade/mass, cost of separation, and economic destination of each of the two separated products per mining unit (see Figure 6).

Please note that Grade Engineering is modelled as another economic destination. Now a mining unit can choose to be treated as direct feed or Grade Engineered feed.

![Figure 6. Schematic illustration of Grade Engineering as a second feed option for a mining unit, and final product destination options after separation.](image)

In Figure 6, ‘product 1’ from the GE plant represents the separated fraction that has upgraded material qualities, while ‘product 2’ is the material with downgraded properties. The sum of the two products equates to the original mass and average grade of the initial mining unit entering the Grade Engineering plant.

There is now the ability to separate a mining unit into two products and treat them individually for optimum processing. This not only increases the original unit value ($/t) per mining unit, but also creates the capability to better fulfill production targets while managing capacity constraints (mill, mining), consequently increasing the value (NPV) of a mining project.

To incorporate these new complexities and benefits into mine planning economic evaluations and optimisations, Grade Engineering value and cut-off grade equations were created having the following form:

\[
VGE = (P_H - K_H) x M_P y_H \hat{g}_H + (P_L - K_L) x (1 - M_P) y_L \hat{g}_L - x M_P G_H - x (1 - M_P) G_L - m - (f + F) \tau
\]

(3)

\[
GECOG_m = \frac{M_P (h_H - G_H) + (1 - M_P) (h_L - G_L)}{(P_H - K_H) M_P y_H f_H + (P_L - K_L) y_L (1 - f_P) M_P}
\]

(4)

Where:

\[VGE = \text{Grade Engineering® Value Equation.}\]

\[GECOG_m = \text{GE Cutoff Grade in a mining constraint operation.}\]

\[P_H = \text{Metal Price for High graded material}\]

\[K_H = \text{Selling Cost for High graded material}\]

\[M_P = \text{Mass Pull (% Of High graded material)}\]

\[y_H = \text{Metallurgical Recovery for High grade material}\]

\[\hat{g}_H = \text{Average grade for Low grade material}\]

\[P_L = \text{Metal Price for Low graded material}\]

\[K_L = \text{Selling Cost for Low graded material}\]

\[M_P = \text{Mass Pull (% Of Low graded material)}\]

\[y_H = \text{Metallurgical Recovery for Low grade material}\]

\[\hat{g}_L = \text{Average grade for Low grade material}\]

\[h_H = \text{Processing cost for High graded grade material}\]

\[h_L = \text{Processing cost for Low graded grade material}\]

\[G_H = \text{Grade Engineering cost for High graded grade material}\]

\[G_L = \text{Grade Engineering cost for Low graded grade material}\]

The above GE COG equation is for an operation constrained by mining capacity.

In applying and testing the above GE equation through studies performed by CRC ORE, results have shown that application of Grade Engineering in domains with Response Rankings (RR) greater than 50 have the capability to convert previous non-economic material into ore and to increase unit value ($/t) of mining units (blocks) already assigned to a processing destinations (mill or leaching). This provides potential to increase total reserves, as well as change the geometry and dimensions of the final pit.

In Figure 7, the cross section of a typical porphyry Cu-Mo deposit flagged by economic destination.

The cross section in Figure 7 shows the deposit been flagged only by ‘direct feed’ (mill, leach, waste) using the traditional value and COG equations. Figure 8 shows the same cross section but this time flagged with ‘Grade Engineered feed’ and ‘direct feed’ according to the new presented GE value and GE COG equations.
Figure 8 also shows that around 30% of the resource is economically exploitable through coarse separation based on preferential grade by size and differential blasting levers. While 70% of the resource is economically exploitable as direct feed to the different destinations. In this case, blocks assigned to coarse separation generate more value than their original destination. This includes a proportion previously assigned as ‘direct feed’ to waste. The overall outcome is a significantly improved NPV for the project.

A good way of visualising cut-off grades and the value that every economic destination creates on a specific ore type across its grade bins is through the plotting of value curves. Figure 9 shows value curves and cut-off grades of an ore type tested on all available economic destinations. In this case the study includes: mill, leach, waste, GE grade by size, GE differential blasting, and GE sensor sorting.

GE value curves were created using the presented GE value equations in the last section. Curves include optimum GE configurations for each grade bin. An optimum GE configuration maximises value through selecting an optimum mass pull factor and optimum economic destination for each separated product (upgraded material and downgraded material).

These curves assume that 100% of a grade bin material is either treated by direct feed (mill, leach, waste) or a Grade Engineering technique.

Value curves in Figure 9 show that GE grade by size and GE differential blasting techniques add more value than mill, leaching and waste in grade bins ranging between 0.10 and 0.62 % Cu. However, GE sensor sorting adds more value than any other economic destination in bins between 0.14 and 0.80 % Cu.

The cut-off grades for direct mill feed with no Grade Engineering intervention is 0.36 % Cu. However, new COG is calculated for direct mill feed with the incorporation of GE. If differential blasting is the technique to be used in the operation, direct mill feed will have a COG of 0.63 % Cu. If sensor sorting is the technique to be used in the operation, then direct mill feed will have a COG of 0.80 % Cu.

These results apply only to this case study and to a specific rock type. Every deposit is unique and will show different coarse separation properties and potential, hence different Ranking Response (RR) domains. Previous studies and theory have showed that larger RR values add more value to a project, compared to lower RR values.

5. Grade Engineering® Final Pit Optimisation

Final pit optimisation (FPO) is the step within long-term mine planning where the ore reserves of a mineral deposit are estimated, the geometry of the final pit is determined and a first estimation of the value of the project is calculated. However, this optimisation does not take time into account. It only considers financial and technical parameters fixed in one period of time. Variable time constraints are taken into account in the production schedule optimisation for NPV which is reviewed in the next section.

A Grade Engineered final pit optimisation evaluates the possible expansion of the initial reserves, and the possible change of the final pit geometry as result of incorporating Grade Engineering coarse separation pre-concentration technologies.

The GE final pit shown in Figure 10 was modelled and optimised using DS Whittle 4.6 software. The input data includes financial, technical, and GE specific parameters.

Figure 10. Grade Engineered final pit shell – 5% Improvement in reserves.

The specific GE parameters imported for every mining unit (block) into DS Whittle 4.6 included; optimum Grade Engineering technique, optimum mass pull factor, optimum grades and optimum economic destinations for upgraded and downgraded material. Optimum in this context is the combination of GE pre-concentration technique, mass pull, and product destination that gives the maximum total unit value ($/t) for an individual mining unit (block).

Based on the current processing facilities of the case study, three possible combinations of economic destinations are possible for every block separated through GE:
GEML (upgraded material is sent to the mill and downgraded material is sent to leach facilities)

GEMW (upgraded material is sent to the mill and downgraded material is sent to waste).

GELW (upgraded material is sent to the leach facilities and downgraded material is sent to waste).

- Optimum separation ‘mass pull’ schedule
- for material per year, and for the life of the mine.

COMET Optimal Scheduler was used as the optimisation software to build the GE production schedule optimisation model.

This software is a sophisticated mine scheduler designed to model multiple ore type processing and interactions between the processing facilities, the objective of each optimisation is the maximisation of Net Present Value, taking into account changing conditions throughout the project life such as economics and productivity. Its optimal scheduler integrates mine planning and financial analysis to assist with understanding the impact of mining decisions on the NPV of the assets.

The COMET model developed as part of this study was configured using a linear programming module to find the best destination for every bin of material mined in every moment of the mine plan. This was achieved by comparison of the value of all different process alternatives including GE options (grade by size, differential blasting and sensor based sorting).

A large matrix of GE scenarios and iterations were evaluated utilising the COMET Optimal Scheduler. The strategic scenarios combined all possible ‘mass pull’ factors, GE techniques (grade by size, differential blasting, sensor sorting) and best processing destination (mill, leach, waste) for upgraded and downgraded material.

The Optimum Grade Engineered mine plan (OGEMP) is the one that found maximum NPV of the project through the use of coarse separation, taking into account all time based financial, technical parameters and constraints. The case study OGEMP for a sensor based bulk sorting scenario had the following configuration and assumptions:

- GE Techniques: sensor based sorting (Bulk).
- GE sensor based sorting plant starts in 2018.
- Yearly variable ‘mass pull’ ranging from 20% to 50%.
- 30 Mtpa (83 ktpd) max Grade Engineering plant capacity.
- GE CAPEX – 85M US$.
- GE OPEX – 0.36 US$/t (Average)
- TPH improvement of 5%
- Discount rate - 8%
- Improvement on base case NPV – 9.1% (640 M US$)
- Payback time – 3 to 4 Years.

Figure 11 shows the results of one of the GE final pit optimisations.

Figure 11. Final pit optimization – Revenue factors.

This case study showed an increment in total reserves of 5% as result of incorporating Grade Engineering into the operation. However there have been studies showing increments greater than 10% in total reserves depending on orebody geometry and the distribution of marginal ore within overburden and waste stripping of the final pushbacks.

6. Grade Engineering® Production Schedule Optimisation and Results

The most vital optimisation step within long-term mine planning is the production schedule optimisation. This step seeks the optimum cut-off grades, extraction sequence and processing schedule for every year throughout the life of mine (LOM) that maximises the NPV of a mining project.

This step, in comparison with those previously described, does consider time in the form of a discount factor, and all time dependent financial, technical, and operational parameters and restrictions.

A Grade Engineered production schedule optimisation incorporates coarse separation pre-concentration techniques (grade by size, differential blasting, sensor based sorting) in the life of mine (LOM) schedule with the intention of finding the following:

- Optimum mining sequence that incorporates coarse separation and maximises the total NPV.
- Grade Engineering and direct feed cut-off grades profiles for the life of mine.
- Amounts of material to be treated as ‘direct feed’ and as ‘Grade Engineered feed’.
- Optimum Grade Engineering techniques to be used and its annual capacity.
The results show a successfully optimised production schedule incorporating Grade Engineering as another economic destination for material, together with mill, dump leach and waste dump as presented in Figure 12.

Detailed results show that for the GE Sensor Sorting case a total of 570 Mt would be treated at the Grade Engineering® plant using an average mass pull of 41% to high-graded material (Product 1).

The total amount of material treated by the GE plant represents 20% of the total material movement (Orange area) as seen in Figure 12. The process plant (Mill) receives material from three main sources; mine, stockpile, and upgraded material from the Grade Engineering Pant. Around 20% of the total Mill feed belongs to GE upgraded material (Product 1) as seen in Figure 12. These combine to increase the total average Cu% head grade at the mill by 6.7% when compared to the base case. Consequently producing 8.0% more Cu in concentrate, and allowing to increase Cu metal in concentrate by 2.8%, and increase Cu cathode metal by 17.3%.

Figure 13 shows the Cu% mill head grade improvement achieved by GE Sensor Sorting throughout the life of mine compared to the Base Case.

Figure 14 shows the life of mine schedule for the GE Sensor Sorting plant including the final destination of treated material. This graph also shows the average Cu grades of material entering the GE plant (dashed black line), and the average grades achieved for the different products. Total head grade feed to the GE plant has an average of 0.37 Cu%, high-graded material sent to Mill 0.67 Cu%, material sent to Dump Leach 0.36 Cu%, and material sent to Waste Dump 0.28 Cu%.

As the mine life schedule continues ‘Mass Pull’ of 50% becomes the main separation ratio treating higher grade material. The average ‘Mass Pull’ for the Grade Engineering® separation schedule is 41%.
7. Conclusion
The work and case study presented in this paper concludes that by the successful implementation of Grade Engineering pre-concentration techniques (grade by size, differential blasting, sensor based sorting) into all steps of long-term mine planning optimisation (3D resource block modelling, final pit optimisation, and production schedule optimisation) the NPV of a project can substantially be improved and ore reserves can be expanded.

This work proved the viability and potential of a ‘Grade Engineered Optimised Mine Plan’, for a porphyry copper-molybdenum open pit mine, were the following was achieved:

- NPV (Net Present Value) improvements in the order of 3% to 15% of the overall project.
- Head Grade improvements ranging from 3% to 10% in the overall mill feed.
- Throughput (TPH improvements ranging from 5 to 10% to the overall mill feed).
- GE supported a lower minimum cut-off grade allowing the economic treatment of marginal material and permitting the final pit to expand total reserves by 5%.

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References
Abstract—Computer planning and design of mining projects is mainly done based on block model, in which the block size is usually determined based upon exploration database in order to achieve the most precise grade and tonnage of the deposit. Even though in production stage block dimension effects the capital and operating costs in addition to profitability, efficiency and capacity of equipment, these parameters are not considered when defining the block size. Therefore, despite the ability of some algorithms to determine the optimum solution, the optimality of these algorithms is in doubt; since these algorithms are based on some assumption such as the known block size. This study uses the industry standard for mining cost estimation as well as O’Hara method to define the capital and operation costs of drilling, blasting, loading and haulage of different sizes of blocks. The results of the research showed that increasing the block size will decrease the overall operating cost of extracting one ton of material.

Keywords: Block size, open pit mine, major mining operations, capital costs, operating costs,

1. Introduction

Almost all algorithms have been developed for design and production programming are based on the block model. A block model divides the area under study into a matrix of blocks each of which is assigned based on grade estimation, ore content and possibly other characteristics derived from geostatistical data. The importance of block dimensions in pit design and mine planning is well-known in the technical literature. During the last decades, the impact of block dimensions on the pit size [1], on the estimated error [2], on the mining dilution [3], on the long term production planning [4], on the grade estimation [5] and the on economic life of open pit mines [6] have been evaluated by different researchers. In this article, the sensitivity of capital and operating costs to the block size will be analyzed.

Clearly, mine equipment size and block dimension are two interdependent parameters. Mostly, the block height is selected equal to the bench height or to some exact fraction of it, in order to balance between the bench face and loading equipment. Indeed; the horizontal dimension of the blocks is determined so as to obtain the most accurate pit slope angle. Slope angle relies on loading equipment as well.

Large blocks decrease the calculation time and the variance of block estimation and selectivity. However, it leads to the possibility of selection larger equipment so the productivity will be increase. As a result, more capital costs and less operating costs is required for large block size. Smaller blocks increase the resolution and selectivity of the model. Therefore, smaller blocks are useful in planning but increase the variance of the estimation in the individual blocks and also the calculation time. Consequently, the smaller equipment should be selected which causes less productivity and total net present value (NPV). Furthermore, the extraction costs of smaller blocks are more than larger blocks.

In mining operation, drilling, blasting, loading and haulage are the major operations. Thus, the cost of these operations is discussed for different block size. The cost of equipment consists of two parts, capital, and operation. The exact capital cost is hard to estimate, as purchase price varies with sales’ volumes, equipment life is hard to quantify, and salvage or resale value is unknown [7]. Many methods are available to estimate the mining costs such as cost estimation from similar mining operations, accounting cost estimation methods and the O’Hara cost estimation method [8]. In this research, O’Hara method is applied to define the capital costs of different operations. Although the constant variables of O’Hara’s equations change during the time and for different situations, this fact has not been accounted in this study. Since the objective of the research is to illustrate the trend in cost variations for different sizes of blocks.

2. Drilling operation

The size, blast-hole diameter, and the number of required drills depend on the tons of ore and waste to be drilled off daily. Hole diameters are matched to the bench heights and consequently to the block height. There are many rules of thumb for matching hole diameter to bench (block) height. A common method for calculating this relationship is that diameter in inches should be less than the height in feet.
divided by five [9]. Accordingly, for high benches, large blast-hole diameter can be accomplished which allows utilizing more explosives, with wider spacing between the blast-holes. This generally means lower drilling costs with fewer holes to be drilled for a given production.

Figure 1 shows the minimum and maximum range of appropriate hole diameter for each block height. Based on this figure, the mean blast hole diameter \( d \) can be determined based on block height \( BH \) as:

\[
\text{d} = 0.4 \times \text{BH} + 0.9 
\]

2.2. Drilling operating costs

The source of operating costs in this study is the information obtained from industry standard for mining cost estimating which is annually published by InfoMine [10]. Considering this database, the operation costs, including maintenance, fuel/power, lube, tire, wear parts, drill bit were collected for different hole diameter. Figure 3 illustrates the relationship between hole diameter and operating costs for percussion, rotary, and percussion-rotary drills. It can be seen that operating costs grows steadily by increasing hole diameter.

Based on Figure 3, the relation between drilling operating costs \( C_{\text{dope.}} \) and hole diameter \( h \) can be defined as:

\[
C_{\text{dope.}} = 16.13 \times h + 16.62 
\]

Due to the high productivity of large equipment, the operating cost per ton of material is less than the small equipment. To determine the cost per ton, the working hours per day \( \text{whpd} \) should be considered. Considering equations 1 and 3, operating costs of drilling per each ton of material is defined as:

\[
C_{\text{dope,}} = \frac{6.45 \times h + 31.14 \times \text{whpd}}{27h^2 + 122h + 137.7} 
\]

Figure 2 shows that capital cost of drilling operation rises exponentially by increasing the block height. Due to the fact that larger blocks eventuate higher productivity, the number of drills also changes in Figure 2.

Figure 3. Drill operating cost vs. hole diameter

Figure 4. Drill operating cost vs. block height

3. Blasting operation

The economic analysis of the use of explosives is an important part of blasting operations in mining. To define the
blasting cost, the number of required blast holes to explode
one ton of material and the amount of explosives loaded in
each hole should be calculated. Having the costs of
explosives, wire, detonators and primer/booster of each blast
hole, the blasting cost for one ton of material can be
estimated. In this study, Anderson’s equation was used to
determine the burden:

\[ B = \left( 2 \frac{D_e}{D_r} + 1.5 \right) \times d \times 0.3048 \]  \hspace{1cm} (5)

where \( D_e \) and \( D_r \) are the density of explosives and rock
respectively and \( d \) is the blast hole diameter per inch.

Ash equations were then applied to compute spacing, sub-
drilling and stemming. The coefficients of \( K_s \), \( K_t \) and \( K_j \)
were selected 1, 0.7 and 0.5 respectively. These numbers were
chosen with the objective of utilizing the maximum possible
amount of explosives. Hence the largest cost would be
obtained. Based on the obtained drilling pattern, the kilogram
of explosives per meter of blast hole (\( Q \)) can be written as:

\[ Q = \frac{0.507(D_e)d^2 \times L}{B^2 \times h 	imes D_r} \]  \hspace{1cm} (6)

where, \( L \) is the charge length (m) and equals \( h-0.2B \).

Applying equation (6), it is possible to define the cost of
explosives (\( C_e \)) for each hole. Considering 5% of \( C_e \) as the
cost of galvanometers and cap connectors and 15% of \( C_e \) as
the overhead costs, the total blasting cost (\( C_b \)) will be
obtained:

\[ C_b = \left[ \frac{0.507(D_e)(0.4h+0.9)}{B^2 \times h \times D_r} \right] \times \left[ \frac{\frac{D_c}{D_r} + 1.5}{0.0136h + 0.031} \right] \times h \times D_r \]  \hspace{1cm} (7)

where \( C_b \) is blasting cost ($/ton), \( C_e \) is explosive cost ($/kg).

Figure 5 shows the variation of blasting costs for different
bench heights, considering ANFO as the explosive in an iron
ore mine.

4. Loading operation
The size of loading equipment and specially loading
height correlate with bench/block height. Moreover, the size
of excavators deals with deposit size; so that the deposit
should be large enough to support the capital cost of large
equipment. However, in the situation of low investment or
high economic risk, companies may prefer to purchase
smaller equipment to increase the capability to overcome the
lack of capital investment.

Considering the correlation between block/bench height
and the bucket capacity of an excavator, the relation between
block dimension and loading cost was obtained.

4.1. Loading capital cost
The capital cost of excavators is dictated by the optimum
bucket size, number of loading equipment (\( N \)) and the tons of
ore and waste will be extracted daily. Figure 6 shows the
relation between the size of bucket and bench (block) height

Figure 6. Definition of the average bucket capacity based
on bench/block height [11]

Regarding Figure 6, the average bucket capacity of an
excavator (\( BC \)) in cubic meters can be determined based on
block height (\( h \) in meters) as:

\[ BC = 0.61 \times \exp(0.22 \times h) \]  \hspace{1cm} (8)

Substituting Equation 8 in O’Hara capital cost estimator
for a shovel, the capital cost of a shovel (\( C_{Lcap.} \)) can be
determined based on block height as:

\[ C_{Lcap.} = \frac{6193 \times T_p^{0.8}}{\exp(0.044 \times h)} \]  \hspace{1cm} (9)

where \( T_p \) is tons of material (ore and waste) which is mined
per day. Figure 7 illustrates the trend of loading capital cost
with block height, considering a constant \( T_p \).
4.2. Loading operating costs

Considering the U.S. cost estimation database, it is possible to ascertain the hourly operating cost based on bucket capacity as is shown in Figure 8. Operating cost of excavators consist of maintenance, fuel/power and lube, wear parts and overhaul costs. Furthermore, the tire cost was considered for wheel loader costs.

Large loading equipment may require more and complex components. Therefore, they are equipped with modern monitoring and control systems to overcome this complexity, which will increase operating costs. In addition, maintenance, repair, and operation of large equipment demand skilled and educated personnel, which will increase the total cost as well. However, the high productivity of these equipment leads to less cost per tonnage of mined material. The operating cost of loading operation for each ton of excavated material can be express as Figure 9.

5. Haulage costs

The size of haulage equipment directly effects the mine design parameters and productivity. The size of transportation machines should be also adequately matches to loading equipment. Moreover, the number of required equipment changes for different size of haulage and loading machines, considering a specific level of productivity. Due to the fact that in almost all open pit mines trucks are applied as mine transportation system, this research studies the variation of truck costs with block height.

5.1. Haulage capital cost

Since the optimum truck size in tones should be well matched with the bucket size of shovels, based on the relationship between bench (block) height and bucket size; it can be concluded that block height also affects the capital cost of trucks. Regarding the O’Hara cost estimator, the capital cost of the truck (C_{tap}) can be determined based on block height as:

\[
C_{tap} = \frac{4240 \times T_r^{0.8}}{\exp(0.024 \times h)}
\]
As it was mentioned, $T_p$ is the daily tonnage of extracted material (ore and waste).

### 5.2. Haulage operating cost

The same as loading operation, the costs of maintenance, overhaul, wear parts, fuel/power and lube as well as tire cost were collected for different truck capacities from InfoMine cost estimation database. The main difference between loading and haulage operating cost is tire consumption which has an extreme influence on overall truck operating cost. Tire cost for a truck is almost the same as its capital cost during its operation life [7]. Figure 12 shows how hourly operating cost increases for large trucks.

![Figure 12. Truck operating cost per hour for different truck capacity](image)

The life of tire in large trucks is less than the smaller trucks due to higher air pressure and operating temperatures. Thus, manufacturers have suggested lower speed for larger trucks. In addition, more requirements of road maintenance and less flexibility of large trucks also demand an extra cost. In spite of that, the high capacity of large trucks causes less operating cost per tonnage of transported material.

To convert the unit of cost from hour to tonnage, O'Hara equation was used to define the relationship between daily transported tonnage and truck capacity. Therefore, operation cost of trucks ($C_{\text{Topc.}}$) can be written based on truck capacity (TC) as follow:

$$C_{\text{Topc.}} = 2.24 \times (TC)^{-0.27}$$  \hspace{1cm} (12)

Figure 12 shows that the operation cost of each ton of material decreases exponentially by increasing the truck capacity.

Since truck capacity should be matched appropriately with shovel capacity, truck size can also be related to the block height. Accordingly, regarding the O’Hara equation of truck size and loading bucket capacity, the operating cost of haulage system is decreased in high blocks due to the equation (13).

$$C_{\text{Topc.}} = 8(h)^{-1}$$  \hspace{1cm} (13)

### 6. Conclusions

Several parameters should be considered to determine the effective block dimension included grade variability, geological continuity, equipment size, mining cost, dilution and slope stability. This paper shows the influence of different block heights on mining costs. The horizontal dimensions of each block are determined based on the slope stability conditions.

Correlating of blast-hole diameter and bucket capacity with bench (block) height made it possible to define capital and operating costs based on block height. The O’Hara cost estimator method was employed as a tool to find out the capital cost and daily production of material with respect to the size of equipment. Moreover, the InfoMine cost estimation database was used to define the detail operating cost of each size of drilling, loading and haulage equipment as well as blasting operation.

The results of cost evaluation for the major mining operation shows that the overall operating cost of one ton of excavated material decreases exponentially by increasing the block/bench height.

### References


Data imputing using genetic algorithms (GA)

A case study of cost data for tunnel fans

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Abstract—Data imputing uses to posit missing data values, as missing data have a negative effect on the computation validity of models. This study develops a genetic algorithm (GA) to optimize imputing for missing cost data of fans used in road tunnels by the Swedish Transport Administration (Trafikverket). GA uses to impute the missing cost data using an optimized valid data period. The results show highly correlated data (R-squared 0.99) after imputing the missing data. Therefore, GA provides a wide search space to optimize imputing and create complete data. The complete data can be used for forecasting and life cycle cost analysis.

Keywords: data imputing, genetic algorithms (GA), R-Squared.

1. Introduction

Data imputing uses to posit the existence of missing values to decrease the computational process, estimate model variables and derive the results that would have been seen if the complete data were used. The common practice is to impute the missing data using the average of the observed values. With imputing, no values are sacrificed, thus precluding the loss of analytic results [1].

Genetic algorithm (GA) is a widely used evaluation technique to optimize and predict missing data by finding an approximate solution interval that minimizes the error prediction function [2]. Several studies of imputing data have used GAs to understand and improve data to avoid bias in decision-making.

Ibrahim Berkan Aydilek et al. [3] proposed a hybrid approach that utilizes fuzzy c-means clustering with combination between support vector regression and a genetic algorithm. This approach used to optimize cluster size and weight factor and estimating missing values. The proposed clustering technique used to estimate the missing values based on the similarity and Root Mean Standard Errors (RMSE) used to estimate the imputing accuracy. The authors found that clustering makes missing value a member of more than one cluster centroids, which yields more sensible imputation results.

Mussa Abdella et al. [4] introduced a new method by combing genetic algorithm (GA) and neural networks to approximate the missing data in database. The authors use GA to minimize an error function derived from an auto-association neural network. They used a standard method (Se) to estimate the imputing accuracy of the missing data that investigated using the proposed method. The authors found that the model approximates the missing values with higher accuracy.

Daiheng Ni et al. [5] developed multiple imputation scheme that provides multiple estimates for a missing value, simulating multiple draws from a population to estimate the unknown parameter. This scheme develops a multiple imputation procedure and as a framework. The procedure consists of estimate multiple values for each missing value in time, quantify the variability of the multiply imputed data, then combing the imputed data values by taking the average. The authors found that scheme has high imputation quality and merits many advantages such as yielding unbiased estimates for the missing values, preserving the natural variability of the observed data, and providing a measure of the uncertainty introduced by missing data.

The aim of this study is to develop a GA to optimize imputing the missing cost data of tunnel fans. GA imputes the missing cost data to derive complete data for forecasting or life cost analysis. We argue that a multi-objective GA decreases the complexity, increases the flexibility, and is very effective when selecting an approximate solution interval for imputing.

2. Methods

2.1. Data collection

The cost data are for Swedish tunnel fans in Stockholm. The data were collected over ten years from the Swedish Transport Administration (Trafikverket) and stored in the MAXIMO computerized maintenance management system (CMMS). In CMMS, the cost data are recorded based on work orders of tunnel fans and contain labour cost. Due to company regulations, labour cost data are encoded and expressed as currency units (cu) for this study.

2.2. Genetic algorithm (GA)

GA is widely applied in imputing because of its ability to optimize valid imputing period in a large space of random populations [6]. The GA operates with a population of chromosomes containing data of work orders. The
chromosomes are proportional to the case and problem statement [7] as seen in figure 1.

GA is applied longitudinally to the data. GA operates with a population of chromosomes that contains labour cost. Forty percent of each cost object is selected randomly at two different times. Multi-objective GA operates on the selected population over different generations to find the appropriate cost range to impute missing data.

2.2.1. Initial population
A longitudinal study of labour cost ($Z_{labour}$) is used to impute missing data using GA.

2.2.2. First GA generation and selection
The first generation is done by randomly selecting 40% of the initial population of labour cost at two different times so that 80% of the data comprise different chromosomes. We select two chromosomes to validate the GA operators. We select 80% because a high range of missing data requires us to find an appropriate range of values to impute missing data.

We find the minimum and maximum for 40% of the population and for labour cost. We generate a random number $P_i$ between the minimum and maximum of labour cost. Data point $z_i$, $fitness(z_i)$ for each selection in labour cost is calculated based on the formula (1) [8]. Then, the chromosome pairs for each object are summed.

$$fitness(z_i) = \sqrt{\frac{1}{N} \sum_{i=1}^{N} (Y_i - P_i)^2}$$  \hspace{1cm} (1)

$N$ : number of training data
$Y_i$: actual value of ith training
$P_i$: generated value of ith training

The fitness function is a statistical method that finds the root mean absolute percentage error of the correlated imputed data. The lowest fitness function is selected to transform data to the next generation after the GA is applied.

2.2.3. Crossover and mutation
In this study, we use a two-point crossover with a fixed crossover probability. For chromosomes of length $l$, a crossover point is generated in the range $[1, 1/3 l]$, $[1/3 l, 2/3 l]$, and $[2/3 l, l]$. The values of labour cost are connected and should be exchanged to produce two new offspring. We select two points to create more value ranges and find the best fit.

Randomly fifty percent of the selected chromosomes undergo mutation with the arrival of new chromosomes. For the cost object values, we swap two opposite data values.

2.2.4. New population
The new population will repeat steps 2 to 3 continuously for 25 generations. Twenty-five generations are enough for these data because the fitness function increases after the fourth generation. The best fit population will have the lowest fitness function. The selected population is used to randomly impute new values between the selected ranges for the missing data for labour cost. This step will yield fully correlated data that can be used to forecast the future and optimize the system life cycle cost.

After this step, R-squared regression analysis is used to validate the imputed cost objects. R-squared gives information about the correlation between the cost values for labour cost before and after imputing.

3. Results and Discussion
The amount of missing data in the labour cost 56.84% as seen in figure 2. Missing data cause a substantial amount of bias, make the analysis of the data more arduous, and reduce analysis efficiency. GA is implemented to impute the missing data. The imputation will help to provide complete data that can be used for forecasting or life cycle cost analysis.
Figures 3 shows labour costs after imputing the missing data with optimal value ranges. The generated random values between the selected appropriate data ranges using multi-objective GA show a smooth trend with the existing contents. This data imputation may be sufficient for life cycle cost analysis, as we now have complete data after imputing.

R-squared regression analysis is used to validate the imputation process using multi-objective GA and to determine the differences before and after imputation for each cost object. Figure 4 shows R-squared is 0.847 for the redacted accumulated labour cost data, and Figure 5 shows a better R-squared value, 0.9926, after imputing the missing labour cost data. This means imputing increases the correlation of the labour cost values.

The time taken for imputing has fewer calculations and requires almost the same amount of time for each population 16 to 17 seconds, as the population size is always 80% of the cost data for each cost object.

4. Conclusions
This study develops GA that shows an optimal range of values to impute missing data. It can provide appropriate data ranges that accommodate data variations, and it shows minimum imputing errors. The correlation between cost data values after the missing data are imputed is high and relevant. The resulting complete data can be used for forecasting and life cycle cost analysis.

5. Future works
The future research possibilities are broad. Prospects can be drawn towards investigating a multi-objective GA to different domains such as clustering, imputing, forecasting and time-series analysis. Multi-objective GA uses in these domains based on different objectives and provides an efficient analysis over the tunnel fan turbine life cycle. Extracting the optimal knowledge aims to provide comprehensible data analysis model.

References


3D GM and resource estimation of Gol-e-Gohar iron mine No.3, Sirjan, Iran

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Abstract— Gol-e-Gohar Fe deposit is situated in Sanandaj-Sirjan metamorphic zone and in 55 km west of Sirjan City. A metamorphic complex with the age of Paleozoic has surrounded the Fe mineralization in this area. 6 Fe anomalies were found in this area by using of airborne geophysical exploration operations. This study has done on the Gol-e-Gohar Fe Deposit No.3. Modelling and resource estimation was performed on Gol-e-Gohar Fe Deposit No.3 using GEMCOM GEMS software. The data analysis includes Lithology, position openings, grade and dip has done on 142 boreholes. Then 19 north-south geological sections and 11 east-west sections have drawn. The geological sections selected vertical on mineralization and geological plans are drawn. Mineralization model was prepared based on GEMGEMCOM software and block model method. We considered code 201 for hematite zone, code 301 for magnetite with Sulphur less than 0.2 % and code 101 for magnetite more than 0.2 %. Finally, total resource was calculated 605 million tons for every 3 defined code of Gol-e-Gohar Fe Deposit No.3 by using block model and considering the average density of 4.14 g/cm³.

Keywords: modeling, resource estimation, GEMCOM GEMS, Gol-e-Gohar, Sirjan

1- Introduction

Gol-e-Gohar iron deposit is located in Sanandaj-Sirjan metamorphic zone, 55 kilometers west of Sirjan City. About six anomaly of iron have been identified in Gol-e-Gohar area by using geophysical exploration. These six anomalies with 1135 million tons resourced, are already one of the largest producers of iron ore Iran. There are many soft wares in the field of mining that the most famous professional multipurpose once are used in mineral exploration including Vulcan, GEMCOM GEMS, SURPAC, Datamine and Minex that the GEMCOM GEMS is one of the most powerful mining software among them which can perform three types of task: modeling and determining of mineral resources, designing opencast and underground mines, and monitoring on extraction and exploitation of resources.

A geological model is formed by a series of regular or irregular blocks that is attributed to each of them the features such as grade, type of rock and codes or other features [1]. Also, the factors such as variation in dimensions of blocks of a model or variation in the number of needles led to change the accuracy of calculations. In order to make an ore model, we serve both ways of solid (wired) model and mining block method. Resource estimation is one of the most important purposes of mineral exploration and just after this stage we can judge about deposit and assess its possibility of economic extraction. Also, estimation of ore resources, especially for mines in operation, is one of the most important achievements to designing and planning for production of ore. Generally, the methods of resources estimation can be divided into two groups that are classic and modern methods. In all of these methods, deposit has been divided into smaller blocks that their ore conditions are similar to each other and finally the total mineral resource is determined by multiplying together the volume, density and grade of each block. In fact, the basic difference between these methods is the manner of ore division into smaller blocks and the manner of calculation thickness, special weight and average grade of the blocks[3, 8].

2- Geology of area

Based on structural zoning map of Iran, the Gol-e-Gohar complex is located in the western margin of the Sanandaj-Sirjan zone, and edge of the Kheirabad depression, a salt pan which has been formed between Sanandaj-Sirjan zone and Urumieh–Dokhtar magmatic zone. (Refer to Figure 1 A, B) [5].
Figure 1. A.B, the location of under Sanandaj-Sirjan area is shown in tectonic map of south west and with changes [6] of the position of the area of study on the map.

The area of study generally has been covered by recent alluvial. Gol-e-Gohar iron deposit is located in the Paleozoic metamorphic complexes with lithology composition of Calc-Schist, Quartz-muscovite Schist, Garnet Schist, Amphibole Schist, Amphibolite, Graphite Schist and marble that have been subjected to Green schist to Amphibolite facies metamorphism. Also, Ar/K dating on amphibole and mica in Gneisses of Neyriz area confirmed the age Paleozoic (300 million years) for the complex. These complexes are the oldest stone unit in the area [7, 2].

The boundary between Paleozoic and late Triassic sedimentary sequences (reef limestone) and Jurassic (oolitic Limestone, Conglomerate) is in compliance with the early Cimmerian discontinuity that was located at the east of mine, [4]. There is a Flysch of Eocene-Oligocene including fragments of the various rocks with different ages, lies on the Mesozoic’s sediments. There is a sequence of recent sediments from Pleistocene to present on the mentioned Flysch formation. Gol-e-Gohar iron deposit has six mineral masses. Anomaly No. 3 of Gol-e-Gohar is being at a distance of 1000 meters from the mine No. 1. This Lens-shaped anomaly in terms of composition and grade is divided into three sections: The upper Magnetite, Oxide Magnetite and lower Magnetite. In the upper Magnetite, sulphur content in the form of Pyrite, Chalcopyrite and Pyrrhotite is low, but in lower Magnetite, Sulphur content is high. The Magnetite Oxide has more iron Oxide rather than two other sections. The host rock of this deposit is mainly including the metamorphic rocks with probable age of Ordovician-Silurian. The basement includes Gneiss, Quartz-Chlorite Schist, Quartz-Muscovite Schist and Amphibolite.

Figure 2. The geology map of area of study (1:100000).

3- Material and Methods

We used the data from 142 exploratory boreholes in the area to modeling and resource estimation of Gol-e-Gohar
anomaly No3 in Sirjan. This data includes: the location of boreholes, inclination of boreholes, lithology along boreholes and the grade. 1164 samples containing ore are gathered that include models resulting from exploratory boreholes were transferred to Gol-e-Gohar mining laboratory after samples. Primarily, the grade of four effective elements in iron resources in each model was determined at the laboratory that are Fe, FeO, P, S and then modeling and resource estimation was accomplished by GEMCOM GEMS software during 5 stages including: preparation of input data in form of a separate Excel file, processing of data, draw geological sections, planning geological sections and block model.

4. Results and Discussion

4-1. Drawing geological sections and showing logs

The foundation of three-dimensional modeling of mineral resources by using mineral specialized soft wares manually is drawing geological section [8]. Also, in order to make a geological model of Gol-e-Gohar Fe Deposit No.3, first, it is necessary to draw geological section at specified intervals; so, the sections were prepared in GEMCOM software at first. These sections were prepared concerning the extent of mineral. Therefore, 19 north-south cross-section and 11 east-west cross-section of ore were prepared at intervals of 51 meters from each other figure 3. It should be noted that these sections in the northern part of the mine concerning denser drilling density and consequently with fewer errors than the southern part of the mine have been prepared.

4-2. Planning geological sections

Traverse for sections was prepared after drawing geological sections using data from boreholes in software and the related geological plans were prepared using their Traverses. A model of prepared plans in software was shown in figure 4.

4-3. Making the model of mineral

A geological model is made of a group of regular or irregular 3D which each of them have the features such as grade, the type of ore, codes or other elements [1]. Also these blocks should be in a form which are useful in designs. Concerning studies done in Gol-e-Gohar Fe Deposit No.3 best dimensions of block was considered as 15*25*25. Also in modeling of Gol-e-Gohar Fe Deposit No.3, borehole plans have been blocked with 100 meters depth separately from borehole plans higher than 100 meters to set a resource estimation separately too. Concerning dimensions of blocks and above coordinates, the model of considered block, in Gol-e-Gohar Fe Deposit No.3 has 113 columns, 11 rows and 45 balances.

After blocking deposit plans elevation were added together by height and finally, the Gol-Gohar ore body No.3 is modeled by GEMCOM software. Which Based on this model, the overall shape is a flat lens-shaped mass with a trend of N-S, and a down going plunge to the south. But In The southern part, some faults have caused an upwards
movement in the ore body. This mine has an area with a maximum length of 2400 meters and a width of 2000 meters.

Figure 5. A: gives an overview of blocks stored in the mine model was developed to estimate the Gol-e-Gohar Fe Deposit No.3. B: Gol-e-Gohar Fe Deposit No.3 final three-dimensional model using GEMCOM software.

4-4 Resource calculation

To determine storage after preparing and processing the data file on them, examples of high iron grade of 20% from which had codes 101, 201 and 301 were prepared geological sections and the sections of the horizontal plan was drawn. And then has been blocked and finally according to the codes and plans were stored in the specified range. The general minerals, plans were based on was estimated about 605 million tons. It should be noted that in our study, the average density of ore body is considered to be 4.14 g/cm$^3$.

5. Conclusions

Ore Mineral of Anomaly No.3 of Gol-e-Gohar has a lens shape and that geological sections, horizontal plans and block method was used. Also, after making the blocky model using GEMCOM software, the probable mineral resource of Gol-e-Gohar anomaly No. 3 was estimated for each three codes defined according to the average density of 4.14 g/cm$^3$ 605 million tons.

Acknowledgement


References

A Simulation-based Optimization Approach for Material Dispatching in Continuous Mining Systems

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Abstract—This paper examines a problem related to dispatching materials to spreaders in coal (lignite) mines operated under the paradigm of continuously excavated material flow. In the real world, complexity in analyzing such systems emerges from numerous factors, including several random variables and frequent changes of extracted materials. Most of the mathematical programming approaches are limited by the amount of the decision variables. Indeed, simplifying assumptions should be made about these factors to develop a manageable mathematical model. In this study, a new simulation-based optimization approach is proposed that can accommodate most of these factors. This approach consists of running alternately a deterministic optimization model and a stochastic simulation model. It combines simulation, transportation problem, and job-shop scheduling problem. The transportation problem provides a mechanism to optimize dispatch decisions. In other words, it finds optimal connections between excavators and spreaders. Because of the nature of the transportation problem, it is possible to have multiple connections for an excavator. Therefore, the job-shop scheduling problem deals with the allocation of spreaders to different excavators over time. Its objective is to find the processing sequences and starting times of each operation on each spreader, in order to minimize the total weighted tardiness. Finally, the simulation uses the dispatch decisions generated by optimization and computes particular performance indicators. The calculated values are then introduced into a control module. The control module suggests refinements to parameters of the optimization model (e.g. transportation costs, jobs order, and jobs weight). The iterative process ends after a stopping criterion is met. The proposed approach is tested on a large continuous mine under given different dumping sequences, and results are reported. The merits and limitations of the proposed approach as pinpointed and farsighted operations management are discussed.

Keywords: simulation-based optimization, dispatching, opencast coal mining, mining, short-term scheduling.

1. Introduction

Short-term production scheduling of a continuous mining system generates a sequence of extraction and dumping operations over time within a predefined production plan. This schedule is seen as the operational guide to meet the long or medium-term objectives of the mine developed under current operating conditions and constraints. It outlines extraction and dumping stages in terms of weeks or days. The optimization of short-term production scheduling is guided by the life-of-mine or long-term mine planning and it is conventionally optimized in two distinct steps, (Hustrulid and Kuchta, 2006). The first step optimizes the physical sequence of extraction of materials. The second step optimizes the dispatch decisions based on the dumping sequences, equipment capacity, performance, and availability. The focus of this study is on the second step of the optimization.

In the real world, there are limitations to the above mentioned distinct optimization steps, which may result in non-optimal or infeasible short-term production schedules. First, uncertainty in input parameters is not considered in the optimization steps. Second, the optimization of the extraction sequence of material ignores operational considerations and equipment availability, and thus can be unrealistic. Lastly, Most of the mathematical programming approaches are limited by the amount of the decision variables. Indeed, simplifying assumptions should be made to develop a manageable mathematical model. The performance of the production scheduling can be adversely affected by these limitations and this may lead to: (a) increased operating costs due to the unscheduled downtimes; (b) uncertainty in equipment performance and lower utilization of equipment; and (c) inability to meet expected production targets. This paper proposes a new simulation-based optimization approach that can accommodate these limitations. This approach consists of running alternately a deterministic optimization model and a stochastic simulation model. It combines simulation, transportation problem, and job-shop scheduling problem. The transportation problem provides a mechanism to optimize
dispatch decisions. In other words, it finds optimal connections between excavators and spreaders. Because of the nature of the transportation problem, it is possible to have multiple connections for an excavator. Therefore, the job-shop scheduling problem deals with the allocation of spreaders to different excavators over time. Its objective is to find the processing sequences and starting times of each operation on each spreader, in order to minimize the total weighted tardiness. Finally, the simulation uses the dispatch decisions generated by optimization and computes particular performance indicators. The calculated values are then introduced into a control module. The control module suggests refinements to parameters of the optimization model (e.g. transportation costs, jobs order, and jobs weight). The iterative process ends after a stopping criterion is met. The proposed approach is tested on a large continuous mine under given different dumping sequences, and results are reported. The merits and limitations of the proposed approach as pinpointed and farsighted operations management are discussed.

In the following sections, a literature review and a brief background about continuous mining systems will be given. It continues by defining the problem. Then, the solution strategy is discussed in detail. After that, the computational framework and its implementation are presented. Finally, the description of a real-size case study is given and the obtained results are reported. The last section concludes the findings of this study.

2. Review of Literature

A considerable amount of literature has been published on the optimization of short-term production scheduling. These studies in early attempts have focused on evolving concepts and related formulation for finding extraction sequences based on mathematical programming, e.g. (Wilke and Reimer, 1977, Wilke and Woehrle, 1980, Gershon, 1983). Their objective function is set to minimize production deviations from the long-/medium-term production targets. While allocating resources, the conventional optimization process considers mining direction and fleet capacity. Nevertheless, it does not integrate the fleet management, i.e. dispatching of mining equipment and uncertainty in equipment availability. More recent attentions thus focus on the provision of real-time fleet allocation for short-term production scheduling (Alarie and Gamache, 2002, L’Heureux et al., 2013) and stochastic optimization of short-term production scheduling (Topal and Ramazan, 2012, Matamoros and Dimitrakopoulos, 2016). They have been successfully applied for over three decades to find optimal solutions for real size case studies. However, a large and growing body of literature has mainly investigated the applications that are in the discontinuous block mining with the diffuse deposits.

In simulation-based optimization literature, Figueira and Almada-Lobo (2014) review the current research on its developments. They provide a taxonomy that gives an overview of the full spectrum of current simulation-optimization approaches. In the field of mining, little research to date has been carried out on this area. Mena et al. (2013) presented a simulation-optimization modeling framework based on the Arena and the LINGO software. Their objective is to maximize the overall productivity of the fleet in a truck-shovel system. In their model, trucks are allocated to transportation routes according to their operating performances. Improvements when using in realistic case are reported. Nageshwaraniyer et al. (2013a) investigated a two-level hierarchical simulation-based planning framework for a coal mining system consisting of multiple pits, using trucks and trains to transport the material and silos to blend it. The Arena software is used for building the simulation model. The problem is divided into sub-problems reducing the decision space of the complex problem. The train-loading problem is at the top level while the machinery-scheduling problem (defining working schedules for machinery at the mine for loading each customer train) is at the lower level. To maximize the revenues of the mine in each shift, the problems are solved using the OptQuest software. Nageshwaraniyer et al. (2013b) proposed a robust simulation-based optimization approach for a truck-shovel system in surface coal mines. It is based on a detailed simulation model created with the help of the Arena software. Response Surface Method (RSM) is applied to derive an expression for the variance of revenue. Their objective is the maximization of the expected value of revenue obtained from customer trains.

Since there are few applications of the simulation optimization approach in mining, it seems wise to focus on related fields, such as process system engineering and supply chain management, to build upon their findings. A supply chain management problem under demand uncertainty was presented by Jung et al. (2004) whereby safety stock levels were determined using simulation-based optimization method in a rolling horizon manner. Their proposed approach consists of running alternately a deterministic planning model and a stochastic Monte-Carlo based simulation model in a loop structure. Their algorithm ends when the difference between the estimation and the target values of the customer satisfaction level is equal to very small number.

3. Background

Continuous mining systems, usually known as opencast mines, consist of excavators, belt conveyors, and spreaders operating in series and under the paradigm of continuously excavated material flow. Figure 1 shows a schematic section view of a continuous mining system. The operation starts with the excavation of materials by excavators at the extraction side. It continues by the transportation of the extracted materials from mining benches to dumping benches or a coal bunker. The transportation process includes a network of conveyor belts consisting of face conveyor belts, main conveyor belts, and a mass distribution center. Finally, lignite is stacked at the bunker or waste materials are dumped at the dumping side. In such a paradigm, the excavators can be seen as supply points and the spreaders together with the coal bunker can be considered as demand points.
The production planning in an opencast mine covers various periods, namely long-, medium-, and short-term planning horizons. The long-term planning affects an opencast mine across its entire life, all the way to the end of mining supervision after the land reclamation. The medium-term planning often covers the next five-year period. Finally, the short-term planning is a yearly seam-focused detailed plan.

Besides operational and economical parameters that are necessary for any production planning process, the major input here is the geological block model. It is divided into two separate block models, namely, the extraction block model and the dumping block model. The former includes the geological strata, quality parameters, volumes-tonnages, and material types. The latter includes dumping profiles and volumes.

As mentioned earlier, the short-term plan is guided by medium and long-term planning. Forasmuch as the complex deposit formations require selective mining of coal as well as overburden on different benches. The objective of short-term planning is to find the sequence of blocks, known as extraction sequence, that meet the defined targets under current operating conditions and constraints. After the creation of the extraction sequence, basically, the first step of the optimization of the short-term scheduling is completed. The created extraction sequence can be used as a guide to create the dumping sequence. It is also an input for the second step of the optimization, which is the focus of this paper. The next section describes the problem with the defined objectives.

### 4. Problem Description

Figure 2 presents the flow diagram of the short-term production scheduling in continuous mining systems. Three major processes can be seen in the diagram namely, short-term planning, dumping sequence creation, and material dispatching. These should be completed in the presented logical order to have a short-term schedule. Here, there are two underlying assumptions; the first is that the extraction block model, the dumping block model, and the extraction sequence are given as discussed in the previous section. Stable dump construction needs different material types with special sequences; while these materials are distributed unevenly at the extraction side, the second underlying assumption becomes very important. It is defined as that the problem should be relatively a balanced problem. In a sense, the difference between the total amounts of different overburden materials at the extraction side with the amounts of available spaces at the dumping side should be a small number. In the presence of finite available space for a material type, when the extraction of that material type becomes sufficiently large, then for any given dumping sequence it will no longer be possible to meet the defined production targets. The optimization of dispatch decisions thus must involve the dumping capacity constraints. Furthermore, uncertainty is associated with input parameters, equipment availability, and their performances thus the resulting problem is a constrained stochastic optimization problem.

The different ranges of the ratio of the expected amount of materials at the extraction side to the dumping capacities of the same materials give rise to three different scheduling scenarios. In scenario I, when the extracted to dumped capacity ratio is sufficiently small, the dumping side has sufficient spare capacity to cope with abrupt changes in the extracted materials due to the uncertainty involved. Therefore, in this scenario, challenges are mostly located on the optimization of the dispatch decisions. In scenario II, characterized by an intermediate range of the extracted to dumped capacity ratio, the production capacity may be quite constrained by the dumping capacity when the extraction of different materials spike at some point in time. In this scenario, even with optimal dispatch decision, the production for some excavators may fail to reach their targets due to the downtimes. Finally in scenario III, the extracted to dumped capacity ratio is sufficiently large that most of the extracted materials simply cannot be dumped and thus excavators will compete for dumping spaces. In this scenario, dispatch decisions and dumping spaces must be assigned strategically to meet the demands of some excavators in preference to others. In this paper, the optimization problem that is under scenario I and II will be addressed. The
optimization of short-term scheduling for the case of scenario III involves strategies for the prioritization of excavators. Such strategies, while of considerable interest, are beyond the scope of this study.

The objective is to minimize downtimes of equipment by effective resource allocations. This will result in decrements in overall costs, including extraction costs, dumping costs, and penalties for deviating from the predefined targets. There are two types of decisions, on the excavator and on the spreader side:

- Decision on the excavator side:
  - Production rate of each excavator (between 0% and 100%)
  - Connection to the spreader
- Decision on the spreader side:
  - Dumping sequence (depending on material type available)

5. Solution Strategy

To address the above-mentioned problem this paper proposes a new simulation-based optimization approach that relies on the use of deterministic optimization model and a stochastic simulation model. The deterministic model is built using a certain feasible dumping sequence and incorporates transportation problem and job-shop scheduling problem. The transportation problem provides a mechanism to optimize dispatch decisions. In other words, it finds optimal connections between excavators and spreaders. Because of the nature of the transportation problem, it is possible to have multiple connections for an excavator. Therefore, the job-shop scheduling problem deals with the allocation of spreaders to different excavators over time. Its objective is to find the processing sequences and starting times of each operation on each spreader, in order to minimize the total weighted tardiness. A discrete event simulation of the system is executed implementing the dispatch decisions obtained via the deterministic model for a given dumping sequence. The results of multiple simulation replications serve to provide an estimate of a particular performance measure (e.g. utilization). The calculated values are then introduced into a control module. The control module suggests refinements to parameters of the deterministic optimization model (e.g. transportation costs, jobs order, and jobs weight). The iterative process ends after a stopping criterion is met. The strategy uses two aspects of “Sim-Opt” architecture, (Subramanian et al., 2001). Figure 3a-b presents the configuration of the discussed simulation optimization approach.

The following will discuss the three key sub-problems, the creation of random dumping sequence, the transportation problem, and the job-shop scheduling problem. In the
subsequent section, the various computational details that are needed to link these sub-problems and to drive the computations to obtain the desired short-term plan will be discussed.

5.1. Random Dumping Sequences

If the dumping benches with their special profiles were discretized in a defined sections (e.g. every 50m), then the evolution of the random dumping sequences over time can be represented by the tree-like structure presented in Figure 4. Starting from each node, a large number of possible dumping options at the next dumping stage are expressed as branches stemming from that node. Assuming $m$ possible next-stage dumping options at each node, the total number of scenarios will amount to $m^S$, where $S$ is the total number of dumping stages. Each scenario as a feasible dumping sequence is an input for the transportation problem as is shown in Figure 3b.

5.2. Transportation Problem

The transportation problem (TP) is concerned with shipping a commodity between a set of sources (e.g. excavators) and a set of destinations (e.g. spreaders). Each source has a capacity dictating the amount it supplies and each destination has a demand dictating the amount it receives, (Winston and Goldberg, 2004). TP is a subset of network models and the set of resources and destinations can be illustrated, respectively, by a set nodes. Nodes are connected to each other via arcs; each arc has two major attributes namely the cost of sending a unit of a material from one node to the others and the maximum capacity of the arc, Figure 5.
An open-cast mine extracts material at \( m \) different benches \((i = 1, ..., m)\). The amount of material to be extracted at bench \( i \) is \( a_i \). The demands for the extracted materials are distributed at \( n \) different dumping benches \((j = 1, ..., n)\). The amount of material to be dumped at bench \( j \) is \( b_j \). The problem is to find connections between excavators and spreaders at minimum cost. Linear programming (LP) formulation of the problem is as follows, (Winston and Goldberg, 2004):

**Objective function:**

\[
\text{Minimize } z = \sum_{\text{all arcs}} C_{ij} X_{ij} \tag{1}
\]

\[
\text{s.t. }\]

\[
\sum_{j=1}^{n} X_{ij} \leq a_i \quad \text{for } i = 1, ..., m \tag{2}
\]

\[
\sum_{i=1}^{m} X_{ij} \geq b_j \quad \text{for } j = 1, ..., n \tag{3}
\]

\[
X_{ij} \geq 0 \quad \text{for all } i \text{ and } j, \tag{4}
\]

where, \( X_{ij} \), number of units of materials sent from node \( i \) to node \( j \) through arc \((i, j)\); \( C_{ij} \), cost of transporting one unit of material from node \( i \) to node \( j \) via arc \((i, j)\). The objective function, denoted by Eq. (1) involves a deterministic optimization in which the total cost of sending materials from supply points to demand points is minimized. In constraint (2), the sum of all shipments from a source cannot exceed the available supply. Constraint (3) specifies that the sum of all shipments to a destination must be at least as large as the demand. Constraint (4) is a binding constraint.

Consider the feasibility of the problem. The only way that the problem can be feasible is if total supply exceeds total demand \( (\sum_{i=1}^{m} a_i \geq \sum_{j=1}^{n} b_j) \). Two conditions can be implied from this:

- When the total supply is equal to the total demand (i.e. \( \sum_{i=1}^{m} a_i = \sum_{j=1}^{n} b_j \)) then the transportation model is said to be balanced.
- A transportation problem in which the total supply and total demand are unequal is called unbalanced. If there is excess demand, a dummy source is introduced (i.e. a fictitious bench). The amount shipped from this dummy source to a destination represents the shortage quantity at that destination. If there is excess supply, a dummy destination is added to the network. Likewise, the amount received from this dummy destination from a source represents the excess quantity at that source.

Due to the nature of the transportation problem, it is possible that an excavator has to send materials to multiple spreaders. The next section will discuss the job-shop scheduling problem, which deals with the allocation of spreaders to different excavators over the time.

### 5.3. Job-Shop Scheduling Problem

The job-shop scheduling problem (JSP) consists of a finite set of jobs \( J = \{1, ..., n\} \) and a finite set of machines \( M = \{1, ..., m\} \). In this paper, excavators are defined as jobs and spreaders are defined as machines. The aim is to find a schedule of \( J \) on \( M \) under the below mentioned conditions:

- For each job \( j \in J \), a list \( (O_{j}^1, ..., O_{j}^k, ..., O_{j}^{m}) \) of the machines which represents the processing order of \( j \) through the machines is given. Note that \( O_{j}^k \) is called the \( k\)-th operation of job \( j \) and \( O_{j}^{m} \) is the last operation of job \( j \).
- The processing order for each job is fixed, thus, machine-sequencing problem for every job should be taken into account.
- For every job \( j \) and machine \( i \), a non-negative \( P_{ij} \) is given, which represents the processing time of \( j \) on \( i \).
- Each machine must always be available and can process at most one job at a time, and once a job starts on a given machine, preemption is not allowed.
- Every job \( j \) has an assigned release time \( r_j \geq 0 \) so that the first operation cannot start before \( r_j \). In this paper, \( r_j \) is given in the task schedule.
- An additional attribute of a job \( j \) is its weight \( w_j \), which represents the relative importance of \( j \) in comparison to other jobs.
- Furthermore, every job \( j \) has a due date \( r_j \geq 0 \) which should, but does not necessarily have to, be met in a schedule.

In this study, the objective is to minimize the obtained total weighted tardiness, as defined \( TWT = \sum_{j=1}^{n} w_j \cdot t_j \), where \( t_j = \max\{0, c_j - d_j\} \) is the resulting tardiness of job \( j \) in a schedule, and \( c_j \) its completion time. From then, this problem is referred to as JSPTWT. (Ku and Beck, 2016) investigated the size of problem that can be solved by Mixed Integer Programming (MIP) formulation. For a moderate sized problem up to 10 jobs and 10 machines, with the recent technology, MIP finds the optimum solution in a very reasonable amount of time. They also compared the performance of the four MIP models for the classical JSP. They concluded that the disjunctive MIP formulation with the use of Gurobi v6.0.4 solver (Gurobi Optimization, 2016) gives the fastest result for a moderate sized problem. The below is the disjunctive MIP formulation of JSPTWT is based on Manne (1960)’s formulations. The decision variables are defined as follows:

- \( X_{ij} \) is the integer start time of job \( j \) on machine \( i \)
- \( Z_{ijk} \) is equal to 1 if job \( j \) precedes job \( k \) on machine \( i \)

**Objective function:**

\[
\text{Minimize } \sum_{j=1}^{n} w_j \cdot t_j \tag{5}
\]

\[
\text{s.t. } \]

\[
X_{\sigma_{h,j}} \geq X_{\sigma_{h-1,j}} + P_{\sigma_{h-1,j}'} \quad \forall j \in J, h = 2, ..., m \tag{6}
\]

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\[ X_{ij} \geq X_{ik} + P_{ik} - V \cdot Z_{ijk}, \quad \forall j, k \in J, j < k, i \in M \]  
(7)

\[ X_{ik} \geq X_{ij} + P_{ij} - V \cdot (1 - Z_{ijk}), \quad \forall j, k \in J, j < k, i \in M \]  
(8)

\[ t_j \geq X_{mj} + P_{mj} - d_j, \quad \forall j \in J \]  
(9)

\[ t_j \geq 0, \quad \forall j \in J \]  
(10)

\[ X_{ij} \geq r_j, \quad \forall j \in J \]  
(11)

Constraint (6) is the precedence constraint. It ensures that all operations of a job are executed in the given order. The disjunctive constraints (7) and (8) ensure that no two jobs can be scheduled on the same machine at the same time. \( V \) has to be assigned to a large enough value to ensure the correctness of (7) and (8). In this paper, it is defined as \( V = \sum_{j \in J} \sum_{m \in M} P_{ij} \). Since the completion time of any operation cannot exceed the summation of the processing times from all the operations, Constraint (9) and (10) measure the resulting tardiness of each job. Finally, constraint (11) ensures that a job cannot start before its release time, and thus, captures the non-negativity of the decision variables \( X_{ij} \).

As an example, Figure 6 shows a simple JSP that three jobs J1, J2, and J3 are to be scheduled on three machines M1, M2, and M3. The graph on the top represents the precedence constraints. The Gantt chart on the bottom displays a feasible schedule that satisfies the precedence constraints.

![Figure 6](image)

6. Computational Framework

In this section, the overall computational approach is described. First, input parameters are explained. Then, the computational logic with the details of the integration of the sub-problems together and with the discrete event simulation is discussed. After that, the simulation based optimization framework is presented.

6.1. Input Parameters

The second step of the optimization of short-term scheduling starts with the assignment of input parameters. The definitions and their functionalities are as follows:

- **Start points of dumping in different benches**, i.e. the start locations of spreaders on benches at the beginning of the working shift. This is an input for the creation of random dumping sequences.
- **The allowed range of movement** for spreaders, i.e. in what range it is allowed to transport spreaders and start a new dumping profile. This is also an input for the creation of random dumping sequences.
- **Transportation costs**, these costs are used to distinguish between different destinations for a source in the transportation problem.
- **Machine sequencing**, the Earliest Due Date (EDD) sequencing method is used to create processing orders of the jobs in the JSP.
- Finally, **Job weights** are some other input parameters for the JSP. For instance, they can be used to prioritize an excavator if a bottleneck is seen after the simulation.

The aim is to find the best combination of these parameters using simulation based optimization approach to achieve the optimum short-term schedule.

6.2. Deterministic Optimization with Embedded Simulation

The following describes the details of the integration of the sub-problems together and to the discrete event simulation in walk-through steps.

**Step 1**: start with arbitrary set of input parameters.

**Step 2**: create a sufficient number of random dumping sequences, \( \{1, \ldots, R_s\} \).

**Step 3**: for a certain dumping sequence, \( d = 1, d \in R_s \), optimal connections can be found using the transportation problem.

**Step 3.1**: check the availability of the equipment based on the given task schedule and create the nodes.

**Step 3.2**: start with first blocks in the given extraction sequence and assign their volumes as \( a_i \) to supply nodes in the TP formulation.

**Step 3.3**: assign the volumes of the first sequence of blocks in the given dumping sequence as \( b_i \) to demand nodes in the TP formulation.

**Step 3.4**: check if problem is balanced, if not dummy nodes will be added to the network.

**Step 3.5**: create arcs between supply and demand nodes. Only the nodes get connected that have the same type of material.

**Step 3.6**: add a capacity to the arcs. In the TP, the capacity is set to be infinite for all the arcs.

**Step 3.7**: add costs to the arcs. In an opencast mine, the potential costs can be:

- Excavators and spreaders on the same level (altitude) get lower cost of transportation.
- Length of belt conveyors between supply nodes and demand nodes, the closer the equipment the lower the costs.
• Difference between the production capacity and dumping capacity of the equipment, the lower the difference the lower the costs.

Step 3.8: build the LP model with the help of Eqs. (1)–(4) and solve it by GUROBI solver.

Step 3.9: calculate the residual volumes and add them to the next iteration of the optimization.

Step 3.10: go to the step 3.1 and repeat steps 3.1–3.10 until all the blocks are extracted in the given extraction sequence.

Step 3.11: check for feasibility of the schedule, if there is residual volume left on the extraction side, set \( d = d + 1 \) and go to the step 3 until \( d = R_x \).

Otherwise, continue.

Step 4: create the input for the JSPTWT and build MIP model using Eqs. (5)–(11) and solve it by GUROBI solver.

Step 5: create the Gantt chart. The output of the JSPTWT is the optimum short-term schedule for the given extraction and dumping sequence \( d \).

Step 6: run the discrete event simulation for the given short-term schedule.

Step 7: record the state (utilizations, amounts) at the end of the time horizon.

Step 8: set \( d = d + 1 \) and go to the step 3 until \( d = R_x \).

6.3. Simulation based optimization framework

A more detailed flow diagram, which summarizes the overall computational framework, is presented in Figure 7. It combines the deterministic optimization with the stochastic simulation in a closed loop. Most of the steps are explained in detail in the previous section. As can be seen, the simulation is implicitly built over the embedded optimization. Once the computations over the simulation loop are completed, a number of best schedules based on the user-defined targets such as shorter makespan, higher utilization of equipment are selected. These are analyzed in the control modules; if the stopping criteria are met, the algorithm stops otherwise a new set of input parameters are introduced to the optimizer. The following section presents more details about the interactions between the components over a simulation-optimization platform.

7. Implementation of the computational framework

The implementation of the proposed simulation-based optimization approach consists of following major components: the computational control module, the databases, the three modules for the creation of random dumping sequences, transportation problem and job-shop scheduling, the discrete event simulation with its interface, the post-processing module, and finally the control module, Figure 8. The computational control module is responsible for controlling interactions of computational components. It has various functionality including:

- Issuing commands for retrieving information from the database.
- Generating/updating and releasing commands for executing the steps of the algorithm.

- Re-processing and controlling the output of each computational component before issuing the next command.
- Selecting a number of best schedules based on the defined criteria to proceed the algorithm to the simulation part.

The database contains information about geological block model, the given extraction sequences, and the task schedule. These data are stored in a Microsoft Excel Workflow file. Since the computational control module is coded in Python, a publicly available Pandas library (McKinney, 2010) is used to access each cell in the Excel sheets. Big datasets can be readily read and stored in DataFrames with the help of Pandas library.

The three major components of the deterministic optimization were explained in detail in the previous sections. It should be noted that to solve the LP or the MIP models, GUROBI Python interface is used. After the selection of a number of best schedules by the computational control module,
Figure 8. Simulation-optimization platform.

The data are recorded in two separate databases, namely, the block model and the schedule. These two are the major inputs for the discrete event simulation of an opencast mine.

The discrete event simulation model is built in Arena® simulation environment. The detail of the construction of the simulation model of an opencast mine can be found in Shishvan and Benndorf (2016). A process worth attention is the simulation model interface, which is situated between the computational control module and the Arena. This process is required because there is no direct way to interact with the Arena using, for instance, the command line. Instead, the program relies on automation via Visual Basic for Applications (VBA), a Microsoft technology for creating interconnection between applications. This technology is not available in Python, thus the simulation model interface is written in Visual Basic and compiled as an executable that can be controlled and run with the relevant parameters. When the simulation run is completed, the VB script releases a command to the controller.

The post-processing module processes the simulation outputs and creates plots and tables. Finally, the control module calculated the differences between the current results with the predefined targets. If another loop of simulation-optimization is required, the new input parameters are suggested to the computational control module.

8. Case study

To demonstrate the performance of the proposed simulation based optimization approach, a real case study has been developed. The Hambach mine is a large opencast coal mine and produces over 100 million tons of coal and over 500 million m³ of overburden materials per year.

8.1. Case Problem

8.1.1. Overview of the Production System

A schematic view of the Hambach mine is shown in Figure 9. In total eight bucket-wheel excavators (BWEs) have to be scheduled to serve continuously seven spreaders with waste materials and two bunkers with coal. Each BWE excavates either coal or waste in terrace cuts and transfers materials to the face conveyor belt, which carries it along the bench to the main conveyor belt. All excavated materials of the eight benches are distributed to their destinations at the mass distribution center. Based on a predefined daily schedule, waste is distributed to the seven spreaders for dumping, and lignite is forwarded to two coal-bunkers. Table 1 shows the technical specifications of the BWEs.
8.1.2. Problem in Mining

Waste materials at the Hambach mine are categorized in three types of mixed soils, dry mixed soils type 1 (M1), semi-wet mixed Soils type 2 (M2T) and wet mixed soils type 2 (M2N). The extraction of M2 type materials is increasingly facing deficiencies in output due to difficult mining materials. This type of soil, specifically M2N, exhibits a high share of cohesive components and is difficult to drain. M2N material cannot be used for stable dump construction and needs to be filled in between prebuilt polders constructed of dry material (Figure 10). The fact that only a limited quantity of these unstable mixed soils can be placed in the waste dump causes downtimes and bottlenecks in the placement process on the dumping side.

Furthermore, historical data show that next to scheduled maintenance, breakdowns of the equipment occur in a random manner. Due to the “in series” system configuration, equipment units feeding or are connected to the ceased equipment are blocked and set out of the operation while the maintenance is

---

**Table 1. Technical specification of BWEs.**

<table>
<thead>
<tr>
<th>Bench</th>
<th>BWE model</th>
<th>Discharge per min</th>
<th>Bucket capacity (m³)</th>
<th>Theoretical capacity (m³/h)*</th>
</tr>
</thead>
<tbody>
<tr>
<td>S1</td>
<td>259</td>
<td>44</td>
<td>2.6</td>
<td>5700</td>
</tr>
<tr>
<td>B1</td>
<td>260</td>
<td>38</td>
<td>3.5</td>
<td>5700</td>
</tr>
<tr>
<td>B2</td>
<td>291</td>
<td>48</td>
<td>5.0</td>
<td>12500</td>
</tr>
<tr>
<td>B3</td>
<td>287</td>
<td>43</td>
<td>5.1</td>
<td>10400</td>
</tr>
<tr>
<td>B4</td>
<td>290</td>
<td>48</td>
<td>5.0</td>
<td>12500</td>
</tr>
<tr>
<td>B5</td>
<td>292</td>
<td>48.6-72.0</td>
<td>5.0</td>
<td>12500</td>
</tr>
<tr>
<td>B6</td>
<td>293</td>
<td>48.6-72.0</td>
<td>5.0</td>
<td>12500</td>
</tr>
<tr>
<td>B7</td>
<td>289</td>
<td>48</td>
<td>5.0</td>
<td>12500</td>
</tr>
</tbody>
</table>

* 19.3 hours per day

The mine operates 24 hours per day and seven days per week. Regular maintenance is carried out on weekly, monthly, and annually based schedules. During the regular maintenance or an unscheduled breakdown, the production process ceases on the bench.
being done or the failure is being repaired. In addition, because of the multi-layer nature of the deposit, changes from one material type (e.g. M1) to another material type (e.g. M2N) or vice versa happens very frequently. Each time a material change takes place, the BWE stops excavating. The combined effect of random equipment breakdowns and frequent changes in extracted materials, makes the prediction of the exact material flow rate at any given future time span as a major source of uncertainty.

The objective is to optimize dispatch decisions to decrease downtimes/increase efficiency of excavators and spreaders by effective resource allocation while ensuring stable dump construction using the proposed simulation based optimization approach. Here, decisions on the dumping side are the length of polders to be built while on the extraction side, decisions are production rates of excavators and their connections to spreaders.

8.1.3. Input Data

The following presents the data for a day of production. The objective is to find the optimum daily schedule that maximizes the utilization of equipment.

Extraction Sequence

Figure 11 shows the given extraction sequence for one day of production. This model is also called the slice block model.

Task Schedule

The given task schedule for a day is presented Table 2. The number “0” denotes that the equipment is unavailable and “1” vice versa. Thus, the transportation problem will have six supply nodes and eight demand nodes.

<table>
<thead>
<tr>
<th>Bench</th>
<th>First Shift</th>
<th>Second Shift</th>
<th>Third Shift</th>
</tr>
</thead>
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<tr>
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<td>0</td>
<td>0</td>
</tr>
<tr>
<td>B1</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
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<td>B2</td>
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<tr>
<td>I7</td>
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</tr>
</tbody>
</table>

8.2. Results

As explained in section 6, the initial set of parameters is set and 1000 random dumping sequences are created. The transportation model is built; in the first and second iterations of simulation-optimization, there was no feasible solution. After 10-th iteration, which in every iteration input parameters have been altered, 464 feasible production schedules were found. For the demonstration purpose, one of the Gantt-Charts is presented in Figure 12.

Figure 11. An extraction sequence given for a day.
Figure 12. A feasible Gantt-Chart.

Figure 13. Utilizations of nine different feasible schedules, output of optimization block.
Figure 14. Utilizations of nine different feasible schedules, output of optimization block.

Figure 15. Utilization of excavators after running in Arena for the best schedule.
Out of these many feasible solutions, based on the below mentioned performance measures, a number of solutions are chosen as $n$ best-solutions to be tested in Arena®:

- Gantt-Charts,
- Utilizations of excavators (Figure 13),
- Utilizations of Spreaders (Figure 14),
- Busy and total available hours.

The aim of this selection is to find solutions with higher and realistic utilization. This is due to the fact that the utilization is equal to 97%, for instance, there is a high chance that will not happen in the reality when the unscheduled breakdown behavior is added to the model in the simulation. This can be true for spreaders as well. The selected schedules are run in Arena® and the result of the best schedule is presented in Figure 15.

9. Conclusions

Throughout this study, a new simulation-based optimization approach has been proposed. The approach is capable of optimizing the dispatch decisions in an opencast mine operated under the paradigm of continuously excavated material flow. A deterministic optimization and stochastic simulation models were combined. The transportation problem and the job-shop scheduling problem composed the optimization model. The performance of the proposed approach was tested in a real-size case study. For this case, 1000 random dumping sequences were created. The results showed that for a given extraction sequence and the random dumping sequences, the optimum dispatch decisions were obtained after 10-th iteration. 464 out of 1000 were found as feasible production schedules. In every simulation-optimization loop, the 10 best schedules based on the defined criteria were selected to be run in Arena. In two steps optimization approach of short-term production scheduling, the scheduling elements, physical sequencing and equipment utilization, are artificially separated so that they do not benefit from their simultaneous optimization. As a future work, a single step optimization approach is recommended, i.e. physical sequencing can be merged into the deterministic optimization.

Acknowledgement

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References


research and development pipeline management. 
AIChE Journal, 47, 2226-2242.


Methodology for assessment of the accessibility of dimension and crushed stone deposits from the environmental and social perspective

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The demand for resources from dimension and crushed stone deposits causes intensification of mining activities involving conflicts of an environmental and social nature. Therefore, it is essential to recognise potential conflict locations and assess the accessibility of deposits explored provisionally or in detail from the environmental and social perspective. This information can be obtained thanks to the methodology for deposit assessment offered in the article which used tools available in the geographical information systems and the multicriteria analysis in a form of the weighted sum method. This article analysed deposits of dimension and crushed stones located within the area of Lower Silesia in Poland.

Keywords: open-pit mining, geospatial data, multicriteria analysis, spatial development, environmental protection

1. Introduction

Mining is defined as a science encompassing the general issues related to mineral mining or activities required in this scope, that is, documentation and exploration of a deposit, deposit preparation and enabling works, mining of the deposit, mineral transportation and processing as well as post-mining area reclamation and development. Mining perceived in terms of a branch of industry has a considerable influence on the national economy. The issues of mining for energy resources and metal ores are commonly known and attracting attention of the government administration and media in Poland. Rock mining is no less important as it provides rock raw materials for, among others, road and rail infrastructure, or the housing industry. This group of minerals includes clay, aggregate and solid raw materials. The mentioned groups of resources are not distributed evenly across the whole country. Considering the administrative division of the country into 16 voivodeships, one can notice that the highest number of deposits (approximately 12% of all rock raw material deposits at different stage of development) is located in Masovian Voivodeship, the second highest number is found in Greater Poland Voivodeship (approx. 10% of deposits), then in Łódź and Lublin Voivodeship (9% of deposits each), and Podkarpackie and Lower Silesian Voivodeship (approx. 7% each). If one analyses the location of dimension and crushed stones (it is the most numerous mineral type among solid resources) in this group of rock raw materials it will turn out that their largest concentration is found in Lower Silesian which constitutes 42.2 percent of the national output of dimension and crushed stones (272 deposits), Świętokrzyskie Voivodeship with 34.9 percent of the output (137 deposits) and Lesser Poland Voivodeship with 12.1 percent of the national output (105 deposits) [1], which is shown in Figure 1.

Dimension and crushed stones embody 33 lithological varieties of igneous rocks, sedimentary rocks and metamorphic rocks characterized by specific features defining their economic use.

Figure 1. Dimension and crushed stone deposits in Poland

Each stage of deposit management results in interference in the natural environment and has an influence on human life and health. The nature and amount of changes vary depending on a type of raw material, volume of output and applied technologies of mineral mining and
processing. On the basis of the literature on the subject concerning the description of a mining activity [2, 3, 4, 5, 6, 7], environmental and social impacts have been identified, which are included in further analyses (Table 1). This article offers the methodology for assessment of accessibility of undeveloped deposits of dimension and crushed stones explored provisionally or in detail considering the impact of mining for these deposits on the environment and society. The aim of this assessment is to obtain information on conflict locations concerning a mining activity and dividing them into deposit accessibility classes.

2. Materials and methods
The article presented the methodology enabling one to determine the accessibility of the undeveloped dimension and crushed stone deposits taking environmental and social aspects into consideration. Figure 2 shows the procedure according to the offered methodology.

The Multi-Criteria Evaluation MCE, which aims at selecting an optimal solution determined according to different criteria which are hard to compare and having the influence on implementation and functioning of a particular solution, was used to determine the accessibility. This analysis in used as a tool supporting a decision-making process in case of having many criteria. The examples of using the MCE in research studies concerning the support of decision-making processes are instances described in the works of [8, 9, 10, 11, 12, 13, 14, 15, 16, 17, 18]. The deposit accessibility according to the MCE was calculated on the basis of the following equation:

\[ S = \sum_{i=1}^{n} w_i \times p_i \]  

where:
\[ S \] – land accessibility
\[ w \] – criterion weight
\[ \sum_{i=1}^{n} w_i = 1 \]  

\[ p \] – parameter value
\[ i \] – criterion
\[ n \] – number of criteria
\[ i \in< 1, n > \]

The environmental and social criteria of accessibility were determined on the basis of the literature on the subject describing the impact of open-pit exploitation of dimension and crushed stone deposits [2, 3, 4, 5, 6, 7], legal instruments related to the environmental protection [19, 20, 21, 22, 23] and own experiences of this article's authors [24, 25, 26]. Criteria of accessibility were identified (Table 1) and subsequently subject to a questionnaire survey in order to determine weights for the social and environmental models by a group of respondents composed of mining, environmental protection and spatial development specialists. The surveyed group was informed of the aim of the analysis, research object and territorial limits defined by the administrative boundaries of Lower Silesia. The respondents were specifying weights for each criterion taking the type of the analysed social and environmental model into account. Having obtained the results of the questionnaire survey, there was the arithmetic mean calculated from weights. Table 1 lists 33 criteria including the determined weights.

To determine the mining impact on the environment and human life and health, one needs reliable data describing a
deposit and its surroundings and, especially, data concerning social and environmental conditions. The sources of input data in the presented example were cartographic digital documents. The vector layers in a shape format were derived from the national resources, including: Provincial Centre for Geodesic and Cartographic Documentation, General Directorate for Environmental Protection, National Water Management Authority and Polish Geological Institute.

After having obtained vector data, each layer was cut to fit the limits of the analysed area. Then the vector data were converted into the raster format. The raster files, after being rectified, which consists in attributing the value of 1 to locations on the raster where the criterion was found, and the value of 0 were there was no criterion, were subject to the multicriteria analysis by the Weighted Sum Method WSM. This method consists in multiplying each of the applied layer by the determined weights and summing up the obtained values. The ArcGIS 10.4. software was used for calculations.

3. Results and discussion

The area of Lower Silesia located in southwestern Poland (Figure 1) was selected for the analysis, and it includes 90 undeveloped deposits of dimension and crushed stones explored provisionally or in detail, of geological balance resources fulfilling economic conditions for mining in the volume of 1689.6 million Mg for economic utilization for building of road, railway and housing infrastructure. Mining for these deposits may turn out to be impossible due to limitations related to, among others: protection of the natural environment, landscape qualities, forests, lands, green areas, surface water and groundwater, and existence of solid infrastructure in a form of built-up areas, roads and railway. Therefore, the research study attempted to assess the accessibility of dimension and crushed stone deposits from the environmental and social perspective, including 33 limitations.

Lower Silesia is a region in which, according to the analysis, there are many areas that could generate social and environmental conflicts during rock mining. For instance, 18.61% of the area of Lower Silesia is covered by legal protection; 0.9% of the area are lands below stagnant and running waters; 6.89 % is covered by built-up areas, 59.62% by agricultural land, 31.35% by forest and wooded areas and shrubland, 3.13% by communication areas in form of roads, and 0.43% in form of railway.

The results of the analysis were two maps informing of a level of accessibility of the analysed area in the context of future mining for dimension and crushed stone deposits in environmental and social aspects. When presenting the resultant maps, one used a five-grade division to classify the lands as: non-conflict and accessible, conflict and accessible with small limitations, conflict and accessible with medium limitations, conflict and accessible with great limitations or conflict and inaccessible. A number of classes in each map is the same. A certain fragment of the map was attributed to the class on the basis of the value attributed to each pixel (gridcode), which was the result of the multicriteria analysis calculations done by the Weighted Sum Method. A higher value means greater accessibility limitations. The obtained results were multiplied by x100 and written down in a form of integers. In the case of the environmental model, it was the value from 0 to 47, while for the social model, it was from 0 to 23. The resultant maps presenting the level of land accessibility for environmental (a) and social (b) models are shown in Figure 3.
On the other hand, a range of values corresponding with each class in the models was dependent on the percentage of the class area within the whole analysed region. For the environmental model, the non-conflict and accessible lands cover 0-1 range and 13.0% of the analysed area; in the case of conflict and accessible lands with small limitations it is 2-5 range and 15.5%; conflict and accessible lands with medium limitations - it is 6-13 range and 35.1%; conflict and accessible lands with great limitations - it is 14-25 range and 25.5%; conflict and inaccessible lands - it is 26-47 range and 19.4%; conflict and inaccessible lands with small limitations it is 1-2 range and 12.9%; in the case of conflict and accessible lands with medium limitations, it is 3-5 range and 29.5%; conflict and accessible lands with great limitations - it is 6-7 range and 21.1%; conflict and inaccessible lands - it is 8-23 range and 12.9% of Lower Silesia. The width of classes was mostly determined by the condition saying that the border classes, compared to other classes, cover the smaller area of the analysed land. The class of the largest area in case of the both models was the medium accessibility class. An impediment in selecting the class ranges - so that each of these cover the same area in the both models - was assuming integer values of the gridcode (values change stepwise). However, this was necessary with regard to the need of data conversion into the vector format. In the next stage, the area of deposit falling to each of the defined class in the environmental and social models was calculated and analysed. The analysis included 90 undeveloped deposits of dimension and crushed stones explored provisionally and in detail. The results were used to assess the accessibility of the analysed deposits in the environmental and social perspective. The percentage of deposit areas within the lands is presented according to the five-grade accessibility scale in Table 2.

Table 2 Classes of accessibility of lands over the undeveloped dimension and crushed stone deposits

<table>
<thead>
<tr>
<th>Environmental model</th>
<th>Classes of accessibility of lands over deposits</th>
<th>Social model</th>
</tr>
</thead>
<tbody>
<tr>
<td>7.6 %</td>
<td>I non-conflict lands</td>
<td>4.2%</td>
</tr>
<tr>
<td>10.8 %</td>
<td>II conflict and accessible lands with small limitations</td>
<td>37.5%</td>
</tr>
<tr>
<td>37.3 %</td>
<td>III conflict and accessible lands with medium limitations</td>
<td>16.6%</td>
</tr>
<tr>
<td>29.3 %</td>
<td>IV conflict and accessible lands with great limitations</td>
<td>26.1%</td>
</tr>
<tr>
<td>15.0 %</td>
<td>V conflict and inaccessible lands</td>
<td>15.6%</td>
</tr>
</tbody>
</table>

From among all the analysed deposits, the location of 21 dimension and crushed stone deposits was considered accessible according to the environmental model. These are deposits of sandstone, porphyry, melaphyre, amphibolite, gneisses, greener stone, basalt, granodiorite, granite, marble, serpentinite and gabbro. According to the social model, among the accessible deposits there were fragments of 17 deposits, and 16 of those were previously defined as accessible in the environmental model. The class of conflict and inaccessible lands was identified in the case of 17 deposits for the environmental model, including the deposits of: granodiorite, amphibolite, granite, marble, basalt, sandstone and melaphyre. Among the social model, among conflict and inaccessible deposits there are fragments of 43 deposits and 11 of those were previously defined as conflict ones in the environmental model. In the case of the environmental model, none of the deposits simultaneously fell within the both border classes (I and V). On the other hand, in the social model, I and V accessibility classes were simultaneously identified within the area of 4 deposits. A number of deposits included in the particular accessibility classes is shown in Figure 4.

Figure 4 Number of deposits according to the accessibility class environmental and social model

On the basis of the aforementioned diagrams, one can state that objects of higher impact (higher weights) in the environmental model are centralized and of larger area (among others, this concerns the forms of nature conservation) while the objects of stronger connection with social conflicts are more scattered over the analysed area (among others, this concerns built-up areas and transport infrastructure). An exemplary map showing the accessibility classification for the selected Stankowice deposit of gneiss is presented in Figure 5. In both cases, there are areas belonging to three classes of accessibility enabling one to start a mining activity over the deposit: I, II and III. In the case of the environmental model, the largest and simultaneously coherent land is class I of 166566.3608 m² area. The social model is characterized by similar areas of class I and II (100000 m²), while lands covered with these classes are scattered within the limits of the deposit in both cases. Areas covered by the particular classes (percentage) were shown in Figure 5.
4. Conclusion

According to the presented methodology, the information on the assessment of accessibility of lands over deposits of dimension and crushed stones located within the analysed area (Lower Silesia, SW Poland) was obtained. The methodology offered the division into five classes of land accessibility for a mining activity in the environmental and social models (Table 2). Thanks to the offered method it is possible to obtain the results of classification of each fragment of an analysed deposit together with its surroundings at any distance from the deposit but within the limits of the analysed area. The information can be made accessible to deposit users, mining companies interested in mining for deposits or administration authorities responsible for issuing decisions on the concession for extraction of deposits so that they have knowledge of conflict areas in the region resulting from the impact of the planned mining activity on human life and health as well as the environment. Therefore, the offered methodology constitutes a useful tool providing information on accessibility of deposits and land surrounding the deposits of dimension and crushed stones and indicating conflict areas related to future mining.

Acknowledgments

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References


[22] Dz.U.2007.120.826 the Regulation of 14 June 2007 of On acceptable noise levels in the environment (in Polish)


<table>
<thead>
<tr>
<th>Group of criteria</th>
<th>Criterion</th>
<th>Weights Social model</th>
<th>Weights Environmental model</th>
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<td>Forms of nature conservation including Natura 2000 areas</td>
<td>Forms of higher nature conservation (national park, natural reserve, landscape park, national and landscape park buffer zones, natural monument)</td>
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<td>0.100</td>
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<td>Forms of lower nature conservation (protected landscape area, landscape-nature protected areas, Natura 2000, ecological site, documentation site)</td>
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<td>0.064</td>
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<td></td>
<td>Production forests</td>
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<td>Flood risk areas</td>
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<td>Surface water</td>
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<td>Main Groundwater Reservoirs</td>
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<td>Value 2</td>
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<td>----------</td>
</tr>
<tr>
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<td>Best and very good quality arable land</td>
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<td>Medium quality arable land</td>
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<td>Worst quality arable land</td>
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Development of Performance Prediction Models for Block-Cutting Machines (S/T) with Circular Saws Used in Natural Stone Factories

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Abstract—Block-cutting machines have the most important position in natural stone factories. It is used for the production of stone slabs in different sizes. These machines offer the most effective way to produce slabs with smooth surfaces that do not require any extra work for shaping. The machine can be easily operated only with one person. All of mentioned features make the machine very cost effective compared to other machines commonly are used in natural stone processing plants. Several researchers have already published some studies about these machines performance using specific methods with some particular physical and mechanical properties of limited types of natural stones. Hence, a wide range of natural stones was selected in this study to create more reliable performance prediction models. Different natural stone quarries located in Turkey were visited to collect natural stone samples and investigate the operational and performance parameters of block-cutting machines. Physical and mechanical property tests (density, uniaxial compressive strength, Brazilian tensile strength, Schmidt hammer rebound values, and Shore scleroscope hardness) were performed on the obtained samples. Then, the abrasivity of natural stones was determined based on Cerchar abrasivity index. Later, mineralogical and petrographic analyses were carried out for determination of modal composition and grain size distribution of the stone samples. In addition, actual areal slab production rates were measured on the blocks of the selected natural stones in the processing plants. The results of field and laboratory studies were used to develop new empirical models for prediction of actual areal slab production rate of block-cutting machines. Finally, actual performance data obtained from past studies was used in order to test the reliability of the suggested models.

Keywords: block-cutting machine, physical-mechanical properties, mineralogical and petrographical properties, regression analysis

1. Introduction
Natural stones have been used worldwide as construction and decorative materials. In recent years, the demand for natural stones has highly increased because of steadily developing technologies and rapidly spreading innovative design ideas from all over the world. Turkey is one of the biggest exporters of natural stones worldwide. For such an important industry, in-depth studies are required in order to meet the need of the growing market demands by supplying high quality products. Therefore, the machines rather used for the extraction of natural stones from quarries or for further stone processing in factories must be examined in detail.

Block-Cutting (S/T) machines with circular saws are one of the commonly used machines in natural stone planting for produce slabs in different size. Block-cutting machines offer the most effective way to produce slabs with smooth surfaces that do not require any extra work for shaping. The machine can be easily operated only with one person. However, they also have some limitations. These limitations generally arise from geotechnical features of the stone. Selection of a suitable machine and its performance generally depends on the physical and mechanical properties of the stone, machine characteristics, saw properties, penetration rate, and tool consumption. Machine performance directly affects the planning of the plants and the cost prediction of the producing companies.

It is possible to find some studies in the literature that correlated the performance parameter of this machine with physical-mechanical and petrographic properties of natural stones. Norling [1] investigated the relationship between sawability of stones with petrographic properties and stated that grain size was more relevant than quartz content in sawability parameter. Gunaydin et al. [2] used the brittleness indexes for sawability prediction of carbonate rocks. Kahraman et al. [3] proposed some models for prediction of the sawability of carbonate rocks based on the multiple curvilinear regression analysis. Delgado et al. [4] found a strong correlation between rock microhardness and sawability based on the investigations performed on the pink Spanish granite. Tumac [5] suggested two statistical models based on the Cerchar abrasivity index.
and deformation coefficient to estimate the areal slab production rate of carbonate rocks.

Accurate performance prediction of block-cutting machines is one of the main parameters that affect the cost prediction and planning of the plants in producing companies. This parameter generally depends on the physico-mechanical properties of stones, specifications and design of the machine, and operational conditions. In the light of these facts, this study aimed to proposed new models for prediction of areal slab production rate of block-cutting machines. To reach this goal, different natural stones processing plants were visited in Turkey to determine the geological conditions, specifications and operational conditions of the block-cutting machines. Then physical and mechanical properties of the samples taken during field visits were identified in the laboratory. The laboratory tests included determination of density ($\rho$), uniaxial compressive strength (UCS), Brazilian tensile strength (BTS), Cerchar abrasivity index (CAI), Schmidt hammer hardness value (SHH) and Shore scleroscope hardness value (SSH). In addition, thin section petrographic analyses were also performed in order to describe the stone textures. Later, by use of the independent variables of the database created by laboratory and field studies, based on statistical analysis different models were suggested for prediction of areal slab production rate of block-cutting machines. Finally, the proposed models were validated with actual performance data obtained from previous studies.

2. Experimental studies

Following nine different natural stone samples were collected from different natural stone processing plants in Turkey: Burdur beige, Söğüt beige, Korkuteli beige, Denizli pink marble, Kavaklıdere white marble, Uşak white marble, Bucak white travertine, Kaklık travertine and Denizli yellow travertine. These samples are categorized based on their origins. Marbles are named as metamorphic natural stones; beige and travertine samples are named as sedimentary natural stones.

Uniaxial compressive strength and Brazilian tensile strength tests were performed according to ISRM [6] suggestions. Uniaxial compressive strength tests were performed on grinded NX core samples with a length to diameter ratio of 2.5. The stress rate applied to core samples was 0.5 kN/s. Brazilian tensile tests were performed on NX core samples with 0.25 kN/s stress rate and the ratio of a length to diameter of 0.5. Cerchar abrasivity tests were performed based on the procedures described by ASTM [7]. At least five individual tests were conducted for each natural stone sample, and then the average value of these tests was selected as CAI value. Schmidt hammer test were carried out with an L type Schmidt hammer in ten different point on surface of samples and were repeated in five sets based on the procedure described by Fowell and McFeat-Smith [8]. The average value of five set reading was selected as SHH. Shore scleroscope hardness tests were performed on stone blocks with a C-2 type instrument. The procedure described by McFeat-Smith [9] was used in determination of SHH. Measurement points were at a distance of at least 5 mm from each other, the minimum number of tests for each stone is taken as 50, and the average value of the results was selected as SSH. Summary of the physical and mechanical properties of natural stone samples are given in Table 1. Thin section petrographic analysis was used to define the stone texture. Figure 1 shows the thin section photograph of each natural stones. In addition, brief petrographic descriptions of the tested natural stones are given in Table 2.

Table 1 – Summary of the physical and mechanical properties of natural stone samples.

<table>
<thead>
<tr>
<th>Natural stone name</th>
<th>$\rho$ (g/cm$^3$)</th>
<th>UCS (MPa)</th>
<th>BTS (MPa)</th>
<th>CAI</th>
<th>SHH</th>
<th>SSH</th>
</tr>
</thead>
<tbody>
<tr>
<td>Burdur beige</td>
<td>2.72</td>
<td>112.3</td>
<td>4.3</td>
<td>1.4</td>
<td>61.0</td>
<td>59.1</td>
</tr>
<tr>
<td>Söğüt beige</td>
<td>2.73</td>
<td>111.9</td>
<td>6.9</td>
<td>1.9</td>
<td>63.3</td>
<td>59.9</td>
</tr>
<tr>
<td>Korkuteli beige</td>
<td>2.70</td>
<td>120.3</td>
<td>9.4</td>
<td>1.3</td>
<td>65.6</td>
<td>62.3</td>
</tr>
<tr>
<td>Denizli pink marble</td>
<td>2.72</td>
<td>70.2</td>
<td>6.4</td>
<td>2.0</td>
<td>47.0</td>
<td>43.0</td>
</tr>
<tr>
<td>Kavaklıdere white marble</td>
<td>2.67</td>
<td>79.3</td>
<td>6.4</td>
<td>3.4</td>
<td>65.3</td>
<td>58.0</td>
</tr>
<tr>
<td>Uşak white marble</td>
<td>2.69</td>
<td>63.8</td>
<td>4.6</td>
<td>1.0</td>
<td>44.4</td>
<td>54.1</td>
</tr>
<tr>
<td>Bucak white travertine</td>
<td>2.40</td>
<td>45.1</td>
<td>4.0</td>
<td>0.6</td>
<td>43.2</td>
<td>37.1</td>
</tr>
<tr>
<td>Kaklık travertine</td>
<td>2.14</td>
<td>11.0</td>
<td>4.4</td>
<td>0.4</td>
<td>40.1</td>
<td>26.6</td>
</tr>
<tr>
<td>Denizli yellow travertine</td>
<td>2.50</td>
<td>50.5</td>
<td>4.8</td>
<td>0.7</td>
<td>39.0</td>
<td>30.0</td>
</tr>
</tbody>
</table>

$\rho$: density, UCS: uniaxial compressive strength, BTS: Brazilian tensile strength, CAI: Cerchar abrasivity index, SHH: Schmidt hardness value, SSH: shore hardness value.

Figure 1. Microphotography of samples (a) Burdur beige, (b) Söğüt beige, (c) Korkuteli beige, (d) Denizli pink marble, (e) Kavaklıdere white marble, (f) Uşak white marble, (g) Bucak white travertine, (h) Kaklık travertine, (i) Denizli yellow travertine (Cal: calcite, Do: dolomite, B: gap, Mi: micritic).

The results of laboratory studies indicate that the density values range from 2.14 to 2.73 g/cm$^3$. Uniaxial compressive strength varies between 45.1 and 120.3 MPa. Based on the strength classification of intact rock [10], samples in this study vary from weak rock to hard rock. Brazilian tensile strength ranges from 4.0 to 9.4 MPa. Cerchar abrasivity values changes between 0.4 and 3.4. Moreover, based on ASTM [7] classification of abrasivity of rocks, samples in this study are...
classified as very low abrasiveness to high abrasiveness. Schmidt hammer hardness values range from 39.0 to 65.6 and Shore scleroscope hardness values vary between 26.6 and 62.3.

Table 2 – Petrographic descriptions of the studied natural stone samples.

<table>
<thead>
<tr>
<th>Natural stone name</th>
<th>Petrographic descriptions</th>
<th>Minerals</th>
<th>Mean grain size (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Burdur beige</td>
<td>Limestone. Fine grained. Consisting of cryptocrystalline calcite within cryptomicrocrystalline calcite. Micritic texture.</td>
<td>Calcite (mic), 2% Calcite (spr)</td>
<td>0.03</td>
</tr>
<tr>
<td>Siğzłt beige</td>
<td>Limestone. Fine grained cryptocrystalline calcite crystals smaller than 5 micron, including irregular distributed dolomite rhombohedrons. Micritic texture.</td>
<td>Calcite (spr), 2%</td>
<td>0.04</td>
</tr>
<tr>
<td>Korkuteli beige</td>
<td>Micritic limestone. Fine grained. Cracks filled with aragonite minerals. Micritic texture.</td>
<td>Dolomite (spr), 5%</td>
<td>0.03</td>
</tr>
<tr>
<td>Denizli pink marble</td>
<td>Dolomitic marble. Fine grained. Including slightly dolomitic grains, granoblastic texture.</td>
<td>Dolomite</td>
<td>0.55</td>
</tr>
<tr>
<td>Uşak white marble</td>
<td>Marbles. Medium grained with polysynthetic twins, granoblastic texture.</td>
<td>Calcite (spr), 2% Opal</td>
<td>0.60</td>
</tr>
<tr>
<td>Bucak white travertine</td>
<td>Micritic limestone. Fine grained calcite is the dominant mineral. Consist of micro-mesocrystalline calcite minerals with little amount of clay. Often contains pores. Micritic (intraclast) texture.</td>
<td>Calcite (spr), 5%</td>
<td>0.07</td>
</tr>
<tr>
<td>Kaklık travertine</td>
<td>Micritic limestone. Fine grained calcite is the dominant mineral. Often contains pores. Sparacalcite filling between the minerals. Micritic (intraclast) texture.</td>
<td>Calcite (spr), 2%</td>
<td>0.06</td>
</tr>
<tr>
<td>Denizli yellow travertine</td>
<td>Micritic limestone. Fine grained calcite is the dominant mineral. Contains abundant cavities filled with secondary calcite minerals. Micritic (intraclast) texture.</td>
<td>Calcite (spr), 3%</td>
<td>0.08</td>
</tr>
</tbody>
</table>

3. Field studies

Different natural stone processing plants were visited to measure the slab production performance of block-cutting machines. All the machines used in this study are four-footed machines. Four-footed block-cutting machine has an ability to move in three directions (x, y, and z). The main bridge unit of the machine is connected to the side bridges differently from two-footed machine. Side bridges are also connected to the foot that are located in pairs to the right and left sides of the machine. The movement of the vertical saw in x-axis takes place on main bridge similar to two-footed machine. The movement in y-axis adjusts the depth of slab. The movement in third dimension (z) helps setting up the thickness of the slab. Diamond segmented saws used in sawing process have diameters of 1200 and 1600 mm. The weight of the machines changes between 16 and 17.5 ton. In addition, rotational speed of saws ranges from 2400 to 2610 rpm. The advance rate of saw varies between 2.3 and 2.5 cm/s. However, the areal slab production rate (ASPR) ranges from 6.1 to 20 m²/h. A summary of the results of the performance of block-cutting machines is given in Table 3.

Table 3 – Measured performance of block-cutting machines

<table>
<thead>
<tr>
<th>Observation numbers</th>
<th>1</th>
<th>2</th>
<th>3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Natural stone name</td>
<td>Burdur beige</td>
<td>Siğzłt beige</td>
<td>Korkuteli beige</td>
</tr>
<tr>
<td>Origin</td>
<td>Sedimentary</td>
<td>Sedimentary</td>
<td>Sedimentary</td>
</tr>
<tr>
<td>Machine weight (ton)</td>
<td>17</td>
<td>17</td>
<td>17</td>
</tr>
<tr>
<td>Saw diameter (mm)</td>
<td>1200</td>
<td>1200</td>
<td>1200</td>
</tr>
<tr>
<td>Rotational speed of saw (rpm)</td>
<td>2215</td>
<td>2220</td>
<td>2310</td>
</tr>
<tr>
<td>Vertical saw motor power (kW)</td>
<td>132</td>
<td>132</td>
<td>132</td>
</tr>
<tr>
<td>Horizontal saw motor power (kW)</td>
<td>18.5</td>
<td>18.5</td>
<td>18.5</td>
</tr>
<tr>
<td>Advance rate of saw (cm/s)</td>
<td>5</td>
<td>140 x 30</td>
<td>5 x 135 x 30</td>
</tr>
<tr>
<td>Slab dimension (cm x cm x cm)</td>
<td>4</td>
<td>7.0</td>
<td>16.6</td>
</tr>
<tr>
<td>Areal slab production rate (m²/h)</td>
<td>98% Calcite, 2% Opaque</td>
<td>95% Calcite, 5%</td>
<td>95% Calcite, 5%</td>
</tr>
<tr>
<td>Observation numbers</td>
<td>4</td>
<td>5</td>
<td>6</td>
</tr>
<tr>
<td>Natural stone name</td>
<td>Denizli pink marble</td>
<td>Kayaköy white marble</td>
<td>Uşak white marble</td>
</tr>
<tr>
<td>Origin</td>
<td>Metamorphic</td>
<td>Metamorphic</td>
<td>Metamorphic</td>
</tr>
<tr>
<td>Machine weight (ton)</td>
<td>17.5</td>
<td>17.5</td>
<td>17.5</td>
</tr>
<tr>
<td>Saw diameter (mm)</td>
<td>1600</td>
<td>1600</td>
<td>1600</td>
</tr>
<tr>
<td>Rotational speed of saw (rpm)</td>
<td>2400</td>
<td>2450</td>
<td>2400</td>
</tr>
<tr>
<td>Vertical saw motor power (kW)</td>
<td>160</td>
<td>160</td>
<td>160</td>
</tr>
<tr>
<td>Horizontal saw motor power (kW)</td>
<td>22</td>
<td>22</td>
<td>22</td>
</tr>
<tr>
<td>Advance rate of saw (cm/s)</td>
<td>2.4</td>
<td>2.3</td>
<td>2.4</td>
</tr>
<tr>
<td>Slab dimension (cm x cm x cm)</td>
<td>5 x 160 x 30</td>
<td>5 x 160 x 30</td>
<td>5 x 200 x 30</td>
</tr>
<tr>
<td>Areal slab production rate (m²/h)</td>
<td>12.7</td>
<td>20.6</td>
<td>21.3</td>
</tr>
<tr>
<td>Observation numbers</td>
<td>7</td>
<td>8</td>
<td>9</td>
</tr>
<tr>
<td>Natural stone name</td>
<td>Bucak white travertine</td>
<td>Kaklık travertine</td>
<td>Denizli yellow travertine</td>
</tr>
<tr>
<td>Origin</td>
<td>Sedimentary</td>
<td>Sedimentary</td>
<td>Sedimentary</td>
</tr>
<tr>
<td>Machine model</td>
<td>Ergenler four footed</td>
<td>Ergenler four footed</td>
<td>Ergenler four footed</td>
</tr>
<tr>
<td>Machine weight (ton)</td>
<td>16</td>
<td>16</td>
<td>16</td>
</tr>
<tr>
<td>Saw diameter (mm)</td>
<td>1600</td>
<td>1200</td>
<td>1200</td>
</tr>
<tr>
<td>Rotational speed of saw (rpm)</td>
<td>2600</td>
<td>2610</td>
<td>2550</td>
</tr>
<tr>
<td>Vertical saw motor power (kW)</td>
<td>110</td>
<td>110</td>
<td>110</td>
</tr>
<tr>
<td>Horizontal saw motor power (kW)</td>
<td>15</td>
<td>15</td>
<td>15</td>
</tr>
<tr>
<td>Advance rate of saw (cm/s)</td>
<td>2.4</td>
<td>2.4</td>
<td>2.5</td>
</tr>
<tr>
<td>Slab dimension (cm x cm x cm)</td>
<td>5 x 160 x 30</td>
<td>5 x 160 x 30</td>
<td>5 x 160 x 30</td>
</tr>
<tr>
<td>Areal slab production rate (m²/h)</td>
<td>12.3</td>
<td>20.0</td>
<td>14.0</td>
</tr>
</tbody>
</table>
4. Statistical models for prediction of areal slab production rate

Statistical analysis is the science of collecting, exploring and presenting large amounts of data to discover underlying patterns and trends. Statistical analysis is also a very useful tool to get approximate solutions when the actual process is highly complex or unknown in its true form. In this study, two forms of statistical analysis are selected to suggest new models for performance prediction of block-cutting machines. These forms are simple regression and multiple regression. The detailed of each models are mentioned below.

4.1. Model based on simple regression analysis

In statistics, simple linear regression has been used to find a linear function between dependent variable and independent variable. This method is the simplest way to find best-fitting line between variables. The best fitting line is called as regression line. In this paper, simple regression analysis performed to find the relationship between areal slab production rate (ASPR) and physico-mechanical properties of natural stones. Figures 2a and 2b shows the relationship between ASPR and other independent variables.

**Figure 2a.** Relationship between areal slab production rate and density, UCS and BTS.

**Figure 2b.** Relationship between areal slab production rate and CAI, SHH, SSH and mean grain size.

As seen in Figure 2, among the independent variables, the uniaxial compressive strength values show the best correlation with ASPR (with R² of 0.81). The main reason is the range of this parameter. As stated in section 2, the range of uniaxial compressive strength covers an acceptable range, in other words, it can be said that this range covers a wide range of natural stone groups. This model is selected as the best correlated model in validation stage with previous studies as mentioned in Equation 1:
ASPR = 74.85 × UCS^{0.48} 

In addition, Schmidt hardness and Shore hardness values show good relationships with ASPR that means these two independent parameters could be used as useful and practical predictor for performance prediction of block-cutting with circular saws. However, the mean grain size values show weak relationship with ASPR (R^2 of 0.14). The main reason for this weak relationship is the narrow range of mean grain size in the selected natural stones.

4.2. Model based on multiple regression analysis

Multiple linear regression is one of the most used form of statistical analysis in the all research fields. This analysis is used to find relationship between one dependent variable from two or more independent variables. The multiple linear regression analysis is carried out using SPSS (IBM® SPSS® Statistics Version 23) program that uses stepwise regression procedure to find the best relationship between variables. Confidence level is selected as 95% (in some case 90%). In this confidence level, the p-value (probability value) of regression analysis should be less than or equal to 0.05 (in case of 90%, 0.10). In this study, analysis of variance (ANOVA) is used to detect the significance of the created regression models. The advantage of ANOVA is ability to testing of three or more variables for statistical significance. ANOVA is a special case of regression where the predictors are indicators of group membership and functions of that and t-test is a special case of ANOVA with just two groups. For confirmation of multiple regression analysis two item are considered as:

- F-ratio: If the F-ratio is greater than the F-tabulated value obtained from the F distribution table, then it can be said that the regression is significant.
- T-test value: If |t| is greater than the t-tabulated value obtained from the t-distribution table then it can be concluded that the independent variables contributes to the model.

The results of multiple regression analysis are summarized in Table 4. As seen in this table, two models are passed from the confirmation stage. In model 1, the confidence level is selected as 95%. In addition, in model 2 confidence level was selected as 90%. In these two models, p-values are less than 0.05 and 0.10. In model 1, the tabulated F-value is 5.14, so the F-ratio is greater than this value and the F-ratio test is passed. Moreover, the tabulated t-value is changed between -2.31 and 2.31. In this case, t-values is greater than this range, this step also is passed and model accepted. In model 2, the tabulated F-value is 5.41, so the F-ratio is greater than this value and the F-ratio test is passed. Moreover, the tabulated t-value is changed between -2.015 and 2.015. In this case, t-values are greater this range, this step also is passed and model accepted.

4.3. Comparison of predicted and actual areal slab production rates

The proposed models based on regression analysis were validated with previous studies in the literature. Therefore, the results of Tumac et al. [11] study was selected to validation of this study models. The results used for validation are summarized in Table 5.

<table>
<thead>
<tr>
<th>Natural stone name</th>
<th>ρ (gr/cm^3)</th>
<th>UCS (MPa)</th>
<th>CAI</th>
<th>SHH</th>
<th>ASPR (m^2/h)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Karahalli withe marble</td>
<td>2.69</td>
<td>63.8</td>
<td>1.04</td>
<td>47.3</td>
<td>11.9</td>
</tr>
<tr>
<td>Karahalli gray marble</td>
<td>2.70</td>
<td>70.2</td>
<td>1.32</td>
<td>40.4</td>
<td>13.4</td>
</tr>
<tr>
<td>Mustafa Kemal pașa withe marble</td>
<td>2.70</td>
<td>77.8</td>
<td>1.11</td>
<td>44.1</td>
<td>12.3</td>
</tr>
<tr>
<td>Sivasü purple marble</td>
<td>2.68</td>
<td>108.0</td>
<td>1.01</td>
<td>61.2</td>
<td>7.1</td>
</tr>
</tbody>
</table>


A summary of the results of the actual and estimated ASPR values is given in Table 5. As seen in this table, that is a significant concurrence between the estimated and actual values. However, it is useful to remind here, the proposed models in this study are based on nine different natural stones and the validity of these models should be verified with other studies.

5. Conclusions

Performance prediction of block-cutting machines with circular saws is one of the important factors to improve the efficiency of cutting and decrease costs in natural stone plants. This study aims to suggest some empirical models for...
prediction of areal slab production rate in these machines. Regression analyses were used in simple and multiple forms to development of new performance prediction models. In these analyses, physical-mechanical properties of tested natural stones included density, uniaxial compressive strength, Brazilian tensile strength, Cerchar abrasivity index, Schmidt hammer hardness, Shore scleroscope hardness and mean grain size were selected as independent variables to find out the relationships with actual areal slab production rate (dependent variable) of block-cutting machines.

In the results of regression analysis, three models are suggested to prediction of areal slab production rate of block-cutting machines. The first model is a function of uniaxial compressive strength. The second model is a function of density and Schmidt hardness values. The third model is a function of density, Cerchar abrasivity index and Schmidt hardness values.

However, the proposed models in this study are empirical models and need to be verified with other studies. In addition, models should be improved by adding some main machine parameters such as cutting depth and tool consumption. Moreover, the effect of operator should be investigated on the performance prediction of block-cutting machines.

Acknowledgement

This study summarizes some of the results of MSc. research work carried out by second author. The authors would like to thank to natural stone processing companies for their contributions and help in this study; this work would be impossible without their support.

References


MPES2017

Session 6 – Mine Production I

11:30-12:30

August 30, 2017
Optimized Path Planning in Underground Mine Ramp Design Using Genetic Algorithm

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Abstract—Majority of traffic in an underground mine flows inside the main haulage way, which is called as the mine ramp. As a navigable haulage path between the portal and production levels, mine ramp is composed of straight and spiral sections with a fixed maximum gradient and minimum turning radius. Most commonly, skilled experts design a so-called satisfactory path based on their professional experience. However, there is no attempt to check whether the design is an optimum solution or not. Considering the initial capital investment and operating cost parameters are directly related with the path, an optimum design minimizing the path length is a necessity. Although, length is the main consideration, there may be other constraints that a ramp is required to satisfy. For example, the path must avoid some undesired underground spaces like aquifers, safety buffers, in order to keep the stability. In addition, some desired regions must be strictly connected by the ramp. Also, rock quality should be considered to drive the ramps inside a higher quality rock zone by decreasing the support cost and increasing the safety and speed of progress. Besides the mentioned constraints some kinematical constraints must be satisfied for the sake of navigability. All these objectives transform the ramp path optimization from a shortest path problem into a least cost estimation problem. This paper presents a genetic algorithm implementation for predicting the near optimum navigable haul road. The algorithm was validated by the benchmark problem and tested on a manually designed real underground mine ramp and improvements were noted.

Keywords: underground mine haul road, path planning, genetic algorithm, optimization

1. Introduction
An orebody underlying deep below the earth’s crust is most preferably accessed via a spiral ramp connected to the surface portal through a decline. Although vertical shaft provides an alternative, low production rate due to restricted material transportation capacity and limited availability for mechanization pushes decision makers to the first choice.

Path planning aims to design an optimum route for a moving object. There are two common approaches to construct these algorithms. The first approach, which is online path planning, is capable of predicting the optimum path based on the live information gathered in the meantime of the vehicle’s movement. This approach is most widely used in robotics and aeronautics where the environment is dynamic. Sensors attached on the vehicle detect unknown or changing environment and an autopilot system decides the optimum path simultaneous to the movement. Researchers proposed an evolutionary algorithm for online path planning in order to maximize the information collected in a mission with multiple unmanned aerial vehicles [1]. The second approach is off line path planning. In this approach, the environment is already recognized and the path is planned prior to the travel of the vehicle. This method is not dynamic and the vehicle moves exactly on the predicted path. In our study, we used off line path planning for the static environment of the underground main haul roads, which is recognized prior to the travel of the vehicle.

Underground mine haul roads are constrained by the kinematic capability of the underground mining equipment. For the sake of navigability, minimum turning radius and maximum gradient are two of the major constraints those need to be considered during the haul road path design. Gradient constrained paths for underground haul road have long been investigated by the researchers [2].

Although minimization of the path length is the most common effort in an underground mine ramp design, specific cases may require other objectives to be satisfied. Brazil et al. [3] proposed a decline optimization tool that keeps a standoff distance between the orebody and the decline. Yardimci and Karpuz [4] presented an evolutionary algorithm to optimize the haul road path without violating the safety buffer between the orebody and the haul road.

This study presents an evolutionary path planning algorithm that minimizes the cost of ramp. Cost items are path length, rock mass quality and the penalty of violating undesired regions. Each cost item is weighted in a single objective function according the desired effect on the total cost. Although the cost items are effective on the predicted path, this method does not guarantee to satisfy each item to the full extent. Result is most likely to be a near optimum solution.
2. Problem Description
This study presents a path planning problem in an underground mine haul road design. Objective is predicting a route with the minimum cost. The path must visit some predefined points and avoid some undesired regions. Apparently, the problem can be considered as an optimization problem. A representative layout of the problem can be seen in Figure 1.

![Figure 1 Layout of the path planning problem](4)

2.1. Objective Function
Cost of haul road path is defined in terms of four factors. In genetic algorithm terminology, objective function provides a fitness value to be minimized for an optimum path. Path length, rock mass quality, avoiding undesired region and passing from desired region are penalized and awarded in the objective function.

**Given:**

\[
\begin{align*}
\text{Main Node Coordinates} &= (x_i, y_i, z_i), \quad \text{for } i = 1 \ldots n \\
tr_{\text{min}} &= \text{minimum turning radius, m} \\
\psi &= \text{Maximum gradient, \%}
\end{align*}
\]

**Objective:**

\[
\text{Min}\ \{w_1 \times LC + w_2 \times RMQC + w_3 \times URC - w_4 \times DRC\}
\]

**Where:**

\[
\begin{align*}
LC &= \text{Cost of path length} \\
RQC &= \text{Cost of rock mass quality} \\
URC &= \text{Cost of violating an undesired region} \\
DRC &= \text{Cost of intersecting a desired region}
\end{align*}
\]

**Subject to:**

\[
\begin{align*}
0 < w_i < 1 & \quad \text{for } i = 1 \ldots 4 \\
\sum_{i=1}^{n} w_i = 1 & \quad \text{for } i = 1 \ldots 4
\end{align*}
\]

Each of the cost factors are weighted in the objective function. Weighting determines the cumulative effect of cost factors on the overall cost. Sum of the weightings are one.

2.1.1. Path Length Penalty
Length of the path is one of the most critical items in the objective function. As a rule of thumb, an efficient design for an underground mine ramp must travel the shortest path possible. By this way, short term (cost of ramp construction) and long term (operating cost) costs can be reduced.

2.1.2. Rock Mass Quality Penalty
Volume of the rock mass is defined by a block model of equally sized blocks, showing the rock mass quality zones. In order to quantitatively describe the rock mass quality, well known Geomechanical classification system, which is Rock Mass Rating (RMR), is used. The path length is weighted by the RMR score and the weighted average denoting the rock mass quality penalty is obtained. Blocks passing inside the structural anomalies like faults or joints are penalized by “0” rock quality.

2.1.3. Undesired Region Penalty
Structural anomalies like faults and joints, or pressurized underground spaces (like aquifers) are regions to be avoided in an underground haul road path. These regions may cause instability problems or increase the cost of construction. Therefore, the objective function includes a penalty factor these items.

2.1.4. Desired Region Award
Extents of the rock mass inside which the main haulage way will be constructed are determined to define the Region of Interest (ROI). The ROI is defined in terms of x, y and z boundary coordinates. In order to avoid violating the ROI boundaries, any node predicted inside this region is awarded by decreasing the fitness value.

3. Mathematical Model of an Underground Mining Vehicle
Motion of an underground mining vehicle must be presented by a mathematical model in order to calculate the exact location of the vehicle throughout the simulation. A realistic model can be as complex to consider the body forces and moments driven by the gravity, propulsion and aerodynamic forces. Researchers have successful attempt in modelling motion of UAVs moving along three axes with six degrees of freedom with non-linear fully coupled ordinary differential equations of motion [5]. Unfortunately, lack of closed form solution makes it hard to solve these equations. Instead, numerical methods perform more steady state solutions. As a simple kinematical model, Dubins car performs sufficiently well to represent the motion of an underground mining vehicle. Figure 2 represents an illustration of a 2D Dubins car controlled by the heading angle and constrained by the turning radius. A Right-Straight-Right (RSR) type Dubins path connects initial and final nodes.
A Dubins vehicle is a bounded speed and no reversing planar vehicle with constriction to move along paths of bounded curvature [6]. Governing equations for the motion of a Dubins car can be seen in (1), (2), (3) and (4).

\[ x_i = v_i \cos(\theta_i) \quad i = 1 \ldots n \]  
\[ y_i = v_i \sin(\theta_i) \quad i = 1 \ldots n \]  
\[ z_i = -v_i \sin(\psi_i) \quad i = 1 \ldots n \]  
\[ \theta_i = \frac{u_i v_i}{r_{\text{min}}} \quad i = 1 \ldots n \]  

Where;  
\( i \) = node number  
\( n \) = maximum number of nodes  
\( x_i \) = x coordinate of the \( i^{\text{th}} \) node  
\( y_i \) = y coordinate of the \( i^{\text{th}} \) node  
\( z_i \) = z coordinate of the \( i^{\text{th}} \) node  
\( \theta_i \) = heading angle of the \( i^{\text{th}} \) node  
\( \psi_i \) = max. gradient of the \( i^{\text{th}} \) node  
\( u_i \) = turn control  
\( v_i \) = velocity  
\( r_{\text{min}} \) = minimum turning radius

Lester Dubins [7] revealed that the minimal curvature constrained path connecting an initial point to a final point is one of the six alternative forms. Motion primitives are denoted by ‘S’ on the straight sections, while turns to the left and right are symbolized by ‘L’ and ‘R’. Consequently, the shortest curved path joining two nodes is one of the six paths given below:

\{RSL, LSR, RSR, LSL, RLRL, LRL\}

Figure 3 presents a sample implementation in Matlab environment. In this example, the initial node and final node coordinates are \((0,0,100)\) and \((100,100,60)\), heading angle is 135° for the initial node and 215° for the final node. Heading angles start from 0° on the East axes and increase in the counter-clockwise direction. Minimum turning radius is 25 m and the maximum gradient is 12%. The implementation reveals that the shortest path is an RSR type Dubins path. However, the path length and type of Dubins path may show variation as the heading angle changes. In our optimization effort, any possible heading angle between 0° and 360° is investigated to determine the minimum alternative.

4. Path Planning of an Underground Mine Haul Road Using Evolutionary Algorithms

4.1. Discretization of the Path

Between each main node, the algorithm divides the path into three equally spaced sub-sections. In each subsection, the Dubins car travels with a fixed velocity. As a result, distance travelled in unit time is also constant. Dominated by the minimum turning radius constraint, the heading angle changes at a fixed rate in curved sub-sections and kept constant in straight sub-sections. Sub-sections are composed of equally spacing dummy nodes.

4.2. Seed Path Generation

A good prediction for the initial solution space satisfying the kinematical constraints provides better estimations from the Genetic Algorithm (GA) before starting generations [1]. Randomly changing the heading angles and the types of Dubins Paths, GA generates a seed path. Chromosome structure includes the heading angles for each main nodes and the type of Dubins path between two main nodes.

4.3. Genetic Algorithm

4.3.1. Population

Population includes some individuals that are a set of candidate solutions to the optimization problem. Individuals are composed of chromosomes. Mutation and cross-over operators work similar to the biological phenomena to improve the fitness value of the objective function.

This path planning study has a population size of 50. Each chromosome has three parts: the first part is the heading angle of the initial node, the second part is the type of Dubins path and the third part is the heading angle of the initial node.
4.3.2. Classical Crossover Operator

Crossover operator produces a child from two parent chromosomes that contains the genetic heritage of both of the parents. First, two individual strings are randomly selected. Later, a random position is selected to define the genes to be kept and the ones to be migrated to the other string. Finally, two parent individuals are swapped from the randomly selected positions.

4.3.3. Classical Mutation Operator

Mutation helps to escape from a local optimum by randomly disturbing the chromosomes. While cross over operator highlights the strong genes in the current population, mutation operator explores the new ones. Another function of the mutation is keeping the genetic diversity in the population.

4.3.4. Special Mutation Operator for Undesired Region Avoidance (URA)

The path should not violate the undesired regions (UDRs). These regions can be low quality rock zones, structural anomalies, aquifers or just license boundaries. The methodology can be described as follows: For the best fitting 3 individuals in each generation it is checked whether the path violates any of the convex UDRs. In case of a violation, dummy nodes between UDR center point ± trmin are deleted and starting from the pre-intersection point, heading angle is changed to avoid the UDR. Thus, a CCC type Dubins path is used to avoid the UDR. Thinking that a CCC can be either a LRL or RLR, path direction can continue either from the left or from the right directions. The proposed genetic algorithm operator adjusted for mining (URA), which is a special type of mutation operator, can calculate the shorter alternative and result in a degenerate Dubins path. The main contribution of this study to the literature is the proposed special mutation operator. A sample view generated in Matlab and showing the effect of URA operator can be seen in Figure 4.

The green polygon is the UDR and the blue path is the original violating path. The red path represents the adjusted path by URA. By this operator, path designing heuristics are included into the algorithm. These heuristics are beneficial in terms of including the expert opinion in order to make local adjustments on the optimized path.

4.4. Flowchart of the Algorithm

In Figure 5, flowchart of the algorithm describes the working principals of the algorithm. Matlab Global Optimization toolbox with Genetic Algorithm solver is used. Inputs are supplied to the software via the developed Graphical User Interface (GUI). Later, the algorithm recognizes the main node locations and discretize the overall path into dummy nodes. Next, Genetic Algorithm solver generates a seed path for the main optimization stage. This initial attempt only makes use of standard mutation and crossover operators of the genetic algorithm. Using the seed path, a population of 50 individuals is generated. Next, the algorithm calculates the optimum Dubins path between each successive main nodes and computes the fitness value for each path. The best 3 individuals are kept as parents of the next generation. In this stage, UDR violations are detected and the proposed URA operator makes local adjustments on the path. Later, classical mutation and crossover operator are applied to obtain off springs with better objective values. The algorithm drops 90% of the total population if there is no longer decrease in the objective values. A new population is generated and merged with the kept individuals. If the decrease in fitness values stops at the same levels the algorithm terminates, if not, the same procedure is applied until the steady state is reached. By this way, trapping on the local optimum solutions is avoided.

Figure 4 Effect of URA mutation operator
4.5. Verification of the Algorithm

Commonly, there aren’t any benchmark problems for path planning. Analytical solution of path planning in a complex geometry environment without violating kinematical constraints is a challenging issue. In spite of this, validation of the generated algorithm is a necessity.

In this study, an idealized approach is used to prove the validity of the generated algorithm. A simple mine layout with a flat topography and orebody in the shape of a rectangular prism was generated. Crosscut entry coordinates in the x-y plane are the same and elevation difference between the successive crosscut entries are equal depth. The rock mass quality is the same throughout the whole ROI. Sample view of the benchmark problem can be seen in Figure 1.

The genetic algorithm avoids all the forbidden regions and follows the apparent shortest path. Therefore, the algorithm can be expected to be capable of calculating near-optimal solution even in the complex problems.

5. Case Study

5.1. Research Area

Performance of the algorithm was checked on a real underground metallic mine that is located in Albania. Production method is sublevel stoping. There are fourteen sublevels. Including the surface portal, the ramp connects fifteen main nodes. The ramp is constrained by a minimum turning radius of 10 m and a maximum gradient of 17.5 %.

Rock mass quality is recorded for each drill hole in terms of the popular Geomechanical classification system, RMR. In order to represent the whole rock mass, RMR scores were interpolated using inverse distance weighting method. Result is a rock mass quality block model.

The case study has two undesired region, which are a safety buffer between the orebody and the planned ramp and a weak rock zone.

5.2. Results and Discussion

Mine design experts proposed a manual ramp design of 1165 m. The manual design has an average RMR of 53. The proposed genetic algorithm considers all the kinematic constraints, rock mass quality and the undesired regions. Prediction has a path length of 1137 m and the average rock mass quality is 61. Manual ramp and prediction of the algorithm can be seen in Figure 6.

Incidentally, both the path length and the rock mass quality that the ramp is passing inside are improved. However, it may not be the case all the times. Because, objective is minimizing the cost. If the goal is to improve any specific parameter inside the objective function, its weighting should be increased.

Figure 5 Flowchart of the algorithm

4.5. Verification of the Algorithm

Commonly, there aren’t any benchmark problems for path planning. Analytical solution of path planning in a complex geometry environment without violating kinematical constraints is a challenging issue. In spite of this, validation of the generated algorithm is a necessity.

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The genetic algorithm avoids all the forbidden regions and follows the apparent shortest path. Therefore, the algorithm can be expected to be capable of calculating near-optimal solution even in the complex problems.
6. Conclusions

Deep orebodies are preferably accessed by mine ramps due to availability for high mechanization and larger production rates. Development and operating of underground mine ramps have short and long term influence on the mine economy. Most commonly, expert view dominates the manual ramp design. Obviously, optimization of the ramp design may improve the mine economy.

This study proposes an evolutionary algorithm to predict the least cost path for underground mine ramps. Cost function includes path length, rock mass quality, penalty of violating undesired regions and award of passing inside the desired regions. For the purpose of validation, a simple underground mine layout was designed as a benchmark problem. Another stage is testing the performance of the algorithm on a real mine ramp. Results have shown a considerable improvement in the cost parameters of the objective. Result is a near global optimum solution.

Although the global optimum is not guaranteed, the near global optimum offers a significant improvement compared to the manual design. In addition, computational performance is plausible.

References


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Abstract: This article examines current land use and degradation status at the Guizhou Huixin Coal Mine, and examines the current regional topography and ecological conditions to propose appropriate land reclamation methods and technologies that would be suitable for the characteristics of this mine. This project aims at providing theoretical evidence and a reference for future land reclamation efforts of mining sites. Key words: Huixin, coal mine, land reclamation, engineering

1. Introduction

Large quantities of slag and tailings from mining operations not only occupy land and cause environmental pollution, but they also lead to natural disasters, such as sinkholes, landslides, and mudslides. Reclamation to return land damaged by mine excavation, collapse, and compression to a usable state is an important step in maximizing land resource use, reducing environmental damage, minimizing land wastage, and addressing conflicts in supply and demand for land. Mine recovery refers to the use of engineering and biological measures to comprehensively address geological and environmental issues caused by mining activities to achieve geological stability, ecological recovery, and landscape beautification.

2. Methods

This article examines the current land use and degradation patterns at the Guizhou Huixin Coal Mine, and examines the current regional topography and ecological conditions to propose suitable and appropriate land reclamation methods and technologies for the characteristics of this mine. This project aims to provide evidence and a reference for future land reclamation efforts of mining sites.

3. Results and Discussion

3.1 Regional Profile

3.1.1 Topography

The study area is located in a hilly region, with high terrain in the north and low terrain in the south, and widely varying elevation creating a complex landscape. The geological rock groups in the study region can be categorized into the three engineering geological rock groups of hard, soft, and loose.

3.1.2 Climate Conditions

The project region is situated in a humid subtropical climate zone, without extreme temperature variations between the winter and summer seasons. Precipitation increases during the summer, whereas winter and spring are drier. The mean annual sunshine duration is 1095 h, with a maximum of 1344 h and a minimum of 902 h. The mean annual precipitation is 1057 mm, with a maximum of 1340 mm and a minimum of 806 mm. The mean annual temperature of the county is about 14.1\textdegree C with a maximum of 36.6\textdegree C and a minimum of -4.9\textdegree C. The region experiences a long frost-free period of 275 days in a year, on average, with a maximum of 364 days and a minimum of 241 days. An eastern wind prevails in this region, with a mean annual wind speed of 1.5 m/s.

3.1.3 Soil and Flora

The native flora of the project region is subtropical evergreen broadleaf forest, which has experienced significant damage from anthropogenic activity. The regional flora is dominated by secondary broadleaf forest species, such as cedars (Cupressaceae), Chinese firs (Cunninghamia lanceolata), Chinese tallows (Sapium sebiferum), and tung (Cernicia fordii), and rattan shrubs such as forsythia, quitch, Pyracantha crenatoserrata, and chestnut rose (Rosa roxburghii). Planted species include firs, cedars, Chinese red pines (Pinus massoniana), Sabina chinensis, oleander, tallow, tung, peach, plum, pear, and tea plants (Camellia sinesis). Vegetation covers approximately 20\% of the region, and consists primarily of sparse and shrub forest.

3.1.4 Hydrogeological Conditions

The project region is situated in the Wujiang River system in the Yangtze River basin, its primary river being the Huatan River and its tributaries. The Huatan River is situated at the southeast border of the mining zone, flowing from the southwest to the northeast. Its mean volumetric flow rate is 671 L/s, with a high-water mark at 808.0 m; this river is the minimum local erosion datum plane. Located south of the mining zone, a tributary flows from northwest to southeast,
with a mean volumetric flow rate of 2.5 L/s, a maximum volumetric flow rate of 33.4 L/s, and a high-water mark at 817.5 m.

The region exhibits large rises and falls in steep topography, with a moderate geological structure. Coal seams primarily lie below the groundwater level of the minimum erosion datum plane, and natural drainage of groundwater is inhibited by the topographical conditions. This region has a long history of coal mining, and mined outcrops have already formed sizeable goaves that retain water. The hydrogeological conditions of the project area are similar to that of a water-filled karst deposit, i.e., filled via fractures on the top and bottom plates, leading to complex hydrogeological conditions.

3.1.5 Lithostratigraphy

(1) Mid-Permian System (P2)

The lithology of this layer is composed of gray to dark gray, semi-thick to thick beds of fine-grained limestone exposed at the northwest and outer edges of the mine, forming massive and layered structures that contain small amounts of flint, producing brachiopod and odonatan fossils. This layer is in pseudoconformity with the overlying Longtang formation (P31).

(2) Late Permian System (P3)

① Longtan formation (P31): The only coal-bearing stratum in the area. A large portion is exposed in the center of the project area, which contains a set of paralic sedimentary rocks. There are 7-10 coal-bearing strata, with 5 harvestable coal strata, ranging in thickness from 106.36 – 128.07 m, with an average thickness of 114.12 m. This formation is in conformable contact with the overlying Changxing formation.

② Changxing formation (P3c): Exposed in the center of the project area, this formation is 45.50-65.50 m thick, with an average thickness of 54.77 m, and is in conformable contact with the overlying Yelang formation.

(3) Yelang Formation (T1y)

① Shabaowan member (T1y 1): Exposed band in the northern section of the project area, with a thickness of 7.88-11.161 m, and an average thickness of 9.17 m. This formation is in conformable contact with the underlying formation.

② Yulongshan member (T1y 2): Wide exposure at the southeastern section of the project area, with a thickness of 133.99-181.41 m, and an average thickness of 161.12 m.

③ Jiujitan member (T1y3): Exposed in the southeastern section of the assessment area.

(4) Quaternary System (Q)

This system is mainly composed of loose, collapsed debris, slope deposits, alluvial valley deposits, and clay, with a thickness of 0-10 m. This stratum is mainly located in the depressions and on gentle slopes in the project area, and is in angular unconformity with the underlying strata.

3.2 Current Land Use Patterns

The Huixin Coal Mine is located in southeastern Jinsha County, approximately 47 km from the county seat, and is under the municipal administration of Shatu Town. Paved roads are the primary source of transportation in the mining zone. It is situated approximately 25 km east of the Guizhou – Zunyi Highway, 11 km from Shatu Town, 47 km from Jinsha, 50 km from the Qianbei Power Plant, and 70 km from Zunyi City. The mining zone covers an area of 2.515 km². See Table 1 for current land use patterns, and Table 2 for current land degradation.

Table 1 - Current Land Use Patterns at Huixin Coal Mine

<table>
<thead>
<tr>
<th>Ownership</th>
<th>Primary Category</th>
<th>Area (hm²)</th>
<th>Percentage (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shatu Town</td>
<td>Farmland</td>
<td>35.11</td>
<td>13.96</td>
</tr>
<tr>
<td></td>
<td>Woodland</td>
<td>40.23</td>
<td>16</td>
</tr>
<tr>
<td></td>
<td>Grassland</td>
<td>67.2</td>
<td>26.72</td>
</tr>
<tr>
<td></td>
<td>Transportati on</td>
<td>0.88</td>
<td>0.35</td>
</tr>
<tr>
<td></td>
<td>Water bodies</td>
<td>28.13</td>
<td>11.18</td>
</tr>
<tr>
<td></td>
<td>Town and Mining land</td>
<td>60.23</td>
<td>23.95</td>
</tr>
<tr>
<td></td>
<td>Others</td>
<td>19.72</td>
<td>7.84</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>251.5</strong></td>
<td><strong>100.00</strong></td>
</tr>
</tbody>
</table>

3.3 Establishment of Responsibility for Reclamation of Degraded Mining Land

According to the combination of site investigation and data collection, the total land degradation area of the mining zone is 182.67 hm². As there are no permanent construction sites, the entire degraded area is designated as an area for reclamation.

Table 2 - Current Land Degradation at Huixin Coal Mine

<table>
<thead>
<tr>
<th>Unit of Degradatio n</th>
<th>Type of Degradation</th>
<th>Degree of Degradati on</th>
<th>Area (hm²)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Current open-pit mine</td>
<td>Digging</td>
<td>Severe</td>
<td>35.72</td>
</tr>
<tr>
<td>Outer dump site</td>
<td>Compaction</td>
<td>Severe</td>
<td>24.99</td>
</tr>
<tr>
<td>Inner dump site</td>
<td>Compaction</td>
<td>Severe</td>
<td>112.06</td>
</tr>
<tr>
<td>Industrial site</td>
<td>Compaction</td>
<td>Severe</td>
<td>9.9</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td></td>
<td><strong>182.67</strong></td>
</tr>
</tbody>
</table>

3.4 Reclamation Process

3.4.1 Land Leveling

(1) Filling of fractures and cracks

According to previous analysis on the predicted degree
of degradation from land subsidence, combined with experience from other mines in the region as a reference, the average volume of earthwork to be filled per hectare of subsidence ranges between 80.00 m³ – 150 m³. The average required fill per hectare is 120.0 m³.

The area of degradation by ground fissures is 68.4063 hm², with a required fill volume of 8208.756 m³.

Agricultural soil is selected from higher ground surrounding the fissures to use for filling. Manual labor will be used to transfer and compact the soil to fill the fissures.

(2) Leveling of subsided land

According to the actual conditions at the Huixin Mine, the volume of earthwork required to level the subsidence fissures in the impacted area is 48069.791 m³.

(3) Construction of earthwork lynchets

Construction of earthwork lynchets is planned, with the following dimensions: top and bottom widths of 0.25 m, and a ridge height of 0.40 m. The ratio of inner to outer slopes is 1:1, with 0.025 m³ of earth excavation. The earthwork volume for lynchet construction is 0.10 m, and the planned length of the structure is 5000 m.

(4) Rural roadwork construction

Production road: Construct a 2000 m long production road. The road surface width is 1.0 m, and the roadbed width is 1.5 m. The road surface is 0.25 m above ground level, with a 0.15 m thick gravel cushion, and a 0.10 m thick C20 concrete pavement. The road sides are lined with stone rubble that is 0.20 m wide on each side.

### 3.4.2 Agricultural Hydraulic Systems

(1) Water reservoir

30 m³ water reservoirs are designed for construction, and the total number of reservoirs needed is as follows: $68,4063 \times 15 \times 5 \times 30 \div 3 = 57$ reservoirs to be constructed (3 refills).

(2) Drainage ditch

The drainage ditch is designed with a rectangular cross-section and dimensions of: 0.50 m bottom width, 0.6 m depth, 0.3 m thick walls, and 15 cm thick C15 cement plates as the base. This proposed drainage ditch is to be 3000 m long, constructed along the area of fissure damage.

### 3.4.3 Fertilization Process

To address the issue of poor fertility of newly reclaimed dry land, the leguminous plant, vetch (*Vicia sativa*), is to be planted as green manure to improve soil fertility. Each acre of land will be sown with 6 kg of green manure seeds, and fertilized with nitrogen, phosphorous, and potassium. For each acre of agricultural land, 6.20 kg of nitrogen, 2.40 kg of phosphorous, and 5.80 kg of potassium fertilizers are used to improve soil fertility after reclamation. A total of 68.4063 hm² of land within the area affected by subsidence fissures will be fertilized.

### 3.4.4 Monitoring

Monitoring networks are established to monitor ground fissure width, length, and surface displacement. These are monitored in residences severely affected by mining operations. Dadongba, Shaba, and Yantoushan will each have one monitoring site.

### 4. Conclusions

Using Guizhou Huixin Coal Mine as an example, the topography and geomorphology characteristics, influencing factors, and land surface variation patterns of mining zones were studied to investigate land reclamation processes that are suitable for open-pit mining operations. Effectively reducing the geological and environmental damage caused by mining operations can promote sustainable development of the mining industry and achieve effective unification of economic and environmental objectives.

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4. Funds of Governor of Guizhou Province for Excellent Education Talents (QIAN SHENG HE ZHUAN ZI [2008]32) Study on the damaged ecology in mining areas of Liupanshui Prefecture, Guizhou;

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Abstract — 3D model of phased development of optimal contours of oval-shaped open pit mines is offered. It is assumed that with sufficient accuracy its volumetric contour of the open pit mine is interpolated by an elongated elliptic hyperboloid. Formulas for mineral resources are derived and optimal volumes of overburden are determined depending on the mining phase. In this case, the total number of mining phases is set in advance. The stripping ratio is used as a quality criterion of the optimization task. The problem of optimal control is solved using the Bellman function in dynamic programming. Necessary calculation formulas are obtained in the final form by solving the optimization problem. Their simplicity and justification of each conclusion ensure that the results of this study can be successfully applied in practical calculations of the design and planning of mining operations in open pit mining.

Keywords: open pit mining, oval shaped open pit mines, 3D model, open pit mine contours, steeply dipping mineral occurrences.

1. Introduction

One of the constraints of implementing innovative technologies in open pit mining is the existing difficulties of their implementation in design practice. We considered the case of round shaped mineral occurrences when creating a 3D model for the gradual development of the pit contours [1-3]. The open pit mine turned out to have a form of a truncated cone, which is characteristic mainly in mining of kimberlite mineral prospects. It was possible to solve this problem on digital models of open pit mines using phased contours of an open pit mine along the horizontal sections of a one-dimensional spline of the second order, and a two-dimensional spline in depicting the lateral surfaces of the ore body in 3D modelling. The inclination of the open pit edges by mining phases of steeply dipping open pit mines has been taken into account by establishing an optimal radius of the contours of the lateral surfaces on each of them while optimizing the volumes of overburden and ore, as well as the step-by-step determination of the stripping ratio. However, practice shows that most of the open pit mines are oval.

Modern design of production at a mining enterprise, planning and allocation of resources are based on mathematical programming models. In practice, the optimal solution cannot be achieved without considering all possible combinations and permutations of the extraction sequence. Methods of studying operations have limited application in large-scale open pit mining operations, as the number of variables becomes too large. A scientific work [4], in which a hybrid basis for simulation of the problem of open pit mining planning has been developed and tested, was of great interest in solving this project's tasks.

Pushback-approach [5] can increase the NPV of the project by minimizing the stripping ratio in the early years of operation of a mine.

Approaches to solving the problem of operational planning of open pit mining operations with the dynamic distribution of technological vehicles, as described in a scientific work [6], will be useful when describing the dynamics of the relocation of the working zone along steep open pit edges. A hybrid algorithm, developed and tested in real conditions, combines the characteristics of two metaheuristics: Randomized Adaptive Search Procedures and General Variable Neighborhood Search.

The described models allow determining some of the desired parameters of the open pit mine during the technical-economic justification phase in the first approximation.

Testing of the developed technology for safe intensive development of working zones along steep open pit edges on the digital model of the Lomonosov iron ore mine showed that there are reserves for minimizing stripping volumes in the phased mining of steeply dipping mineral occurrences without the formation of temporarily non-operating open pit edges [7-9]. A methodology was developed to optimize the phased volumes of overburden and ore on the 3D model of an oval-shaped open pit mine in order to improve the design quality of this technology.

2. Material and Methods

2.1 Characteristics of oval shaped mineral occurrence
We should consider the case when the mineral occurrence has an oval shape. We assume that the boundary of this region is depicted by an elongated elliptic hyperboloid. The elongated axis of the ellipsoid is depicted as a longitudinal, and perpendicular to it, axis of the elliptical hyperboloid will be the transverse axis. The horizontal cross section of hyperboloid is ellipse, the largest axis of which coincides with longitudinal direction (Ox axis), the smallest axis of hyperboloid coincides with transverse direction of ore shoot (Oy axis). The origin is located in the center of uppermost ellipse, Oz axis passes through it vertically downwards (Fig. 1). After fulfillment of necessary preliminary conditions, volumetric surface of ore shoot would be described by the following equation:

\[ \frac{x^2}{a^2} + \frac{y^2}{b^2} + \frac{z^2}{c^2} = 1 \]  

\[ \frac{x^2}{a^2} + \frac{y^2}{b^2} = 1 \]  

On uppermost boundary of ore shoot \( z = 0 \). Horizontal cross section of this surface with plane \( z = 0 \) is described by following expression:

\[ \frac{x^2}{a^2} + \frac{y^2}{b^2} = 1 \]  

Figure 1. Oval shape of described ore shoot

It is assumed that this cross section is an extended ellipse, largest axis of which is located on Ox axis, with \( a \gg b \) (Fig. 2). The area of the figure would be determined by limiting line (2). We should switch to polar coordinates:

\[ x = ar \cos \varphi, \quad y = br \sin \varphi \]  

Figure 2. Horizontal cross section of ore shoot on plane \( z = 0 \)

We obtain an equation of ellipse in polar coordinates system (3) by putting into equation (2). It is written in the form \( r = 1 \).

Geometric figure 2 is symmetrical relative to Ox and Oy axes. Therefore, area of the figure (Fig. 2) located in the first quadrant would be:

\[ S = \frac{ab}{4} \int_{0}^{\pi/2} \sin \varphi d\varphi = \frac{ab}{4} \left[ -\cos \varphi \right]_{0}^{\pi/2} = \frac{ab}{4} \]

We obtain formula for calculating ellipse area:

\[ S = \frac{ab}{4} \]  

Hence, the product of the semi-axes of the ellipse is the area of the ellipse.

Using this formula, we can calculate the area of any horizontal section of an elliptic hyperboloid (Figure 3). Thus, the section of the surface (1) on the plane would be written as following:

\[ \frac{x^2}{a^2} + \frac{y^2}{b^2} + \frac{z^2}{c^2} = 1 \]

here, \( z \) - fixed value. Therefore, semi-axes of the ellipse shown in Fig. 3 would be written as following:

\[ a(z) = a \sqrt{\frac{z^2}{c^2} + 1}, \quad b(z) = b \sqrt{\frac{z^2}{c^2} + 1} \]  

where, \( 0 \leq z \leq h, \ h - \) depth of ore shoot

Figure 3. Arbitrary horizontal cross section of ore shoot

Using formula (4), area of the geometric figure from Fig. 3 is determined by following formula:

\[ S(z) = ab \sqrt{\frac{z^2}{c^2} + 1}, \ 0 \leq z \leq h \]

Total volume of minerals is determined using following expression:

\[ V = \int_{0}^{h} S(z) dz. \]

Calculating this integral:
Thus, the total volume of extracted ore would be:

\[ V = ab \int_0^h \left( \frac{z^2}{c^2} + 1 \right) dz = ab \left( \frac{z^3}{3c} + 1 \right) \]

(6)

Assuming that on mining phase \( t + 1 \) minerals are extracted from \( z_i \) to \( z_{i+1} \) by \( O_z \) axis. In this case, volume of extracted ore would be calculated using formula:

\[ V_{(t+1)} = ab \int_{z_i}^{z_{i+1}} \left( \frac{z^2}{c^2} + 1 \right) dz = ab \left( \frac{z_i^3}{3c} + 1 \right) \]

(7)

If \( a \) and \( b \) are known (Fig. 2), then \( c \) is calculated from the following equation:

\[ \frac{x^2}{a^2} + \frac{y^2}{b^2} = \frac{h^2}{c^2} + 1 \]

It is an equation of a boundary of lower base of ore shoot from Fig. 1. In order to determine \( c \), values of \( a(n) \) or \( b(n) \) should be known, where \( a(n) \) – longitudinal semi-axis of ore shoot base; \( b(n) \) – transverse semi-axis of ore shoot base. Taking \( a(0) = a \) and using equation (5), we will have the following expression:

\[ a(n) = a(0) \sqrt{ \frac{h^2}{c^2} + 1 } \]

We will obtain following expression by solving the last equation relative to \( c \):

\[ \frac{a^2(n)}{a^2(0)} - 1 = \frac{h^2}{c^2} \]

Consequently,

\[ c = \frac{a(0)h}{\sqrt{a^2(n) - a^2(0)}} \]

(8)

2.1.1 Characteristics of oval shaped mineral occurrence by overburden

We consider that only stripping works are performed at phase zero of mining. In this case, the overburden volume is an elongated cone, in which the long axis coincides with the longitudinal direction of the open pit mine, and the short axis coincides with the direction of the transverse axis of the open pit mine (Fig. 4). The depth of phase zero of stripping is set as \( H \). Zero slope angle \( \gamma_0 \) of phase zero of stripping (Fig. 5). Surface of the cone would be:

\[ \frac{x^2}{A^2} + \frac{y^2}{B^2} = \frac{z^2}{C^2} \]

(9)

\[ \frac{x}{A} - \frac{z}{C} = 0 \]

(10)

Second line is symmetrical relative to \( O_z \) axis. We obtain following equation from Fig. 6:

\[ z = a_0 f g \gamma_0 \].

Figure 4. 3D model of ore shoot with covering overburden rocks during phase zero of mining

Figure 5. Vertical longitudinal cross section of overburden during phase zero of mining
We obtain coordinates of point \( D(a_0, a_0 \tan \gamma_0) \).

Line (10) passes through this point. Thus, we get:

\[
\frac{a_0}{A} \frac{a_0 \tan \gamma_0}{C} = 0
\]

then,

\[
C = A \tan \gamma_0 \quad \text{or} \quad A = C \tan \gamma_0
\]  

(11)

Reviewing cross section of a cone (9) on plane \( x = 0 \), we obtain:

\[
A = C \tan \sigma_0
\]  

(12)

Equations (11) and (12) are put into (9). Then we get the following equation:

\[
\frac{x^2}{C^2 \tan \gamma_0} + \frac{y^2}{C^2 \tan \sigma_0} = \frac{z^2}{C^2},
\]

or

\[
(x \tan \gamma_0)^2 + (y \tan \sigma_0)^2 = z^2
\]  

(13)

We obtain following equations from Fig.7:

\[
h_0 = a_0 \tan \gamma_0, \quad h_0 = b_0 \tan \sigma_0.
\]

Figure 7. Complemented overburden frustum

Then:

\[
\tan \sigma_0 = \frac{a_0}{b_0} \tan \gamma_0.
\]

Putting into (13), we get following equation:

\[
x^2 \tan^2 \gamma_0 + \frac{y^2 a_0^2}{b_0^2} \tan^2 \gamma_0 = z^2,
\]

or

\[
\frac{x^2}{a_0^2} + \frac{y^2}{b_0^2} = \frac{z^2}{h_0^2}.
\]

In general case, equation of cone corresponding to phase \( t \) is written as:

\[
\frac{x^2}{a_t^2} + \frac{y^2}{b_t^2} = \frac{z^2}{h_t^2}, \quad t = 0, 1, 2, \ldots, n.
\]  

(14)

where, \( h_t = a_t \tan \gamma_t \).

Volume of frustum corresponding to phase \( t \) is determined by the following formula:

\[
V_{gm}(t) = \int_{z_i}^{z_f} S(z)dz.
\]

Figure 8. 3D model view after first mining phase

Putting it into formula of \( V_{gm} \), we get:

\[
V_{gm}(t) = \frac{a_t b_t}{h_t} \left[ \frac{z_f^{a_t \gamma_t} - z_i^{a_t \gamma_t}}{a_t \gamma_t} \right] + \frac{a_t b_t}{3h_t} \left[ \frac{z_i^{a_t \gamma_t} - z_f^{a_t \gamma_t}}{a_t \gamma_t} \right] = \frac{1}{3} a_t b_t \tan \gamma_t \left[ (z_f + h_t)^{a_t \gamma_t} - (z_i + h_t)^{a_t \gamma_t} \right]
\]

with \( t = 0, 1, 2, \ldots, n \).

If \( t = n \), then we get a formula for calculation of total volume of rock mass:

\[
V_{b}(n) = \frac{1}{3} b_n \tan \gamma_n \left[ (h + H)^{a_n \gamma_n} - h_n^{a_n \gamma_n} \right]
\]

here, \( h_n = a_n \tan \gamma_n \).

The overburden volume is determined by the following formula during phase zero:

\[
V_{b}(1) = V_{gm}(1) - V_{gm}(0) - V_{f}(1)
\]

The overburden volume during next phase (Fig. 9) would be:

\[
V_{b}(2) = V_{gm}(2) - V_{gm}(1) - V_{f}(2)
\]
Using expression, taking (Fig. 9) as a starting point. 

Inequalities hold for all values of \( z_{r+1} \). Inequality \( f''(z_{r+1}) > 0 \) shows that function \( y = f(z_{r+1}) \) is increasing. \( f''(z_{r+1}) > 0 \) - graph of this function is a concave line. It follows from Fig. 9 that equation (17) has single solution. Root of this equation is determined by method of Newton. We could take \( z_0 \) (Fig. 9) as a starting point.

Following equations are solved to find optimal values of \( a_i \) and \( b_i \):

\[
\frac{a_i^2}{a_0^2} = \frac{z_{i+1}^3}{c^3} + 1 \quad \text{and} \quad \frac{b_i^2}{b_0^2} = \frac{z_{i+1}^3}{c^3} + 1 \,.
\]

Then:

\[
a_{i+1} = a_0 \left( \frac{z_{i+1}^3}{c^3} + 1 \right) \quad \text{and} \quad b_{i+1} = b_0 \left( \frac{z_{i+1}^3}{c^3} + 1 \right) \,.
\]

We use formula (15) to find optimal values of angles \( \gamma_i \) and \( \sigma_i \). Rewriting it:

\[
V_b(t) = b_c \frac{1}{c} \left[ z_i + H \right]^3 - h_i^3 - b_c \frac{1}{c} \left[ z_{i+1} + H \right]^3 - h_{i+1}^3 - a_i \frac{1}{c} \left( \frac{z_{i+1}^3}{c^3} + z_i - \frac{z_{i+1}^3}{c^3} + \frac{z_i^3}{c^3} \right).
\]

In this equation \( h_i = a_i \gamma_i \). Using expression:

\[
f'(z_{r+1}) = a_0 b_0 \left( \frac{z_{i+1}^2}{c^2} + 1 \right)
\]

Equation (17) has single solution. We rewrite (17) to prove it:

\[
f(z_{r+1}) = a_0 b_0 \left( \frac{z_{i+1}^2}{c^2} + \frac{z_i^3}{3c^2} - z_i \right) - V(t+1)
\]

First derivative of this equation:

\[
f'(z_{r+1}) = a_0 b_0 \left( \frac{z_{i+1}^2}{c^2} + 1 \right)
\]

Second derivative:

\[
f''(z_{r+1}) = \frac{2}{3} a_0 b_0 z_{i+1}.
\]
A cubic equation is obtained relative \( f_i = \tan \gamma_i \):

\[
a_i^3 f_i^3 + A_i f_i - (z_i + H)^3 = 0,
\]

The \( f_i \) is determined at \( t = 1, 2, \ldots, n - 1 \).

Optimal values of slope angles of open pit edges in longitudinal direction are determined by following expression using solution of cubic equations:

\[
\gamma_i = \arctan(f_i) \quad t = 1, 2, \ldots, n - 1.
\]

We will use following equations to find slope angles in transverse direction of open pit contours:

\[
a_i \tan \gamma_i = b_i \tan \sigma_i, \quad t = 1, 2, \ldots, n - 1.
\]

The \( \sigma_i \) is determined at \( t = 0, 1, 2, \ldots, n - 1, n \).

We will use following variables to find external dimensions of oval shaped open pit mines: \( M_i \) - ellipse axis on its surface in longitudinal direction, \( N_i \) - ellipse axis on its surface in transverse direction (Fig.10). Trigonometric equations are derived to find parameters of interest using optimal values of \( z_i, \gamma_i \) and \( \sigma_i \).

\[
M_i = a_i + (z_i + H)g \gamma_i; \quad N_i = b_i + (z_i + H)g \sigma_i
\]

where, \( t = 0, 1, 2, \ldots, n - 1, n \).

Optimization of the mining operations modes for the created 3D model of phased mining of steeply dipping mineral occurrences has its own peculiarities. It is necessary to ensure an even distribution of ore reserves being extracted with the minimization of the phased stripping ratio on the perimeter of the open pit edges during each mining phase.

Let, \( S_w \) be a total volume of extracted rock mass, \( S_o \) - volume of extracted ore, \( u(t) \) - volume of extracted rock mass during phase \( t \), \( v(t) \) - volume of extracted ore during phase \( t \), then stripping ratio would be determined by using following formula:

\[
k(t) = \frac{u(t)}{v(t)} \quad t=1, 2, \ldots, n.
\]

It is known that for positive values of \( u(t) \) and \( v(t) \) minimum values of functions \( k(t) \) and \( k^2(t) \) are reached in the same points in the domain of functions \( u(t) \) and \( v(t) \). Thus, further we will solve the following problem:

\[
J(u, v) = \sum_{i=1}^{n} \frac{u^2(t)}{v^2(t)} \to \min \quad (18)
\]

It follows from a practical standpoint of rock mass volume \( u(t) \), that volume of rock mass should be minimum during each phase. On the other hand, from mining operations technology standpoint, all volume of rock mass and ore should be extracted during last phase \( n \). Thus, we derive following expressions:

\[
\begin{align*}
x(t) &= x(t-1) + u(t), \\
y(t) &= y(t-1) + v(t), \quad t = 1, 2, \ldots, n,
\end{align*}
\]

where, \( x(t) \) - total volume of extracted rock mass including phase \( t \); \( y(t) \) - total volume of extracted ore during phase \( t \).

We also have following equations:

\[
\begin{align*}
x(n) &= S_w, \\
y(n) &= S_o
\end{align*}
\]

Setting \( x(0) \) as volume of extracted rock mass during phase zero of mining, \( y(0) \) - volume of extracted ore during phase zero of mining. Therefore, we get:

\[
x(t) \in [x(0); S_w] \equiv X, \quad y(t) \in [y(0); S_o] \equiv Y,
\]
\[ u(t) \in [0, S_x - x(0)] = U, \quad v(t) \in [0, S_y - y(0)] = V \quad (21) \]

Here, \( X \), \( Y \) and \( U \), \( V \) are permissible values of states \( x(t) \), \( y(t) \) and controls \( u(t) \), \( v(t) \). It means, that optimum solution of systems \((19)-(21)\) should be found including following expression:

\[ x(t) \in X, \quad y(t) \in Y, \quad u(t) \in U, \quad v(t) \in V; \quad t = 1, 2, 3, \ldots, n \quad (22) \]

Vector notation is used sometimes:

\[ \vec{x} = (x(0), x(1), \ldots, x(n)) \quad \vec{y} = (y(0), y(1), \ldots, y(n)) \]

In our case these are states of extracted rock mass and ore volumes. Each component of vectors \( \vec{x}, \vec{y} \) has positive value, with following inequality:

\[ x(0) < x(1) < x(2) < \ldots < x(n), \quad y(0) < y(1) < y(2) < \ldots < y(n) \]

Similarly,

\[ \vec{u} = (u(1), u(2), \ldots, u(n)), \quad \vec{v} = (v(1), v(2), \ldots, v(n)) \]

- group of control vectors, which is called control (management) of mining operations.

Function \( J(u, v) \) is separable. Hence, dynamic programming method developed by Bellman [13-16] was used to minimise function \((18)\). Its implementation is described in the scientific works [11, 12].

### 4. Conclusions

A 3D model for the phased development of contours has been created for oval shaped open pit mines. The volumetric contour of the mineral was interpolated by an elliptical hyperboloid of the oval form. It is assumed that such a method of interpolation satisfies the practice of open pit mining steeply dipping oval mineral occurrences when developing a working zone along steep open pit edges without temporary non-operating open pit edges with sufficient accuracy. It is clear that the functional type of interpolation and the substantiation of its reliability is an independent task. It is necessary to use the actual data of the studied digital model of the open pit mine. The optimal control problem discussed in this paper depends on the solution of the cubic equation of nonlinear algebra. It was proven that the cubic equation has a unique solution. This fact indirectly proves the correctness of the chosen method of solving the studied problem. The algorithm for solving the cubic equation developed in this paper is very simple. It can be implemented using any programming language, in particular C++ or Java. Using the roots of cubic equations, the optimum values of the stripping ratio, the slope angles of the open pit edges and the volumes of stripping operations are determined, depending on the mining phase.

### References


DEVELOPMENT OF A NEW FLOTATION REAGENT BASED ON RICE FLOTTATION

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Abstract - Liquid organic product produced from rice husk using pyrolysis process was tested as a flotation reagent in the flotation process of polymetallic lead-zinc ore in the pilot plant under semi-industrial conditions. The optimal conditions for lead and zinc flotation cycles such as pH, dosages of reagents, flotation time were determined. It was shown that a new flotation reagent did not affect technological characteristics of zinc flotation as compared to the routine flotation process. At the same time it improved the lead flotation characteristics. It was found that addition of 15-20 g/t of studied liquid organic product increased lead recovery by 1.1% and quality of lead concentrate by at least 0.5%.

Key words: lead-zinc ore, flotation, rice husk, liquid organic product

1. Introduction
Nowadays, investigation of alternative sources of raw materials and creation of green technologies of production of new materials becomes more and more important. Therefore, using renewable raw materials in the manufacturing process gains a great interest. Rice husk is a unique large-tonnage waste that is produced as a side product of rice cultivation. More than 100 countries worldwide are engaged in rice cultivation with up to 600 million tons per year cultivation volume. Thus, the annual world output of rice husk is 120 million tons [1]. Rice husk is a plant waste which includes silica and, as a result, it does not decompose, but accumulates in the dumping areas creating environmental problems. Many research groups work on rice husk processing [1-4]. It was shown that rice husk can be utilized for the production of different valuable products [5-9]. However, the majority of novel technologies were tested only in laboratory, but efficient industrial technologies are not yet available. Therefore, processing of rice husk is an acute problem. Although in Kazakhstan rice is cultivated in only a small amount (annual mass of rice husk waste achieves about 100 thousand tons) the problem of rice husk utilization exists there as well. It is known that the rice husk composition varies depending on the sort of rice, the geographical location and agronomic method of rice cultivation [10]. In reality these distinctions are not significant and they are mainly affected by the method of analysis. In general, rice husk consists of cellulose, hemicellulose, lignin, mineral components, and others. Therefore, the quality and the properties of the end products produced from rice husk primarily depend on technological parameters of rice husk processing.

As we reported earlier, the complex technology of thermal recycling of rice husk, including a necessary equipment, has been developed [11-12]. It was shown in our previous works that the main advantage of the developed technology was its complexity. Two end products were produced with almost the same yield value (about 35% from the rice husk). The first product was a solid silicacarbon consisting of nanoparticles of carbon and silicon dioxide. Due to the fact that silicacarbon is a nanocomposite it has unique properties and can be used in different fields of industry and agriculture [10]. Another product is a liquid product representing an aqueous solution of organic compounds. It can be used as a raw material for production of clear organic compounds. However, it is an expensive process. Therefore, it is important to find a reasonable way for its utilization. Based on its composition it is expected that the liquid product can be used as a promising flotation reagent. Moreover, we have reported that liquid organic product has properties of a blowing agent similar to such known reagent as methylisobutylcarbinol [13].

The objective of the current work is to test liquid product produced from rice husk as a flotation reagent in the flotation process of polymetallic lead-zinc ore in the pilot plant under semi-industrial conditions.

2. Materials and methods
Rice husks (RH) obtained from Almaty (5000 kg) and Kyzylorda (5000 kg) regions were used as a raw material. Rice husk was, firstly, washed and then dried to a residual moister of 3-5%. The rice husk pyrolysis process was carried out using a laboratory equipment and a pilot plant with yield of rice husk of 300 kg per day. The pyrolysis of rice husk under laboratory conditions was conducted in the vertical elevator oven with the reactor...
made of stainless steel with the upload volume of 200 g in the flue gases atmosphere. The heating temperature was increased till 400 °C with the heating speed 15°C/min. Next, the oven was kept for 30 minutes at the following temperature intervals 400-550, 550-650, 650-750, 750-850, 850-950 °C. The steam gas was concentrated in the condensation system. Non-condensable gas was directed to the exhaust ventilation tube.

Heating of rice husk in the pilot plant was performed in the flue gases atmosphere at 400 °C and 600 °C for 30 min in the rotating reactor into which the raw material was continuously supplied through the feeding bunkers. Final solid product was removed from the opposite end and it was collected in the water-cooled bunker. Flue gases were allowed to enter the condensation system and they were collected in the condensate collector as a liquid product (LP). Non-condensable gases were used as a gaseous fuel to dry rice husk in the head of this process before rice husk thermal decomposition. Produced liquid product was stored in 100 liters plastic barrels. 5 liters of the liquid product from every barrel were poured into an extra barrel to prepare an average sample.

Compositions of the produced liquid product samples were analyzed by gas phase chromatography mass spectrometry (GC-MS). GC was performed with Agilent 6890 instrument with a mass selective detector.

Lead-zinc ore from Akzhal deposit was used to test the liquid product as a flotation reagent in the flotation process. Preparation of the initial ore sample was carried out according to the standard procedure, including crushing, screening, homogenization of ore by the methods “bin-transporter-bin” and “ring-cone” (Figure 1).

To study of mineral, chemical and grain compositions an average sample of 20 kg was taken from an average ore sample with a total mass of 2000 kg and grain size of -2 + 0 mm. X-ray (DRON-4) and chemical analysis characterized mineral and chemical compositions of the ore, respectively. The grain composition was determined by a wet sieve method, and the distribution of lead, zinc, iron, and silver was studied for all classes of fineness.

The bulk of the ore sample of 1980 kg was used for technological tests. Tests of the flotation enrichment of the Akzhal ore using a new reagent were carried out according to the factory technological scheme (Figure 2). A new liquid organic product was tested instead of the T-92 reagent. At the same time, the consumption of the new reagent in all operations fully corresponded to the consumption of T-92. The main, control and recleaning operations of lead and zinc flotation were carried out in four-chamber flotation machines “Mekhanobr” with the chamber volumes of 12 dm³, each. Preliminary laboratory studies were carried out in flotation machines with the following chamber volumes, dm³: 3.0; 1.5, 1.0; 0.75; 0.5.

3. Results and Discussion

3.1. Characteristics of the liquid organic product

Liquid product samples produced under various conditions (chapter 2) split into two layers upon storage. The upper layer (water soluble part) was a light-brownish liquid, the lower layer consisted of tar oils. It was found that with the growth of the temperature of the thermal destruction of rice husk overall yield of condensate increases from 33% at 400-550°C to 38% at 850-950°C. Within the same temperature interval the amount of tar oils in the condensate composition was increased from 12% to 19%.
Table 3 - Results of granulometric analysis of Akzhal lead-zinc ore crushed to -2+0 mm

<table>
<thead>
<tr>
<th>Classes of fineness, mm</th>
<th>Yield, %</th>
<th>Content, %</th>
<th>Distribution, %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Pb</td>
<td>Zn</td>
<td>Fe</td>
</tr>
<tr>
<td>0.044+0.074</td>
<td>5.17</td>
<td>0.41</td>
<td>18.44</td>
</tr>
<tr>
<td>0.074+0.044</td>
<td>5.17</td>
<td>0.41</td>
<td>18.44</td>
</tr>
<tr>
<td>0.1+0.074</td>
<td>5.00</td>
<td>0.43</td>
<td>21.77</td>
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<tr>
<td>0.2+0.1</td>
<td>5.12</td>
<td>0.46</td>
<td>20.47</td>
</tr>
<tr>
<td>0.5+0.2</td>
<td>5.45</td>
<td>0.39</td>
<td>20.25</td>
</tr>
<tr>
<td>1+0.5</td>
<td>5.30</td>
<td>0.38</td>
<td>19.30</td>
</tr>
<tr>
<td>2+1</td>
<td>5.20</td>
<td>0.42</td>
<td>16.95</td>
</tr>
<tr>
<td>44+0</td>
<td>0.41</td>
<td>5.17</td>
<td>18.44</td>
</tr>
</tbody>
</table>

Figure 2. Technological scheme of Akzhal ore flotation in the pilot plant.
GS-MS analysis of the of the chemical composition of the water soluble part of liquid products derived at various temperatures under the laboratory conditions from Kyzylorda rice husk after keeping them for 2 months showed that the products have nearly the same composition. These are aqueous solutions of such compounds as carboxylic acids, phenols, ketones, closed-chain aliphatic hydrocarbons, heterocyclic compounds, esters and alcohols. Moreover, the analyzed samples possess similar composition. Average content of carboxylic acids was 22%, phenols – 14%, ketones – 12%, closed-chain aliphatic hydrocarbons – 5%, heterocyclic compounds – 4%, alcohols and esters – 5%, respectively. Water content was between 21-27%. Average amount of the unidentified components in some of the samples was up to 15%.

Acetic acid formed the main part of carboxylic acids (94%). Phenols were represented by dihydroxybenzenes (33%; ~23% of them consist of pyrocatechin together with derivatives), phenol itself (20%) and 2-metoxynaphthalene (16%). Ketones were represented by 1-hydroxy-2-propanone (69%); 1-hydroxy-2-butanone and acetone with 69%, 19%, and 10%, respectively. Closed-chain aliphatic hydrocarbons included mainly (78%) 2-cyclopenten-1-one and its derivatives. In the group of heterocyclic compounds mainly the derivatives of furan (75%) were present with almost 50% of furfural; 25% (on average 4-5% of the condensed mass) belongs to pyridine with derivatives.

Similarity in qualitative and quantitative compositions of liquid product samples derived at different temperatures may be considered as a factor of relative stability of the systems under consideration that appears to be of great importance from the technological point of view.

During the pyrolysis of rice husk from Kyzylorda region in the pilot plant at 400°C the condensate was derived, which contained ~35% of carboxylic acids, ~13% of phenols, ~12% of ketones and aldehydes, ~5% of heterocyclic compounds, ~3% of closed-chain aliphatic hydrocarbons, ~1% of alcohols and esters, up to 25% of water and ~6% of unidentified components (Table 1). As it can be seen its composition is very close to the composition of liquid products derived under the laboratory conditions. The main difference was found in the number of carboxylic acids. Probably, it can be explained by the conditions of the pyrolysis, which was performed in a steady state in the laboratory facility and in a continuous mode in the pilot plant.

A liquid product of almost similar composition was derived in the pilot plant during pyrolysis of rice husk from Almaty region at 400°C (Table 1). Prevalence of acetic acid (83% and more) in the content of carboxylic acids was typical for both samples. Phenol fractions are represented by phenol itself (20-25%) and its derivatives. Ketone and aldehyde groups were mostly represented by 1-hydroxy-2-propanone (61-69%).

A liquid product derived during pyrolysis of rice husk from Almaty region at 600°C significantly differs from the other samples by high content of carboxylic acids (56% of formylic acid and 43% of acetic acid) and by almost complete absence of phenols and closed-chain aliphatic hydrocarbons. Along with this, the content of alcohols and esters in its composition increased, with more than 70% of 2-hydroxy-propanoic acid methyl ester.

Thus, the liquid product of rice husk pyrolysis represents a mixture of organic compounds which have heteropolar structure, i.e. contains hydrophobic (aliphatic and closed-chain hydrocarbon radicals) and hydrophilic (-OH, -COOH, -NH2, -CO) functional groups. Basing on this fact and taking into account the pH value of the medium the liquid product can be considered as an acid foaming agent possessing collective properties.

### Table 1 - The composition of the liquid products of rice husk pyrolysis

<table>
<thead>
<tr>
<th>Substances</th>
<th>Kyzylorda, 400 °C</th>
<th>Almaty, 400 °C</th>
<th>Almaty, 600 °C</th>
</tr>
</thead>
<tbody>
<tr>
<td>Esters and alcohols</td>
<td>1</td>
<td>1</td>
<td>11</td>
</tr>
<tr>
<td>Ketones and aldehydes</td>
<td>12</td>
<td>9</td>
<td>6</td>
</tr>
<tr>
<td>Acids</td>
<td>35</td>
<td>45</td>
<td>51</td>
</tr>
<tr>
<td>Phenols</td>
<td>13</td>
<td>12</td>
<td>1</td>
</tr>
<tr>
<td>Heterocyclic compounds</td>
<td>5</td>
<td>3</td>
<td>4</td>
</tr>
<tr>
<td>Closed-chain aliphatic hydrocarbons</td>
<td>3</td>
<td>3</td>
<td>-</td>
</tr>
<tr>
<td>Water</td>
<td>25</td>
<td>25</td>
<td>26</td>
</tr>
<tr>
<td>Other/ unidentified</td>
<td>6</td>
<td>2</td>
<td>1</td>
</tr>
</tbody>
</table>

#### 3.2. Characteristics of lead-zinc ore from Akzhal deposit

It was found that the lead-zinc ore from Akzhal deposit was a sulfide ore formed by sphalerite (6-7%) and galena (1.0-1.2%), which are the main industrially valuable minerals. Pyrite (0.5-1.0%) was found in the content as well. Calcite (76-77%), dolomite (8-9%), quartz (2%), barite (3%), chlorite (trace) were rock-forming minerals.

According to the chemical analysis (Table 2) Akzhal lead-zinc ore consists of such valuable components as Zn - 5.20 -5.60%, Pb - 0.62-0.68%, Ag 15.0-16.0 g/t.

Granulometric analysis shows (Table 3) that Pb and Ag minerals tend to distribute in finer classes of fineness. Distribution of Zn mineral did not depend on crushed particles size and its content was almost the same in different classes of fineness. Mineralogical analysis showed that the most complete disclosure of Pb and Zn minerals occurred when the content of class -0.074 mm reached 62-65%.
cycles: pH=8.1; concentration of liquid product of rice husk pyrolysis – 15-20 g/t; flotation time – 12 minutes for lead cycle, 17 minutes for zinc cycle. Using liquid product of rice husk pyrolysis under these flotation conditions allows to increase lead recovery by 1.1% and to improve quality of lead concentrate by at least 0.5%.

Table 4 – Experimental results on the selection of optimal pH value

<table>
<thead>
<tr>
<th>Products</th>
<th>Yield, %</th>
<th>Content, %</th>
<th>Recovery, %</th>
<th>pH/K*</th>
</tr>
</thead>
<tbody>
<tr>
<td>Main Pb product</td>
<td></td>
<td></td>
<td></td>
<td>7.53/</td>
</tr>
<tr>
<td>Control Pb</td>
<td></td>
<td></td>
<td></td>
<td>1.07/</td>
</tr>
<tr>
<td>Pb flotation</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore</td>
<td>100.0</td>
<td>0.67</td>
<td>5.20</td>
<td>100.0</td>
</tr>
<tr>
<td>Main Pb product</td>
<td></td>
<td></td>
<td></td>
<td>8.7/</td>
</tr>
<tr>
<td>Control Pb</td>
<td></td>
<td></td>
<td></td>
<td>1.11</td>
</tr>
<tr>
<td>Pb flotation</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore</td>
<td>100.0</td>
<td>0.68</td>
<td>5.30</td>
<td>100.0</td>
</tr>
<tr>
<td>Main Pb product</td>
<td></td>
<td></td>
<td></td>
<td>9.0/</td>
</tr>
<tr>
<td>Control Pb</td>
<td></td>
<td></td>
<td></td>
<td>1.11</td>
</tr>
<tr>
<td>Pb flotation</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore</td>
<td>100.0</td>
<td>0.68</td>
<td>5.30</td>
<td>100.0</td>
</tr>
</tbody>
</table>

*K – Mazayoshi-Wada criterion for assessing of the flotation effectiveness, calculated by the formula:

\[ K = \frac{\beta - \alpha}{100 - \varepsilon} \]

\( \beta \) – Lead content in concentrate, %;
\( \alpha \) - Lead content in the ore, %;
\( \varepsilon \) - Recovery of lead in concentrate, %.

Acknowledgement

We acknowledge the Ministry of Education and Science of the Republic of Kazakhstan (Project No. 2252/GF4) for the financial support of the current work.

References

Table 5 - Results of comparative experiments of flotation of Akzhal Ore using the basic regime and a new flotation agent

<table>
<thead>
<tr>
<th>Products</th>
<th>Yield, %</th>
<th>Pb Content, %</th>
<th>Zn Content, %</th>
<th>Pb Recovery, %</th>
<th>Zn Recovery, %</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Basic regime</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Lead concentrate</td>
<td>0.955</td>
<td>57.72</td>
<td>13.90</td>
<td>84.81</td>
<td>2.41</td>
</tr>
<tr>
<td>Zinc concentrate</td>
<td>8.14</td>
<td>0.55</td>
<td>60.77</td>
<td>6.88</td>
<td>89.98</td>
</tr>
<tr>
<td>Dump tail</td>
<td>90.905</td>
<td>0.06</td>
<td>0.46</td>
<td>8.31</td>
<td>7.61</td>
</tr>
<tr>
<td>Ore</td>
<td>100.0</td>
<td>0.65</td>
<td>5.50</td>
<td>100.0</td>
<td>100.0</td>
</tr>
<tr>
<td><strong>Regime with liquid product of rice husk pyrolysis</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Lead concentrate</td>
<td>0.960</td>
<td>58.22</td>
<td>14.36</td>
<td>85.91</td>
<td>2.51</td>
</tr>
<tr>
<td>Zinc concentrate</td>
<td>8.10</td>
<td>0.56</td>
<td>61.0</td>
<td>7.09</td>
<td>89.88</td>
</tr>
<tr>
<td>Dump tail</td>
<td>90.94</td>
<td>0.049</td>
<td>0.45</td>
<td>7.00</td>
<td>7.61</td>
</tr>
<tr>
<td>Ore</td>
<td>100.0</td>
<td>0.65</td>
<td>5.50</td>
<td>100.0</td>
<td>100.0</td>
</tr>
</tbody>
</table>

A hybrid network and CFD model for analyzing DPM concentration as part of underground mine plan optimization

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Nick Vayenas
Laurentian University, Sudbury, Canada

Abstract—This paper presents a preliminary study that proposes the use of a hybrid solver for analyzing diesel particulate matter (DPM) in underground mines. The hybrid solver is based on a methodology that integrates two conventional techniques: a network solver and a computational fluid dynamics (CFD) solver. Because DPM is a significant issue in underground metal mines, DPM studies are a requirement to achieve better ventilation planning. Network solvers are computationally efficient but do not simulate detailed flow patterns of the contaminants such as DPM and dust. On the other hand, CFD solvers use Navier-Stokes equations and flow patterns of the contaminants can be defined more accurately; however, they are not as computationally efficient as network solvers. The hybrid ventilation modeling presents an alternative methodology.

A field study is discussed in which DPM concentrations were monitored on a diesel machine and at four locations near a dead-end heading in an underground metal mine. A network model and a CFD model were built to simulate the DPM concentrations at the monitored locations.

The study demonstrates that the DPM concentration over time in a dead-end heading can be simulated either by using the CFD or the network model. The proposed methodology for use of a hybrid ventilation model for mine plan optimization is feasible.

Keywords: underground mine ventilation, hybrid model, computational fluid dynamics (CFD), diesel particulate matter (DPM), mine plan optimization.

1. Background

Mine schedule optimization is important for increasing the life-of-mine (LOM) net present value (NPV) of mining operations, particularly during a downturn in the mining industry [1].

Because ventilation is a significant portion of the total energy cost of a mine [2], it is essential to consider ventilation constraints and apply these constraints into mine schedules.

For metal and non-metal mines, the ventilation constraints are often mainly related to diesel emissions from various diesel equipment operating underground such as a mine truck and a load-haul-dump (LHD) vehicle. Currently, there are several countries regulating the diesel particulate matter (DPM) concentration level by law. For instance, 0.06 $m^3/s$ per kW power is the standard airflow requirement for underground diesel equipment in Ontario [3]. In the United States, the DPM concentration cannot exceed 160 $\mu g/m^3$ in a shift [4].

Currently, the “0.06 $m^3/s$ per kW power” rule in Ontario is used to generate the airflow requirements for various equipment. However, this is the minimum requirement and certain underground areas may not be able to achieve the acceptable DPM concentration level. On the other hand, most mines usually supply more than the required airflow to underground working areas, which leads to additional costs for the mine. Hence, this research is carried out to better understand the DPM behaviour and to suggest applicable ventilation requirements for various working areas by using a hybrid approach.

2. Introduction

To better understand the DPM concentration distribution and to have reasonable DPM-related ventilation constraints, a hybrid methodology is proposed. The hybrid methodology combines the advantages of a network ventilation solver and a CFD solver. The network solver is quick to perform ventilation simulations but may not be accurate enough in areas like a low-airflow zone [5]. At the same time, CFD is a powerful computational tool for solving fluid-related problems and providing accurate results [6]. However, it is more computationally expensive and takes a relatively long time to obtain the results. For the day-to-day mining operations, it is infeasible to perform the simulation for the whole mine in CFD because decisions on the operation of the ventilation system need to be made shortly after receiving the requests from the underground crews. Therefore, the hybrid approach of the two solvers is presented below and some preliminary results are discussed. For this purpose, real-world DPM concentration data was collected in a gold mine in western U.S.A. Comparisons are then made among the results from the network solver, the CFD solver, and the experiment.

3. Experiment details

An on-site experiment was conducted in an underground gold mine. Five Airtec DPM monitors [7] were installed at various locations near a dead-end heading area in the mine. One monitor was mounted near the air conditioner inlet of an LHD vehicle working in the heading. The rest of the monitors were installed on the back of the drifts. The experiment area and monitor locations are shown in Figure 1.
As seen in Figure 1, the four Airtec monitors were installed at locations A, C, D, and E. No monitor was installed at location B. Measurements of airflow velocity and drift dimensions were taken at all five locations shown in Figure 1.

In this experiment, the LHD is the focus because it is assumed as the main DPM source. DPM concentration over time in an LHD’s work cycle, which consists of tramming into a heading, loading (mucking) in the working face, and tramming out of the heading, was simulated in Ansys Fluent [8] and the results were compared with those from the network solver, Ventsim [9].

4. Numerical models

4.1. CFD model

To simulate the DPM concentration over time in an LHD’s work cycle, a three-dimensional (3D) CFD model was built in Ansys Workbench 17.1. Figure 2 gives an overview of the CFD model.

The mine geometry was established in DesignModeler [10] and was exported to and meshed in Meshing. Cut-cell mesh was used [11]. Inflation layers were applied to deal with the near-wall airflow. As a result, the model has 426,862 elements; a 3D view of the mesh is shown in Figure 3. Then, the model is solved in Ansys Fluent. Boundary conditions applied in the CFD model are shown in Table 1.

The fan was assumed to operate at constant pressure in the CFD model. In Table 1, the velocity from the ventilation tubing was calculated based on the airflow quantity measurements made at locations A and B. Equation (1) was used to calculate the airflow velocity from the fan.

\[
\text{Velocity from the ventilation tubing} = \frac{\text{Difference of air quantity at A and B}}{\text{Area of the ventilation tubing}}
\]

The DPM concentrations shown in Table 1 were measured at location A. Since the fan was located in the downstream of location A, the DPM concentration was the same as that at location A. There were two main DPM sources in both the CFD model and the Ventsim model. One is the DPM originating from other diesel equipment working upstream of the drift. The other is the 18 ‘LHD tailpipes’. The diesel exhaust from each ‘tailpipe’ of the LHD engine was turned on for a certain period of time by applying 18 User Defined Functions (UDFs) (written in a C program and executed in Ansys Fluent). The UDFs define the inlet-velocity (m/s) and DPM concentration (in mass fraction) of the diesel exhaust along with time (in seconds). DPM concentrations measured in the experiment are in \(\mu g/m^3\). Because Fluent uses mass fraction in the species transport equation, Equation (2) was used to convert DPM mass fraction to \(\mu g/m^3\) is used in the equation.

\[
\text{DPM mass fraction} = \frac{\text{DPM concentration} (\mu g/m^3)}{\text{air density} (kg/m^3)} \times 10^9 \mu g
\]

The standard k-ε turbulence model was utilized to simulate turbulence in the model. A transient model was applied in Fluent because the monitors were left underground for more than 8 hours, continuously collecting DPM concentration data. The Species transport model [12] was used to simulate the
movement of the DPM particles. Because DPM is so small, it can be treated as gas and its movement is assumed to be only affected by airflow [13]. Octane (C$_{8}$H$_{18}$) was used in Ansys Fluent to represent DPM [13].

As shown in Figure 2, there are 18 ‘tailpipes’ in the fluid domain and they represent the path of an LHD. Each individual tailpipe has its own inlet and releases DPM at a certain concentration (shown in Figure 4) and a certain speed (shown in Figure 5) over time. This is intended to simulate the movement of the LHD in the area of the heading. The LHD enters the area from the outlet and trams into the heading to muck. After completing mucking at the working face, the LHD trams out of the area and exits the area through the outlet.

Figure 2. There are 18 ‘tailpipes’ in the fluid domain and they represent the path of an LHD. Each individual tailpipe has its own inlet and releases DPM at a certain concentration and speed over time.

In Figure 4, the DPM concentration profile for the 18 pipes is presented. The DPM concentration data are obtained through trial and error to represent the DPM concentration levels for tramming in, mucking, and tramming out in the LHD’s work cycle, in this CFD model. The boundary conditions for the inlets in the CFD model mainly include the velocity and DPM mass fraction. Since the velocity and the DPM concentration of the diesel exhaust plume from the tailpipe of the LHD engine were not measured, the data from Zheng [13] was used as a reference in the CFD model. After running the CFD model with various sets of velocity and DPM mass fraction data, the best DPM concentration and velocity data are shown in Figure 4 and Figure 5, respectively.

According to the DPM data collected in the Airttec monitors, the time for tramming in, mucking, and tramming out for the LHD’s work cycle is inferred. The profile is used to imitate an LHD’s work cycle, which consists of tramming into a dead-end heading, mucking in the heading, and tramming out of the heading. Based on Figure 4, it takes the LHD 42 s to tram into the working face from the outlet (shown in Figure 2). Then the LHD stays in the working face and mucks for 68 s. After mucking, with the bucket loaded, the LHD trams out of the heading and dumps the muck in an orepass near the outlet. As seen in Figure 4, it takes the LHD 190 s to tram out.

In a like manner, the velocity profile of the diesel exhaust over time from the tailpipes is shown in Figure 5.

Figure 3. Velocity profile of the diesel exhaust from the tailpipes in the CFD model.

In a work cycle, the DPM concentration from an LHD’s tailpipe is the lowest when it is tramming into a heading with the bucket empty. The DPM concentration is the highest while the LHD is mucking in the heading. The DPM concentration is between the two concentrations mentioned above while the LHD is tramming out of the heading with the bucket loaded with ore (or waste).

It takes an LHD much longer to move out of a heading than to move into the heading because of the additional weight from the transported ore or waste.

The DPM concentration levels are stable in each part of an LHD’s work cycle.

The 18 pipes in the CFD model are not moving. In reality, they are on the LHD and close to floor. To reduce the computational effort and maintain the accuracy of the results from the CFD model, the pipes are located on the floor of the drift. Each pipe is represented by a 0.3 m × 0.3 m × 0.3 m cube in the CFD model.

The DPM concentration values are averaged (with the “moving average” function in Microsoft Excel) to match the DPM concentration measured by the Airttec DPM monitors.

5. Ventsim model

The network model was built in Ventsim [9], a well-known commercial 3D mine ventilation software. It is able to conduct both static and dynamic simulations of airflow, heat, pressure,
DPM, gas, and fire [14]. In this case, the DPM dynamic simulation is used to simulate the DPM concentration changes in a work cycle of an LHD.

As shown in Figure 6, similar to the CFD model, there are 18 diesel sources in the Ventsim ventilation model. The drift sizes are the same as in the CFD model. Through a ventilation tubing and an auxiliary fan (represented by the fixed flow), fresh air is directed to the working face. Several DPM monitors are placed in the ventilation model to record the DPM concentration over time. The DPM concentration data are recorded every second and the total simulation time is 300 s.

In Ventsim, the boundary conditions applied for the inlets were equivalent to that used in the CFD model although the values were not the same. Ventsim uses the product of a diesel power (kW) and a DPM rate (gram/(kW·hour)) of a diesel engine to determine the DPM concentration [9]. The DPM sources (shown in Figure 6) in the Ventsim model occupied the whole cross-sectional area while the DPM sources (shown in Figure 2) were much smaller and located on the floor in the CFD model. Then the DPM concentration data used in the Ventsim model was converted from that used in the CFD model. As a result, the boundary conditions for the inlets were equivalent in the Ventsim model and the CFD model.

Because the field data were collected every minute, the recorded DPM data in both the CFD and network models are averaged by using the “moving average” function in Microsoft Excel.

6. Results

In the CFD model, a monitor point and a monitor plane have been created to record the DPM concentration data over time at locations C, D, and E. The monitor points are created because the DPM monitors in the experiment were installed on the back of the drifts. An example of the location of a monitor point is shown in Figure 7. As shown in Figure 7, the monitor plane is the cross-sectional area where the monitor point is located.

Since the CFD model only simulated one of the LHD work cycles, the DPM concentration residuals from the previous LHD work cycle should be added to the results from the CFD model. For instance, the work cycle chosen from the experiment (at location D) is highlighted inside the green rectangle shown in Figure 8. It is obvious that the DPM concentration does not start at zero, and this DPM concentration is defined as the baseline DPM concentration in this paper. The baseline DPM concentration was added to the CFD results for comparison with the experiment results.

6.1. CFD results

Results from the three monitor points C, D, and E in the CFD model are compared with those from the experiment in Figure 9, Figure 10, and Figure 11, respectively. The vertical axis represents the DPM concentration and the horizontal axis represents the duration of an LHD’s work cycle.

For the network model, three monitor planes are placed at locations C, D, and E. The work cycle chosen as the experiment data is the cycle inside the green rectangle.
the Airtec monitors at every minute. To accurately compare the results, the DPM data in the CFD model are averaged to one-minute data using the “moving average” built-in function in Microsoft Excel.

As seen in Figure 9, the trend of the DPM concentration over time from the CFD model at location C has a good agreement with that from the experiment. There are differences in the DPM concentrations from the start to 160 s. One possible reason for the low DPM concentration in the experiment is that the shape and size of the LHD changed the airflow distribution and affects the DPM concentration.

The comparison at location D is shown in Figure 10. The DPM concentration over time at location D in the CFD model does match that in the experiment. From 0 to 120 s, the DPM concentration in both the CFD model and the experiment increases. After 120 s, the average DPM concentration in both the CFD model and the experiment decreases.

The comparison at location E is shown in Figure 11. It can be clearly seen from Figure 11 that the trend of the DPM concentration over time in the CFD model follows a similar trend as that of the DPM monitors in the experiment, although the magnitude of the DPM concentration in the CFD model is lower than that in the experiment and the timing of the peak value is not aligned. As shown in Figure 1, location E is between the end of the ventilation tubing and the working face. There were airflows coming from both sides of location E and it made the fluid domain near location E complex. The complex airflow distribution near the monitor point may have affected the Airtec DPM monitor data because it measures the weight of elemental carbon accumulation over time [7].

6.2. Network model results

DPM concentrations over time at the three locations are simulated in the network model and the results are presented in Figure 12 to Figure 14. As mentioned above, these results cannot be directly compared with those from the experiment because they are area-weighted average DPM data. Therefore, the following comparisons are made between the CFD and the network models. The DPM concentration data in the CFD model is recorded through the monitor planes at the locations C, D, and E.

Figure 12 shows the DPM concentration comparison at location C between the CFD and the network models. The DPM concentrations from the Ventsim model are higher than those in the CFD model. However, the trends of the DPM concentration over time in the two models are similar. At 0 and 240 s, the DPM concentrations in the CFD model are lower than that in the network model. This may be caused by the fact that the ventilation tubing is extended to the working face in the network model, which changes both the airflow and DPM...
concentration distributions. The DPM concentration comparison at location D is shown in Figure 13.

As seen in Figure 13, the trend of DPM concentration over time at location D in the network model matches that in the CFD model. On average, the magnitude of the DPM concentrations in the network model is higher than that in the CFD model.

According to Figure 14, on average, the DPM concentrations obtained from the CFD model are lower than that from the network model. The trend of the DPM concentration over time is similar between the two models.

According to Figure 12, Figure 13, and Figure 14, the DPM concentration data obtained from the network model is not in a good agreement with that from the CFD model. There are differences between the CFD model results and the network model results although the boundary conditions are the same for the two models. These differences, especially at location C, the outlet in both models, can be resolved by using the calibrated CFD results to, in turn, calibrate the network model results.

6.3. Error analysis
According to the results shown above, an error analysis is performed to measure the accuracy of the CFD model in the work cycle of the LHD. A regression analysis is performed to calculate the coefficient of determination, which is R-squared ($R^2$) [15]. In this case, the R-squared means the proportion of variations described by the CFD model to the total variations in the experiment. The higher the R-squared value, the better correlation between the CFD model and the experiment. It is considered as a significant relationship if the R-squared value is greater than 0.5 [15]. Table 2 represents the R-squared computed according to the results from the CFD model and the experiment. As discussed, the LHD cycle time was estimated to be four minutes (240 s) as shown in Figure 9, Figure 10, and Figure 11. The Airtec DPM monitor was set to collect DPM data every 1.0 minutes. Therefore, there were five DPM concentration data points collected in the experiment. The R-squared (shown in Table 2) are calculated based on the five DPM concentration data collected at the locations C, D, and E, respectively.

<table>
<thead>
<tr>
<th>Location</th>
<th>R-squared</th>
</tr>
</thead>
<tbody>
<tr>
<td>C</td>
<td>0.67</td>
</tr>
<tr>
<td>D</td>
<td>0.94</td>
</tr>
<tr>
<td>E</td>
<td>0.50</td>
</tr>
</tbody>
</table>

According to Table 2, the R-squared is greater than 0.5 at location C, and location D; it is 0.5 at location E. It means that the CFD model has a good correlation with the experiment at locations C, D, and E.

7. Conclusions and future work
For dead-end headings, the results from the CFD model are more readily assessed against experimental data than those from the network model. The approach of using multiple tailpipes to simulate the DPM concentration generated from a work cycle of an LHD is feasible. Due to its simplicity, this approach can be used to support optimization of airflow usage in a dead-end heading.

In the network model, a closed-loop is required to be formed if the airway is not connected to the working face. This leads to the ventilation tubing’s extension all the way to the working face, which does not represent the reality. The accuracy of the network model has not been assessed through a direct result comparison with the experimental data.

In conclusion, the CFD model is capable of offering detailed contours of airflow and DPM concentration, but it is computationally demanding and it takes a longer time to generate results. The network model is relatively easy to set up and quicker to derive the results for both airflow and DPM but has some limitations in a dead-end heading. Therefore, a combination of the CFD and network models is indicated for a better understanding of both airflow and DPM distribution in the mine. These preliminary results of ongoing research provide a solid foundation for the planned future work.

The planned hybrid model will use the network model for the main development drifts and the CFD model for the dead-end headings connected with the drifts. The DPM and airflow results from the CFD model will be used as the new inputs for the network model and overwrite the existing results in the network model. Then, an overall ventilation model for an underground mine can be anticipated to have a higher accuracy and to require less computational effort.

Acknowledgements
The authors would like to thank Newmont Mining and the Ultra-Deep Mining Network (UDMN) at the Centre for Excellence in Mining Innovation (CEMI) for their technical and financial contributions to this work. This research was
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References
MPES2017

Session 7 – Optimization in Mines

13:30-15:00

August 30, 2017
Short and Medium Term Mine Planning in Selective Underground Mining considering Equipment Performance

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Abstract— Generally, mine planning is undertaken towards the development of production plans using fixed parameters, which provide little flexibility towards changing production plans in case of unplanned events. Therefore, it is important to introduce variables during the planning process, which would allow the mine planning to have a better alignment with real mining conditions and would allow the decrease in the operational uncertainty towards the development of a more agile production plan.

The mine layout grows every day but it is very common that the equipment productivity is calculated with fixed parameters, which does not consider the size of the new developments and, thus, it does not consider the changing performance of the equipment, which will affect the mine production because of the associated changes in haulage time, variability in the equipment’s tasks and other influences resulting from the changing mine layout.

This paper evaluates influence of the changes in the equipment performances over a short- and medium-term mine plans on the mine production. The results show that the change in scheduling and sequencing of the activities and the new mine plan are more realistic when equipment performance is included in the evaluation in comparison with a scheduling that only considers resources constraints. The methodology was developed using tools currently available at the DELPHOS Mine Planning Laboratory, UDESS and DSIM, and by simulating the working scenario with varying performance of the equipment.

Keywords: mine planning, selective mining, equipment selection, UDESS, DSIM.

1. Introduction
The main objective of mine planning is the optimization of economic value for the different stakeholders. It is, therefore, natural for mine planners to model the production and the economic value of a project in terms of different parameters or decisions, which are then optimized to obtain the best possible economic value. The range of techniques that can be used is extensive and ranges from the manual evaluation of few scenarios to the utilization of advanced computational techniques, for example, mixed integer programming, to model the production of mine operations to arrive at the best-value plans.

The planning process requires various data, such as operational data that includes data related to the performance of the equipment. The equipment performance indicators are obtained from nominal equipment productivity parameters, which are adjusted using operational multipliers such as mechanical availability, operational losses, and others.

The planning process based on the static approach does not account for the variability of various mining tasks, the evolution of the layout over time, interactions between various pieces of equipment is complex to estimate. Indeed, the actual values of these parameters depend on the long-term plan; for example, the transportation capacity of a mine depends on the relative transportation distances and, therefore, is not a constant parameter over the life of the mine.

Therefore, the drilled meters and productivity (KPI’s) used in the long-term change over time (Figure 1) and the indexed values evolve depending on the mine size, due to travel times and events occurring during the life of the mine.

Therefore, planning process should follow an iterative approach (Figure 2) commencing from an initial plan that can be obtained from an optimization process and using simulations to estimate equipment productivity. The simulation results, new parameters, are then used as an input into the optimization process to update the plan accordingly [8].

![Figure 1. KPI’s function of time](image-url)
Unfortunately, implementing this methodology can be very difficult because optimization solvers, such as, Gurobi®, CPLEX®, and others, as well as simulation softwares (Arena®, ProModel®, etc.) are specialized to perform specific tasks and possess only limited capabilities to interact with other software in an efficient way. These limitations led to the development of the optimization and simulations models for mine planning that allow the integration between solvers and/or simulation softwares by means of scripting. The models (which are also available as software tools) are called UDESS and DSIM and are for optimization and simulation, respectively.

1.1. Optimization Model

The optimization model used is a general scheduling model that takes as inputs: (a) activities (or tasks), their lengths and operational resource requirements, (b) the logical precedences for these activities, and (c) the net profit of performing such activities. The model computes the schedule of activities that complies with precedence and resource availability in such a manner as to maximize the economic value (or minimize cost). This model has been successfully tested in scheduling of production and preparation for panel caving in the deterministic scenarios [13], under operational uncertainty [10], and in scheduling of projects under price uncertainty [9]. It has also been used in an interactive way, as shown in Figure 2, but using other models; for example, material flow in a caving mine [4], seismic risk [3], and dilution in a cut and fill [11].

1.2. Simulation Model

Contrary to commercial solutions, the simulation model is specifically oriented to material handling in open pit mines as well as production and preparation in underground mining. It implements: (a) a set of functions that allow to easily define a layout and modelling of movement of equipment, (b) several agents (trucks, shovels, LHDs, etc.) that can be used as is or extended to model more complex situations, and (c) reports specially tailored to mine operations (cycle times, production).

To-date it has been mainly used in open pit mines, for example, to study the variability of production due to operational and geometallurgical uncertainty [5] or to simulate autonomous hauling systems [6].

2. Methodology

The methodology used in this research is summarized in the following steps:

1. Layout creation, which considers both preparation and extraction activities simultaneously.
2. Development of a production plan using an optimization model (UDESS), considering KPI inherent to the production equipment that is in operation.
3. Simulation of the production plan and material movement at certain time intervals using a simulator (DSIM).
4. Elaboration of a new production plan using the results of the simulation and the new KPI’s as inputs.
5. Analysis of the changes in the production plan, considering the new information obtained
6. Iteration from Step 3.

The mine layout used in the study (Figure 3) corresponds to a mine extracted by a bench-and-fill method, which considers two productive sectors, East and West, each with 5 levels, which must be prepared before commencing the operation. It should be noted that only drifts require to be developed in the simulation and that all the infrastructure required separately (mine entrance and inter-levels ramps, access drifts, ventilation, ore passes) are ready at the beginning of the simulation. Material handling from the ore passes out of the mine is not part of the research.
Non-Available Time: interval in which the equipment is in the workshop, due to scheduled or unscheduled maintenance.

Effective Time: time in which the equipment are performing an assigned task or the time of travel to their destination.

In development task, the activities carried out are drilling, explosive loading, blasting and ventilation, muck removal, hang-up removal, shotcreting and roof support.

In operation task, the activities carried out are drilling, loading of explosives, blasting and ventilation, ore extraction and stoping filling. The filling is carried out after three exploitation processes.

The list of equipment used is shown in Table 1.

<table>
<thead>
<tr>
<th>Type</th>
<th>N°</th>
<th>Type</th>
<th>N°</th>
</tr>
</thead>
<tbody>
<tr>
<td>LHD</td>
<td>2</td>
<td>Jumbo</td>
<td>2</td>
</tr>
<tr>
<td>Scaler</td>
<td>1</td>
<td>Simba</td>
<td>2</td>
</tr>
<tr>
<td>Explosives</td>
<td>1</td>
<td>Boltec</td>
<td>1</td>
</tr>
<tr>
<td>Shotcrete</td>
<td>1</td>
<td>Backfill</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Truck</td>
<td>3</td>
</tr>
</tbody>
</table>

Since each of the simulations has variability, each of the possible scenarios can either fulfill the plan or not and, therefore, the simulations must be compared with the result of the optimization process.

There are two different models, mathematical and simulated, that must interact; indexes are proposed that compare the operability of the proposed development plan because the mathematical model only considers precedences and resources that must be used but does not consider the operation of the system. The simulation provides several scenarios in which the production plan generated might be replicated, checking whether the production plan is fulfilled and whether each task individually, both development and extraction, may or may not be performed.

Assuming that an activity is the complete process referred to the development of a complete gallery or the extraction of all the stopes in an extraction gallery and an iteration is the generation of a plan between the process of optimization and simulation (Figure 2), the following parameters are defined:

- \( i \): Activity. \( i \in 1,...,N \)
- \( j \): Replica. \( j \in 1,...,R \)
- \( h \): Iteration. \( h \in 1,...,L \)
- \( B_{ih} \): period in which \( i \) activity begins in the UDESS plan in \( h \) iteration.
- \( F_{ih} \): period in which \( i \) activity finishes in the UDESS plan in \( h \) iteration.
- \( SB_{ijh} \): period in which \( j \) activity begins in the \( j \) DSIM replica in \( h \) iteration.
- \( SF_{ijh} \): period in which \( j \) activity finishes in the \( j \) DSIM replica in \( h \) iteration.

Therefore, for

\[
y_{ijh} = \begin{cases} 1 & \text{if } SB_{ijh} \leq B_{ih} \\ \text{otherwise} & \end{cases}
\]

\[
z_{ijh} = \begin{cases} 1 & \text{if } SF_{ijh} \leq F_{ih} \\ \text{otherwise} & \end{cases}
\]

the following relationships can be written:

\[
PB_h[\%] = \frac{\sum_{i=1}^{N} \sum_{j=1}^{R} y_{ijh}}{N \times R}
\]

\[
PF_h[\%] = \frac{\sum_{i=1}^{N} \sum_{j=1}^{R} z_{ijh}}{N \times R}
\]

For the present research, the amount of development and extraction activities is defined as \( N = 95 \) and \( 35 \), respectively, while the number of replicas is \( R = 100 \) and the dilution is defined as:

\[
\text{Dilution}[\%] = \frac{\text{Waste}}{\text{Ore} + \text{Waste}}
\]

Dilution is considered as the amount of waste that enters to the material extraction. In the present case, using Equation 5, it is considered that the waste corresponds to 20% of the ore with dilution equal to 16.6%.

Optimization model is considered for the short-medium term of one year with periods of 1 month, while simulation model has a length of four months, where the values observed in the fourth month are used for the rest of the year.

### 3. Results and Discussion

As the iterative process was undertaken, it was observed that the productivity obtained did not change between the iterations in both cases, without and with dilution in Figure 4 and 5, respectively, resulting in the equipment hauling the same quantity of material, although there was an increase in the ore extraction each month. When considering dilution, daily production increased slightly.

![Figure 4 – Average Productivity in DSIM Plan without dilution](image_url)
The drilled meters using horizontal and radial drilling equipment were compared showing small variation between the iterations (Tables 2 and 3).

The amount of dilution does not depend on the drilled meters and thus it is expected that this variable remains the same in both cases, without and with dilution (Table 2 and 3).

### Table 2 - Horizontal Drilling Performance

<table>
<thead>
<tr>
<th>Iteration</th>
<th>Month 1</th>
<th>Month 2</th>
<th>Month 3</th>
<th>Month 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Without Dilution</td>
<td>76,906</td>
<td>60,324</td>
<td>60,520</td>
<td>57,192</td>
</tr>
<tr>
<td>Iteration 1</td>
<td>76,810</td>
<td>60,555</td>
<td>60,323</td>
<td>57,254</td>
</tr>
<tr>
<td>Iteration 2</td>
<td>76,919</td>
<td>60,323</td>
<td>60,466</td>
<td>56,911</td>
</tr>
<tr>
<td>Iteration 1</td>
<td>76,821</td>
<td>60,830</td>
<td>59,776</td>
<td>56,845</td>
</tr>
<tr>
<td>Iteration 2</td>
<td>76,683</td>
<td>60,758</td>
<td>59,910</td>
<td>57,022</td>
</tr>
<tr>
<td>Iteration 3</td>
<td>76,623</td>
<td>60,402</td>
<td>60,146</td>
<td>56,923</td>
</tr>
</tbody>
</table>

### Table 3 - Radial Drilling Performance

<table>
<thead>
<tr>
<th>Iteration</th>
<th>Month 1</th>
<th>Month 2</th>
<th>Month 3</th>
<th>Month 4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Without Dilution</td>
<td>366</td>
<td>11,013</td>
<td>14,858</td>
<td>14,918</td>
</tr>
<tr>
<td>Iteration 1</td>
<td>492</td>
<td>10,478</td>
<td>14,701</td>
<td>14,919</td>
</tr>
<tr>
<td>Iteration 2</td>
<td>552</td>
<td>10,553</td>
<td>14,779</td>
<td>14,784</td>
</tr>
<tr>
<td>Iteration 1</td>
<td>443</td>
<td>11,003</td>
<td>14,565</td>
<td>14,714</td>
</tr>
<tr>
<td>Iteration 2</td>
<td>640</td>
<td>10,492</td>
<td>14,703</td>
<td>14,874</td>
</tr>
<tr>
<td>Iteration 3</td>
<td>687</td>
<td>10,276</td>
<td>14,459</td>
<td>14,695</td>
</tr>
</tbody>
</table>

Table 4 indicates the resource constraints used in the optimization model, considering the average of horizontal and radial meters perforated according to the simulation model. The first period constraints are also obtained by using simulations, taking the maximum drilling value obtained, using an extraction schedule from the bottom up.

### Table 4 - Constraints in optimization model (UDESS)

<table>
<thead>
<tr>
<th>Iteration</th>
<th>Drilling</th>
<th>Period</th>
<th>Upper Limit [m/month] Without Dil.</th>
<th>Upper Limit [m/month] Dilution</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Horizontal</td>
<td>1-12</td>
<td>76,500</td>
<td>76,830</td>
</tr>
<tr>
<td></td>
<td>Radial</td>
<td>1-12</td>
<td>15,000</td>
<td>14,660</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1</td>
<td>76,900</td>
<td>76,820</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2</td>
<td>60,325</td>
<td>60,830</td>
</tr>
<tr>
<td></td>
<td>Horizontal</td>
<td>3</td>
<td>60,520</td>
<td>59,780</td>
</tr>
<tr>
<td></td>
<td></td>
<td>4-12</td>
<td>57,200</td>
<td>56,850</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>1</td>
<td>370</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>2</td>
<td>11,000</td>
</tr>
<tr>
<td></td>
<td>Radial</td>
<td></td>
<td>3</td>
<td>14,900</td>
</tr>
<tr>
<td></td>
<td></td>
<td>4-12</td>
<td>14,900</td>
<td>14,715</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>1</td>
<td>76,800</td>
</tr>
<tr>
<td></td>
<td>Horizontal</td>
<td></td>
<td>2</td>
<td>60,550</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>3</td>
<td>60,320</td>
</tr>
<tr>
<td></td>
<td></td>
<td>4-12</td>
<td>57,250</td>
<td>57,025</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>1</td>
<td>490</td>
</tr>
<tr>
<td></td>
<td>Radial</td>
<td></td>
<td>2</td>
<td>10,480</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>3</td>
<td>14,700</td>
</tr>
<tr>
<td></td>
<td></td>
<td>4-12</td>
<td>14,920</td>
<td>14,875</td>
</tr>
</tbody>
</table>

Figures 6 and 7 show the mine plan obtained by the optimization model without and with dilution, respectively. In the first iteration, the result only considers the resources constraints and the preceding activities, with the operational parameters not considered as part of the input data. In both cases, the amount of material extracted is much higher for the first 3 months and then declines in the last periods, decreasing 30% and extracting irregular material amounts.

The case without dilution (Figure 6) shows that certain periods do not have material extraction; these periods correspond to the filling of the stopes only.

Figure 6 – UDESS Plan without dilution

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Figure 5 – Average productivity in DSIM Plan with dilution

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As shown in Table 5, the variation of the NPV is very small, decreasing slightly with each iteration. However, the plan delivered by the optimization model shows that the amount of material extracted on a monthly basis differs; it commences with no material extracted with a gradual decrease of extraction towards the end periods.

When considering these adherence indexes the most apparent changes are shown in Table 6, indicating that the new plans tend to improve adherence to the assigned plan as compared to an original plan that does not necessarily consider the mine operation. The change in the index does not necessarily increase when performing more than one iteration, but if it considers that the new plans consider a major operative sequence.

4. Conclusions

The optimization model considers precedence and resource constraints in the activities performed and, from the point of view of NPV maximization, the optimum result is achieved, but it does not consider the system's operation. On the other hand, the application of the methodology presented in this paper allows to feed back the new information gathered from the simulations and to achieve an operative and optimal development and production plans.

It was observed that the changes in both the optimization and simulation models show remarkable improvements between the first and second iterations in the cases analyzed, achieving an improvement in the adherence index parameters of almost 35%, although in none of the cases this compliance rate exceeded 92%, which means that on average the plan is considered as overestimated in terms of available resources.

The production plans generated a remarkable change between first and second iteration made, but the result remains practically constant between the second and third iteration. However, for the second and third iteration, the scheduling of activities changed in some periods, which might have influenced the outcomes of the simulation. Therefore, it is recommended to undertake further studies on the influence of the variable scheduling on the stabilization and productivity indexes.

Acknowledgement

We acknowledge the financial support of the Advanced Mining Technology Center and the CONICYT Basal Project FB0809, which allowed for this work to be undertaken.

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Table 5- Optimization model NPV

<table>
<thead>
<tr>
<th>Iteration</th>
<th>Without Dil.</th>
<th>Dilution</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>761.89</td>
<td>756.4</td>
</tr>
<tr>
<td>2</td>
<td>760.26</td>
<td>751.94</td>
</tr>
<tr>
<td>3</td>
<td>760.14</td>
<td>752.83</td>
</tr>
</tbody>
</table>

Table 6- Adherence Indexes in Development and Extraction Plan

<table>
<thead>
<tr>
<th>Iteration</th>
<th>Without Dil.</th>
<th>Dilution</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Figure 7- UDESS Plan with Dilution

[12] Promodel. (n.d.) in Simulation Software, Available at: http://ampl.com/products/solvers/solvers-we-sell/plex/?gclid=Cj0KESjwioHIBRCSesnP56Ti1lsBEiQAxh5GwbMy5ofsPgBTFDLOXbQP-QqNRwLumKZjkgxr9F4aAkL8P8HAQ
Rationale of retaining wall design of reloading points during usage of combined types of transport

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Abstract—The analysis of the main technical solutions for construction of retaining walls of the loading points represented by their structural features and application prospects. The new design of ore roll for rock mass and ore transfer up to the surface level due to weight is offered. A sequence scheme of mining a deep upland mineral occurrence to reduce the stripping ratio by using mining order is offered.

Keywords: an open pit mine, non-operating pit bank, gabions, ore roll, mining scheme.

1. Introduction

Operation of the combined automobile and conveyor transport on the operating iron ore pits is conducted with installation of KKD-1500/180 type crushers on the stationary points of an overload located at a depth of 150-250 m from a surface. Costs of their arrangement are rather big. In this regard, with constantly growing development depth for the purpose of decreasing transportation distance of rock mass by dump trucks, it is expedient to arrange temporary loading points of ordinary rocky mass from automobile to specialized railway or conveyor transport with a step of 60-90 m.

2. Material and Methods

Increase in a slope angle of a non-operating pit bank during construction of a reloading point plays the main role in reduction of overburden volume. The slope is fixed by retaining walls of various design for stability purposes. The analysis of construction of retaining walls on foreign open pits is made. Consequently, the possibility of construction of 30 m high retaining walls is proved during construction of Mikheevsky mining and processing plant (Russian Federation) while constructing entrance platforms of heavy-load dump trucks in a loading zone of the crusher. The classical decision using reinforced concrete technologies was due to high labor costs, the considerable construction height and also heavy loads from dump trucks (to 330 tons) were reasons for application of classical decision.

As the alternative decision, the LLC “Makkaferri gabions ICU” offered to construct retaining walls using the armogrunt system Terramesh strengthened by geo-grids Paragrid 200 and Paralink 500 [1]. The design of a retaining wall represents the armogrunt embankment consisting of 3 10 m high blocks divided by 2 intermediate 2 m wide berms. Concrete blocks will be placed in the base of the lower block, and on an adjunction to the rock in cross-cut ends. Use of reinforced soil system has allowed using as much local construction materials as possible and reducing to minimum volumes of expensive constructions from monolithic reinforced concrete.

Box-shaped gabions are placed in the basis of a retaining wall, which were also used in landscape works. At the same time, they function as the protective apron preventing washout of the construction base. Mattress type gabions are rectangular structures with a height in the range of 17-50 cm. The name “mattress” was given due to the small relation of height to length and width. For durability purposes, long mattresses are also divided by cross diaphragms from the inside to ensure rigidity of a mesh design. They are filled with stones, forming a monolithic design. Mattresses are used as the base of retaining walls from box-shaped gabions, protect the construction base from a washout, protect and stabilize the soil from an erosion. Cylindrical gabions made from a metal grid filled by a natural stone are also used. Cylindrical gabions are used as the underwater bases during construction of retaining walls near water reservoirs. Cylindrical gabions can also be used for decorative design of the site of a pit bank edge during reclamiation of open pit sides on mined open pits. Company are developing and using technologies of constructing a retaining wall from the soil reinforced by synthetic materials. Panels of geotextiles (Fig. 1) are used for external coating and reinforcing a wall.
Retaining walls built from the soil reinforced by geo-grids in combination with geotextiles were successfully implemented in practice. Such walls are well adapted for uneven precipitation, and they compensate for temperature and shrinkable tensions. A geo-grid is the reinforcing geotechnical material. It is a set of sheet strips, 1.35 mm to 1.8 mm thick and 50 to 200 mm high. Sheet strips are interconnected by seams, forming cells (geo-grids). Depth and the sizes of cells are selected depending on loading and structure criteria of filler materials. The expanded geo-grid forms a cellular structure which is filled with mineral material. Sections of geo-grids have high physics-mechanical characteristics and maintain temperature conditions of all climatic zones. Sections of geo-grids are made from strong and flexible polyethylene tapes that allow building retaining walls of various configuration. The steepness of the reinforced slope is not limited and can be vertical.

Loading and reloading of a rock mass in gabion body could be organized by a mechanized method. The construction machine can be placed both from the face of a retaining wall and from backfilling material (the figure 2, a). during the mechanized filling of gabions. However, it is worth noticing that driving the construction equipment on gabion structures, without the use of special measures, is not allowed.

Figure 1. The scheme of placing cylindrical gabions on a loading point. 1 – gabion structure; 2 – filler between a wall and the mineral massif; 3 – base of a retaining wall; 4 – a platform of cylindrical gabions

Retaining walls are manufactured structures, constructed to prevent collapse of massif located behind them. Retaining walls are categorized according to usage, height, construction materials, operation principle and construction method.

Retaining walls are subjected to different loads, among which are weight of a wall (q), pressure from massif (PM), pressure from weight of constructions, located on a supported massif (PK), pressure of mining-transport
equipment (PO), also pressure coming from surrounding water (PB) (Fig. 3).

It should be noted, that deep mines are operated in conditions of significant inflow of surrounding water. Inflow of water in conditions of SMEP mine is equal to 871 m$^3$/hour, and for IMEP mine – 1096 m$^3$/hour. Drainage system is constructed to prevent negative impact from water inflow. However, this measure is accompanied with additional costs.

New designs of ore roll and loading bunkers are developed within the scope of grant financing of the Ministry of Education and Science of the Republic of Kazakhstan, which could be used in constructive design of combined types of transport (Fig. 4) [3].

### Results and Discussion

It is expedient to calculate economic efficiency of the developed method by the minimum operational costs of waste transportation and transportation of ore to the daylight surface. In this case efficiency of new technology $E$, mln. USD is determined by a formula:

$$E = 10^{-6} \cdot C_a \cdot (V_{z_l} - V_{v.p_l})$$  \hspace{1cm} (1)

where, $C_a$ – transportation costs of 1 m$^3$ of rock mass to a distance of 1 km, USD; $V_{z_l}$, $l_{z_l}$ - respectively a volume and distance of transportation of rock mass by dump trucks to the surface level, one million m$^3$ and km; $V_{v.p_l}$, $l_{v.p_l}$ – respectively a volume and distance of transportation of rock mass to the surface level due to weight influence, one million m$^3$ and km.

It is offered to carry out development of the 2 km long open pit field according to the developed method. Currently, the upland part of the field is being operated and the open pit start to function. Overburden volume of $V_z = V_{v.p} = 153.3$ million m$^3$ is transported for warehousing to distance $l_z = 5$ km. The same volumes of waste will be transported on a mountain downslope by the influence of the weight at the corresponding average transportation distance of $l_{v.p} = 3$ km.

Cost price of 1 tkm operation of the motor transport and loaders is taken to be identical and has a value of $C_a = 0.925$ USD.

Economic efficiency of implementing this method with minimum transportation distances will be following:

$$E = 10^{-6} \cdot 0.925 \cdot 153.3 \cdot (5 - 3) = 28.6 \text{ mln. USD}$$

According to the provided data, annual economy (million USD/year) due to introduction of new technology of ER with the operation term of 55 years can be:

$$E_p = \frac{E}{55} = \frac{28.6}{55} = 0.516$$  \hspace{1cm} (2)

Overall effectiveness of the developed method will be proved in the course of implementation of the mining project documentation and significantly increased due to reduction of capital investments on construction and operation of ore and mineral rolls.
Stripping ratio could be decreased if areas with different thickness of seams are mined with specific order (Fig. 5).

It is offered to mine the open pit field is mined by order while mining extended field with bigger length compared to width [2].

Mining of 8km long open pit field should be divided into 3 phases, the first of which is 4 km long and must be mined with construction of the 2 km long pioneering open pit mine, with width of 1 km and depth up to 90 m. Overburden with volume of 7.5 million m$^3$ is stripped by an excavator and stored on an external dump near border of an open pit, then the open pit of the first order is put into operation, and its developed space serves further as an object for development of an internal dump. Currently, the mountain part of the field is put into operation and the open pit of the second order starts to function. Its overburden is also transported for storing on the mined space of an open pit of the first order. A third order open pit would be operated after development of infrastructure of the enterprise.

Overburden volumes for storing on an internal dump by mining phases are equal to 41, 153.3 and 184 million m$^3$ at the corresponding average transportation distance of 1, 3 and 5 km according to a general regime of stripping operations. Cost price of Prime cost of 1 tkm of the motor transport is equal to 0.925 USD.

4. Conclusions

The analysis of efficiency of using a retaining wall from gabions showed that they are 10 times cheaper and 9 times less labor-consuming, than a wall from the reinforced soil, and one third times cheaper than traditional concrete retaining walls. It is possible to reduce costs of injection works by using small fractions of crushed waste in solutions, not only in the form of filling agents, but also as active mineral additives.

Layer-by-layer mining of rock mass by steeply inclined layers decreases the overburden transportation distance by 2 km. Annual economy due to this reaches 28.6 mln. USD.

Economic efficiency of implementing offered method would be 425.87 mln. USD. Annual economy due to introduction of new technology of ER with operation term of 55 years can be 7.74 mln. USD.

References


Abstract—Availability of mining machines plays a significant role in mine production. Dragline’s reliability has a great impact on sustaining economic feasibility of open cast coal mining projects. In that sense, reliability of draglines and optimizing its preventive maintenance are key issues to be addressed. The objective of this study is to apply machine learning methodologies for classifying failure types of a dragline based on real data. The mean time between failure data was acquired from an operating open cast coal mine in Turkey. Three modified forms of K-Nearest Neighbors algorithms have been used as predictors for failure classification. An approximation function has been generated based on the time to failure and breakdown type. In case of parameter tuning, cross validation method has been utilized. This caused more reliable evaluation of the test sample, so average testing performance has been used for test data estimation. The basic model was for parameter tuning; moreover, for achieving more efficient parameter Grid Search method was utilized. Since, usage of the algorithm is computationally expensive, so Randomized Search method has been carried out in order to figure out the functionality of modeled function in the high dimension dataset. The results of the study revealed that the application of K-Nearest Neighbors method reached the Regression Analysis of 73 percent. Thus, the higher accuracy of prediction of failure type can be helpful in prognostic of dragline’s procedure. The main novelty of this study is the utilization of K-nearest neighbor algorithm and existing health status can be estimated. Prognostic is defined [1] as estimation in regards to not only time-to-failure but also risk of existing failure modes. The maintenance team can act and avoid the catastrophic failures by means of RUL; conversely, an appropriate design of prognostic system is tough action which needs superior consideration. Prognostic can be done by three methodologies: i) model-based, ii) data-driven based, and iii) hybrid model (i.e. combination of model-based and data-driven based). In the data-driven based model, physical model describes the degradation of a system. However, in data-driven based model, real-time data gathering as well as making a reasonable connection between the data causes a model which is based on mathematical concepts. For instance, Artificial Neural Network (ANN), Bayesian network, Fuzzy Logic, and K-Nearest Neighbor (KNN) can be used as data-driven models. In the hybrid model, in primary step modeling is based on physical knowledge and the update of shaped model is done by data-driven model. Hybrid model requires physical knowledge which is a hard-hitting step; indeed, utilization of data-driven model is computationally expensive [2].

In the conventional method of prognostic four input arguments are considered i) degradation, ii) health indicator estimator, iii) threshold of failure(s), and iv) RUL. Yet, in the methodology utilized in this study, data set which is gathered from an operating dragline in opencast coal mine in Turkey has been fed to agent and as result the classification/prediction function has been generated. Therefore, based on the capability of the predicted function RUL can be estimated. Direct mapping between dataset and RUL has been modeled by K-Nearest Neighbor (KNN) algorithm. By utilization of the generated approximation function, the RUL can be estimated in any step of operating status. In this study, two steps have been considered; firstly, an approximation function, based on the gathered data was shaped, and secondly, RUL of system has been done by the shaped approximation function (Figure 1).
As shown in Figure 1, RUL assessment is determined from knowledge which is projected using KNN algorithm. Remaining time-to-failure is estimated regarding to knowledge discovered from the approximation function. In order to increase RUL, prediction of failure type can be helpful, so classification of failure type which causes prognostic action has been considered in this study. Time-to-Failure (TTF) is used as input vector to the approximation function and the prediction of failure type as an output vector (i.e. this study copes with a supervised learning machine learning issue). In the utilized methodology, eliminating the setting of threshold for failure is an advantage; on the other hand, tuning the algorithm is a disadvantage. The novelty of proposed methodology is utilization of the supervised prediction model without reprocessing the same data for the same dragline. The remaining part of this study is divided as follow. Section II represents the background information of machine learning and its application. Indeed, it encompasses the KNN background information. In section III, general back ground information about a dragline as well as the KNN background information. In section III, general back ground information about a dragline as well as implementation of KNN on the dragline will be discussed. Finally, in section IV, conclusion and some comments is included.

2. Background information about machine learning

Machine learning is one of the Artificial Intelligence (AI) sub-subjects, which gives computer an ability to learn based on any type of data without explicitly programming. It contains pattern recognition as well as computational learning theory. Machine learning shapes an algorithm based on data (i.e. is called data-driven prediction or decisions). The machine learning scopes are very close to computational statistics which uses computer as a mean of prediction-making tool. A core concept of machine learning is optimization which involves mathematical optimization tools. Machine learning has three types such as: i) supervised learning, ii) unsupervised learning, and iii) reinforcement learning. In the supervised learning, dataset contains the input and desired output and machine is being learning based on the mentioned type of dataset. However, in the unsupervised learning, machine faces with dataset which only contains input and desired output does not exist. In the reinforcement learning, machine has an interaction with a dynamic circumstance which learns from trial and error as well as punishment. Furthermore, supervised learning is divided into two categories which are classification and regression. Recent studies (Hurtado et al. [3], Hurtado[4], Zhiwei [5], Gomes et al. [6] Schueremans et al. [7], Roco et al. [8], Chua [9]) have implemented the machine learning as a methodology for handling the prediction/ classification issue(s). One of the famous methods for classification is KNN. KNN is a non-parametric algorithm and, it does not get any assumption on the data distribution. Since it gets all dataset in itself and tries to assess based on all data it is called lazy algorithm. Besides KNN made a decision based on whole of training dataset, so it requires more time and memory in contrast to other machine learning algorithms. Regards to non-parametric ability of KNN, it can be utilized for data which does not have any typical assumption. KNN implement the concept of Euclidean distance or any common used distance measuring algorithms in order to make relation between data (i.e. called feature space). Because of evaluated distances, data which are in the same distance put in the same bins, so classification is done by KNN. In this study, KNN has been utilized in order to make a classification for a dragline’s failure type.

3. The dragline background information and KNN Implementation

Overburden stripping is done by draglines which are extensively used in opencast mine. In this case, reliability as well as availability of draglines has a great effect on the overall productivity of mine. Annually production of dragline is around 35 million cubic meters [10]. In order to increase the reliability of a dragline (i.e. the whole system) the increment of a draglines sub-system’s reliability should be considered. The sub-systems of draglines are hoisting, dragging, walking, and finally swing. A draglines sub-system failure causes production losses which is around one million dollar per day [11]. In order to increase the reliability of dragline, an accurate prediction of failure type before it takes a place can be avoided (i.e. which is called prognostic). In this sense, KNN, in this study, has been utilized in respect to handle the prediction procedure. The Time to Failure (TTF) (Table 1) has been fed to KNN algorithm as an input (feature) vector; indeed, a dragline’s failure types as an output (response) vector is fed to KNN (i.e. Table 1. represents the part of input as well as output). Number of observation, which is utilized in this study, is 1300; indeed, number of feature and response is one (i.e. the sample part of dataset has been illustrated in Table 2). Python and scikit-learn library has been used as programming language and related library, respectively.
There are some benefit in order to the utilization of Scikit-learn library such as: i) consistent interface to machine learning, ii) tuning the parameter and hyper-parameter can be done with sensible default, iii) it contains lots of helpful documentation, and iv) there is an active community for support as well as development. However, there is a drawback for scikit-learn library which is less emphasis on model interpretability. In this sense, KNN in respect to the dragline’s data has been implemented in the scikit-learn in two fashions (i.e. it has been divided to two fashions in case of evaluation procedure):

1. Training and testing data are based on the entire dataset,
2. Splitting the dataset to training and testing data for cross-validation methodology based on parameter tuning by using of:
   - Grid Search cross-validation,
   - Searching multiple parameters simultaneously, and
   - Searching multiple parameters in order to reduce computational expense by randomized search cross-validation.

Based on the first fashion which is utilization of whole dataset for training and testing, it causes over-fitting, so the training accuracy will be high; on the other hand, over-fitting causes lower accuracy for out-of-sample data which will be utilized for prediction. Therefore, the accuracy of generated approximation function is low. Due to low accuracy of model, cross-validation fashion has been utilized in three manners, so the result was 73 percent which is acceptable regression analysis.

### 4. Conclusions

Draglines has a critical role in open cast mine particularly in coal mine. In this sense, availability and reliability of dragline is a vital task which is necessary to be considered. Losses in profit (i.e. direct and indirect) as well as higher operating costs are some of dragline’s breakdown causes. In order to handle the breakdown, a preventive action can be taken into account such as prediction of failure type. By accurate prediction of failure and by its avoiding, the Remaining Useful Life of dragline can be increased. In this sense, K-Nearest Neighbor algorithm as a supervised machine learning method has been utilized in this study. Based on dataset, which was gathered from an operating dragline in a coal mine in Turkey, a model has been trained and test data has been fed to it, so the generated model represents its 73 percent of accuracy. In the shaped model (i.e. shaped by KNN), splitting training and test data has been done by cross-validation methodology in order to better evaluation of model. Indeed, regards to tuning of parameter as well as hyper-parameter some manners have been implemented on the model such as: i) grid search cross-validation, ii) searching multiple parameters simultaneously, and iii) randomized search of multiple parameter. Finally, in order to improve the regression analyze utilization of more features with the higher number of observations should be considered in future study.

### References


Constructive solutions of equipping crushing and dumping station on an open pit mine while using automobile-conveyor transport

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Abstract— The implementation and operation of cyclical and continuous method on a deep iron ore open pit mines is analyzed. Technical characteristics of preliminary crushers are shown. A basic scheme of constructing loading points for transportation of non-crushed rock is considered. Design solutions of equipping loading points operating the truck-conveyor transport are developed.

Keywords: deep iron ore mines, cyclical and continuous method, the concentration horizon, preliminary crusher, buffer bunker.

1. Introduction

Design of receiving points, having significant dimensions, is known for transportation of non-crushed rock mass by truck-railway transport and by cyclical and continuous method (CCM) [1]. Loading bunkers should ensure rapid unloading of dump trucks and loading of railway transport. It is necessary to ensure that individual units are stable, as well as construction as a whole and efficiency of operations. In this regard, new designs of loading stations have been developed, corresponding to the intensity of mining operations on the open pit mine and its current depth, with smallest possible dimensions in design. Placing them at the center of gravity of rock mass excavation will significantly reduce the transportation distance by truck.

2. Material and Methods

Analysis of the features of the development of the large open pit mines over time allows to make certain conclusions on the transition process from quantity to quality. Phased construction of open pits has proven that even for largest "super mine", time-space allocation and development phases allow initially designing open pits by traditional methods [2]. However, increasing the depth of some open pits to 300-400 m and more (second-order mines) has clearly revealed the limitations or even unacceptability of traditional design methods for third-order mines (Table 1. At the same time, one of the main directions of technical progress in the development of deep open pit mines is the implementation of the TSC of mining operations, using combined automobile-conveyor transport. The most widespread implementation was in the iron ore industry. Road transport in this case is used inside the open pit mine as an assembly at a transportation distance of 1.0-1.5 km from the working face of the loading point.

<table>
<thead>
<tr>
<th>Name</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Open pit order</td>
<td>1 2 3</td>
</tr>
<tr>
<td>Open pit depth, meters</td>
<td>100 400 1000</td>
</tr>
<tr>
<td>Productivity, million tons / year</td>
<td>10 30 100</td>
</tr>
<tr>
<td>- for mineral resources</td>
<td></td>
</tr>
<tr>
<td>- by the rock mass</td>
<td>15 60 400</td>
</tr>
<tr>
<td>Operation term, years</td>
<td>20 50 100</td>
</tr>
<tr>
<td>Number of reconstructions (sequences)</td>
<td>1 3-4 10</td>
</tr>
</tbody>
</table>
3. Results and Discussion

The implementation of cyclic and continuous method with a decrease of the transportation distance by 1-3 km reduces the number of dump trucks BelAZ-7519 by 9-26 pieces, frees 40-120 workers, reduces the consumption of diesel fuel by 2.3-6.8 thousand tons per year and contamination of the atmosphere of the open pit mine 1.5-2.0 times. Using conveyor transport to lift rock mass from a depth of 100-500 m instead of automobile transport reduces the specific metal consumption of the technological process by 2.5-3.8 times and reduces the specific energy consumption by 2.6 times. The most widespread application in the CIS countries was cyclic-flow technology for ore transportation (Kryvbas, Poltavsky, Stoi lensky, Kachkanarsky, Kovdorsky, Olenegorsk ore mining and processing units) and overburden mining at the Muruntau Novoinsk mining and metallurgy complex [3].

The experience of operating deep open pits shows that the development of trend of finding technological solutions makes it possible to minimize the number of dump trucks operated on the mined space, which can be ensured by using mobile handling and conveyor systems as part of existing stationary conveyor systems of the TSC. The expediency of such usage is predetermined by the fact that the construction and transfer of loading points to new concentration horizons as the depth of the open pit increases is usually delayed and; consequently, the length of transportation by vehicles increases significantly.

High power open pit mines, with depth of 250-300 m, and the boundary values of 500-1000 m, are characterized by large volumes of mined rock, reaching 50-150 million tons per year. At the same time, 30-70% of the extracted rock mass is an overburden, which makes it necessary to transport to the surface not only minerals, but also overburden. When the mining depth is reached, the transportation distance of minerals from the lower horizons to the surface of the open pit mine or to the concentration point is basically not less than 3-6 km, and transportation distance of overburden to dumps up to 5-15 km.

The interconnection of combined types of transport is carried out through reloading points. Such points are set at the concentration horizons within the open pit mine. There is a need for the arrangement of points on the underlying horizons, with the decrease of mining operations, which predetermines their insignificant service term (5-7 years). Therefore, such loading points should be modular, portable (mobile) and with a minimum amount of construction work.

The expediency of using one or the other type of loading station is predetermined by the quality of the rock mass preparation by the explosion, and by the type of rock mass (ore and overburden). Mainly crushing stations are operated on iron ore open pit mines, while using cyclical and continuous method in mining operations, as a result of which the first stage of crushing is eliminated at the concentration plants [4]. Such points ensure the efficiency of the entire transport and technological complex. There is no need for additional crushing of the rock while conducting stripping operations. Therefore, the minimum energy costs for the preparation of the rock for transportation by the conveyor are ensured by the use of effective screening points.

Disturbance of cyclical transport stability, while using any structures of transport and transshipment complexes, leads to various delays, the underload of transport and change of the amount of work. Therefore, the establishment of rational technological schemes and parameters of transport and transshipment facilities, taking into account the reliability of their operation and traffic fluctuations, precise interconnection between transport means are needed to carry out the planned work during the use of multilink transport complex. There is a tendency to increase the use of combined transport and buffer stores with an increase of the open pit mine depth in foreign countries [5, 6].

Crushing stations include the following main units: a receiving hopper, a plate feeder, and crushers. Commercially available cone and jaw crushers are used for crushing hard and very hard rocks (f = 12 ÷ 20).

The use of crushers at a loading station ensures the transportation of all incoming rock mass by belt conveyors. At the same time, with the use of crushers, the metal and energy consumptions, the construction height of the loading station and the capital costs for its construction increase. Crushing stations are characterized by a significant increase of the loading cost, increased construction terms, the complexity of transfer to a new concentration horizon [7].

The height of the loading points with conical (KKD-1500/180) and cone-shaped (KVKD-1200/200) crushers reaches 25-30 m, which leads to the secondary lift of the rock mass by road. A crushing station with a single cone crusher limits the annual capacity of conveyor systems to 20 million tons with a possible carrying capacity of up to 30-35 million tons. This requires the installation of two crushing stations or a combination with screening plants.

An important parameter of a loading point is the linear dimensions of the hopper along its upper edges at the loading points of dump trucks. They affect its overall dimensions and weight. On the other hand, the bunker parameters should ensure unloading of dump trucks with a given intensity of traffic. Considering the constructive schemes of crushing and loading points (CLP) for automobile-conveyor transport (Fig. 1), it should be noted that the necessary volume of reception and subsequent transfer of crushed rock can be formed before the preliminary crusher (Fig 1a, b, c), and after it (Fig. 1d).
Figure 1. Designs of equipping dumping stations while using automobile-conveyor transport: a – without installment of storage hopper; b – with storage hopper above crusher; c – near storage hoppers above crusher; d – with storage hoppers under crusher; 1 – preliminary crusher; 2 – ordinary rock mass; 3 – dump truck; 4 – vibrating feeder of VFR type; 5 – belt conveyor; 6 – high wall of unloading area; 7 – storage hoppers; 8 – conveyors of a receiving hopper.

In the first case, volume of receiving hopper should be sufficiently large, designed for significant intensity of unloading of rock mass by required amount of dump trucks. Order of their reception and unloading for one receiving hopper is limited by the carrying capacity of dump trucks, and by time, they were received. In interval between shifts, crusher is idle. Usually annual capacity of crushing and dumping station of such design is limited by 20 million tons.

Placing several storage hoppers, usually no more than three, is associated with possibility of storing significant amount of rock mass. It should be noted, that placing storage bunkers above the crusher (Fig. 1c) is more preferable by organizational matters. Thus, during the shift they could be loaded simultaneously with significant amount of dump trucks on crushing and dumping station. Increase of intensity of rock mass reception could be at the level of 40-50 % of a shift’s mean value. Limitation of freight transport would be provided only by capacity of the crusher.

Placing storage hoppers under the crusher (Fig. 1d) does not solve the problem of full loading. In this case, reception of rock mass is limited by the capacity of the hopper, which in the main period of shifts is equal to average operational value, in intervals between depends on reception intensity and carrying capacity of dump trucks. It is idle in the interval between shifts, what was reflected in the analysis of crushing and dumping station’s operation on enterprises, where research was conducted.

4. Conclusions

It should be noted from conducted research, that perspective design of crushing and dumping station on deep open pit mines should include placement of buffer hoppers above preliminary crusher. Annual capacity of such crushing and dumping station would be predetermined by corresponding capacity of the crusher, its operation regime and technical service, also ensuring filling and unloading of each of the three storage hoppers.

Loading bunkers should ensure rapid unloading of dump trucks and loading of conveyor lines. It is necessary to maintain high reliability of individual units and construction as a whole, as well as the efficiency of the operations. Since mining deep open pit mines is characterized by a relatively narrow working platforms and haulage bench, dimensions of loading bunkers should match the height of pit banks and fit constructively to the configuration of operating and non-operating open pit edges.
References


Whole Body Vibration Assessment of the Mining Truck Drivers; a Case Study

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Abstract- Mining haul trucks expose their drivers to the dangerous level of whole body vibrations because of their various working cycles and also passing on the roads with different qualities. This paper reports an experimental study to fine the effects of working conditions on the whole body vibration of mining truck driver. To achieve this goal, first, the root mean square of vibration at different truck speed, weight and geometry of load in the truck bucket and also in different road qualities are measured. Then, the health risk levels of vibrations are analyzed according to ISO 2631-1 standard. Moreover, the artificial neural network used to predict the vibrational health risk level at different working conditions.

Keywords: Mining truck, whole body vibration, working condition, artificial neural network.

1. Introduction

Equipment and machineries in heavy industries such as mining expose their operators to harmful vibration [1-2]. If this vibration beyond the tolerable levels, will be associated with work-related back pain and harmful effects on the operator’s health. Reviewing the past studies shows that the vibrational health risk analysis of many machineries in open pit mines such as trucks [3-6], shovels [7] and drilling machines [8-9] have been done and analyzed. Among these equipment, mining trucks usually operate at different working cycles including; loading, transferring materials from loading to dumping area, dumping and returning to the loading area and also in roads with different qualities, therefore they are exposed to the high level of vibrations.

Smets et al. [3] studied the drivers’ whole body vibration (WBV) of eight dump trucks at metalliferous surface mines in Canada. The results showed that the truck operators were typically exposed to moderate through high level of WBV risk and truck type had no significant effect on vibrations. Mandal et al. [7] measured WBV of 66 dump trucks in 10 open pit mines of India. Results of this study showed that 26%, 54% and 20% of dump trucks, respectively were in the high, moderate and low risk levels. Wolfgang and Burgess-Limerick [10] measured the WBV exposure of 32 haul trucks at a surface coal mine during normal operating conditions. Regarding to the results of this study, the root mean square (RMS) values of vertical vibrations for 12 trucks were in low health risk level. Results of this study shows that the haul road conditions had a large effect on vibrations. In a recent study, Burgess-Limerick and Lynas [5] measured and analyzed WBV of 16 rear dump trucks (ranging from 177 to 313 tons) and other equipment at a surface coal mine. Results of this study showed that haul trucks were frequently exposed to vertical vibration in the moderate to high levels.

From the studies mentioned above, there are limited researches to consider the effect of operational conditions such as speed, payload and load geometry or loading quality in the truck dump body. Therefore, in this study to overcome these deficiencies, a 60-ton truck at Sungun Copper Mine of Iran is selected for vibrational health risk analysis during the various working conditions. These working conditions are road quality, truck speed, payload and load geometry or concentration of materials on the left and right sides of the truck dump body and also uniformly accumulated materials. Then, an Artificial Neural Network (ANN) is used to predict the vibrational health risk.

The rest of this paper is organized as follows. In section 2, ISO 2631-1 standard is introduced as the most popular standard in analyzing the WBV risk level. In section 3, artificial neural network, as a powerful tools in prediction problems, is introduced. In section 4, the case study and data collection procedure are discussed. Finally, in section 5, the WBV at different operational conditions are analyzed then the ANN is used to predict the vibrational health risk levels.
2. Evaluation of Whole Body Vibration Exposure

There are different standards to analyze the WBV, while, ISO 2631-1 is the most commonly used standard to describe whole-body vibration amplitude. This standard applies two main parameters for vibration analyzing; the frequency-weighed root mean square (RMS) of acceleration ($a_{\text{w rms}}$) and Vibration Dose Value ($VDV$) as follows [11]:

$$a_{\text{w rms}} = \sqrt{\frac{1}{T} \int_0^T a^2_w(t) \, dt}, \quad (1)$$

$$VDV = \sqrt[4]{\frac{1}{T} \int_0^T a^4_w(t) \, dt}, \quad (2)$$

Where, $a_w$ is the frequency-weighted of acceleration over a time period $t$ (m/s$^2$) and $T$ is the measurement duration (s). ISO 2631-1 uses Health Guidance Caution Zone (HGCZ) to show the human WBV exposure thresholds. On the other hand, health risk is evaluated by upper and lower bounds of HGCZ. The upper and lower bounds of HGCZ respectively are 0.47 and 0.93 m/s$^2$ in terms of RMS and 8.5 and 17 m/s$^{1.75}$ in term of $VDV$ [11].

Crest Factor ($CF$) defined as the ratio of peak and RMS of acceleration which is used to measure the shocks within the vibrations signals. When the $CF$ is greater than 9, vibration effects on the driver’s body may not be estimated and the $VDV$ is used to estimate the exposure levels.

3. Artificial Neural Network

The artificial neural network (ANN) is an information processing tools used to simulate the human brain structure and functions which is able to represent the complex relationship between input and output of system. Neural networks has two main steps. At the first step, the knowledge for NN is required through learning and at the second step, the network knowledge is stored in the strengths of inter-neuron connection known as synaptic weights [12].

The most common NN model is multilayer perceptron (MLP) which requires a desired output in order to learn. One of the supervised algorithms in MLP is feed-forward back-propagation network which consists of one input layer, one or more hidden layer(s) and one output layer. The output layer comprises neuron(s) corresponding to the value to be predicted. Each node of the output layer is linked to all nodes in the hidden layer and all hidden layer nodes are linked to all input layer nodes. All nodes are linked with each other by the weighted connection [12]. The error signal, between the input and the output, represents the network power in the knowledge learnt. Accordingly, to recognize the optimum network, different network architectures are tried by calculating Mean Square of Error (MSE) as the total error function.

4. The Case Study

Data collection was done in Sungun Copper Mine, Iran. This mine is located in the distance 130 km from Tabriz City in northwestern of Iran. The probable reserves of mine has been estimated as about 1700 million tons [13].

A Komatsu HD785 dump truck with 60 tons capacity was selected for all tests and data collections. During the data collection, truck speed was recorded in the range of 15 to 35 km/h using a portable GPS. To consider the load geometry in each loading cycle, it was requested loader operator to load materials, uniformly or accumulate them on the left (driver) or right sides of the truck dump body, as much as possible. Different accumulation sides of materials are shown in Figure 1. Moreover, the weight of materials was recorded using the truck payload meter.

Figure 1. Accumulation of materials (a) uniformly or on the (b) left and (c) right sides
To consider the effect of haul road, the entire haul road of the mine was inspected and classified into two classes; good quality with compacted and smooth gravels and poor quality with roughed surface because of poor maintenance. In this study, ADXL335 accelerometer was used to measure vertical vibrations. Each sensor was placed in a compacted plastic pad was secured on the seat of truck. In this study, accelerometer was programmed to collect data and saved onto a recording program installed on a laptop.

5. Data Analysis

In this section, the collected data at the various working condition are analyzed and discussed. To study the effect of different levels of each working conditions on the mean RMS values the analysis of variance (ANOVA) is used. Analysis of variance is the most known statistical method to define independent parameters which have significant difference on the dependent parameter. The results of ANOVA at 5% significant level obtained by SPSS software [14] are given in Table 1. According to Table 1, there is significant difference between truck speed, road quality, load geometry and payload in the RMS values ($p < 0.05$). Other two-way interactions are given in Table 1, too. The interaction between the road quality and truck speed revealed the significant differences in RMS values.

Table 1. Results of ANOVA for RMS data set

<table>
<thead>
<tr>
<th>Variable</th>
<th>df</th>
<th>$F$-Value</th>
<th>Significant</th>
</tr>
</thead>
<tbody>
<tr>
<td>Speed</td>
<td>3</td>
<td>76.443</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Road quality</td>
<td>1</td>
<td>58.532</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Payload</td>
<td>4</td>
<td>14.791</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Load geometry</td>
<td>2</td>
<td>14.211</td>
<td>&lt;0.001</td>
</tr>
</tbody>
</table>

Two-way interaction

<table>
<thead>
<tr>
<th>Two-way interaction</th>
<th>$F$-Value</th>
<th>Significant</th>
</tr>
</thead>
<tbody>
<tr>
<td>Road quality × speed</td>
<td>3</td>
<td>3.526</td>
</tr>
<tr>
<td>Load geometry × payload</td>
<td>7</td>
<td>1.510</td>
</tr>
<tr>
<td>Road quality × load geometry</td>
<td>2</td>
<td>1.136</td>
</tr>
<tr>
<td>Road quality × load geometry</td>
<td>4</td>
<td>1.014</td>
</tr>
<tr>
<td>Load geometry × speed</td>
<td>6</td>
<td>0.587</td>
</tr>
<tr>
<td>Payload × speed</td>
<td>12</td>
<td>0.518</td>
</tr>
</tbody>
</table>

To study the effect of truck speed levels and load geometry on the RMS values, Scheffe’s Post Hoc Test (SPHT) is applied. Scheffe test is the mean comparison test used for finding relationships between sub-groups of significant parameters [14]. Results of SPHT are given in Table 1. In this Table, there is no significant difference between each subgroups of operational conditions which have the same symbol ($p > 0.05$).

Scheffe test reveals that there is significant difference between all truck speed ranges. The uniform load or accumulated on the right side has the lowest effect on RMS values, while, when the materials are concentrated on the left side of the dump body, the effect of load geometry of vibrational health risk is in the highest level.

Table 2. Results of SPHT for RMS data set

<table>
<thead>
<tr>
<th>Truck speed (km/h)</th>
<th>Load geometry</th>
<th>Left</th>
<th>Right</th>
<th>Uniform</th>
</tr>
</thead>
<tbody>
<tr>
<td>30-35</td>
<td></td>
<td>1.184</td>
<td>0.936</td>
<td>0.787</td>
</tr>
<tr>
<td>25-30</td>
<td></td>
<td>0.400</td>
<td>0.902</td>
<td>0.823</td>
</tr>
<tr>
<td>20-25</td>
<td></td>
<td>0.787</td>
<td>0.823</td>
<td>0.741</td>
</tr>
<tr>
<td>15-20</td>
<td></td>
<td>0.587</td>
<td>0.853</td>
<td>0.782</td>
</tr>
<tr>
<td>10-15</td>
<td></td>
<td>0.518</td>
<td>0.8549</td>
<td>0.7795</td>
</tr>
<tr>
<td>5-10</td>
<td></td>
<td>0.518</td>
<td>0.8549</td>
<td>0.7795</td>
</tr>
</tbody>
</table>

In the rest of this section, the artificial neural network is used for predicting the health risk levels. To achieve this aim, four operational conditions including; haul road quality, speed range, payload and load geometry are considered as input layer data. The output layer is composed of a single parameter; the $a_{rms}$. In this study, 304 field data are considered for the vibrational health risk prediction. The first data set consists of about 85% of all field data are used for creating the network architecture and the others, selected randomly, are used for network performance evaluation.

A feed-forward back-propagation network is applied. Logarithmic sigmoidal and purely linear are, respectively, selected as transfer function for the hidden and output layers. Also, training of the network is implemented by Levenberg-Marquardt Algorithm. To select the best network, several networks with different architectures are created using the back-propagation algorithms available in MATLAB software. The best results are obtained according to different neurons and layer numbers and given in Table 3. According to Table 3, the best values of correlation coefficient for the training and validation stages are 87.03 and 94.90%, respectively. Therefore, the 7-7-1 network architecture is selected as final network.

Table 3. The best results obtained from different neuron and layer numbers

<table>
<thead>
<tr>
<th>ANN</th>
<th>$MSE$</th>
<th>Training</th>
<th>Validation</th>
<th>Test</th>
</tr>
</thead>
<tbody>
<tr>
<td>9-1</td>
<td>0.0165</td>
<td>0.8641</td>
<td>0.8233</td>
<td>0.9124</td>
</tr>
<tr>
<td>10-1</td>
<td>0.0199</td>
<td>0.8766</td>
<td>0.8549</td>
<td>0.7795</td>
</tr>
<tr>
<td>6-7-1</td>
<td>0.0173</td>
<td>0.8738</td>
<td>0.8901</td>
<td>0.9080</td>
</tr>
<tr>
<td>6-8-1</td>
<td>0.0183</td>
<td>0.8875</td>
<td>0.8231</td>
<td>0.7827</td>
</tr>
<tr>
<td>7-6-1</td>
<td>0.0205</td>
<td>0.8804</td>
<td>0.8356</td>
<td>0.8850</td>
</tr>
<tr>
<td>7-7-1</td>
<td>0.0085</td>
<td>0.8703</td>
<td>0.9490</td>
<td>0.8176</td>
</tr>
<tr>
<td>7-8-1</td>
<td>0.0251</td>
<td>0.8845</td>
<td>0.8536</td>
<td>0.8669</td>
</tr>
<tr>
<td>8-8-1</td>
<td>0.0148</td>
<td>0.8648</td>
<td>0.8759</td>
<td>0.8754</td>
</tr>
</tbody>
</table>

In order to performance analysis of the selected network, 60 random actual data are compared with the data estimated by ANN. Regarding the performance analysis results, there is 80.13% correlation between the actual and estimated RMS values (Figure 2). Therefore, the proposed
network can be applied for estimation of the health risk level.

![Figure 2. The correlation between the actual and predicted RMS values](image)

6. Conclusions

In this paper the vibrational health risk of a mining truck during different operational conditions including speed, payload load geometry and also haul road quality is investigated and discussed as a case study: A 60-ton truck in Sungun Copper Mine, Iran. Then, the artificial neural network is used to predict the vibrational health risk at the various working condition. Results of this study show the truck speeds load geometry and also haul road qualities have the significant effect on truck vibrations. Materials which are distributed uniformly and accumulated on the left side of the dump body respectively have the lowest and highest effects on the vibrational health risk. In the use of the proposed feed-forward back-propagation neural network with 7-7-1 architecture, there is high correlation between the observed and predicted RMS values.

Acknowledgement

The authors would like to thank workers and managers of Sungun Copper Mine of Iran for their kind support during the field studies and data collections. Special thanks also go to Eng. Marami and Eng. Salehi from Mobin Mining and Construction Company and Dr. Hamid Aghababaei from Department of Mining Engineering of Sahand University of Technology for their kind helps and guidance.

References


MPES2017

Session 8 – Mine Equipment II

13:30-15:00

August 30, 2017
Changes in the mineralogy of rocks from the region of underground coal gasification in the Upper Silesian Coal Basin, Poland

Robert HILDEBRANDT, Zbigniew BZOWSKI
Central Mining Institute, Katowice, Poland

Abstract—Examinations of changes in the mineralogy of rocks originating from the region of underground in situ coal gasification were carried out after the completion of the process executed as a part of a research project. The transformed material and the reaction product (waste) were analyzed by means of X-ray powder diffraction using the Bragg-Brentano geometry. Results of the mineralogical investigations of the rock samples from the region of underground coal gasification presented three interesting systems of phases. The first concerns a temperature-based transformation of primary minerals such as: kaolinite, illite, chlorite, biotite, muscovite, pyrite, calcite, dolomite, siderite and mixed-layer minerals. The second system consists of coexistent phases of augite, forsterite, and magnetite. The third system of phases concerns the cooling period and is characterized by the presence of secondary and frequently hydrated minerals such as gypsum and hexahydrate.

Keywords: underground coal gasification, mineralogy, X-ray.

1. Introduction
One of the alternative ways of obtaining energy from coal is the conversion of this raw material to gas, carried out directly at the place of the deposit occurrence. This process, called underground coal gasification, takes place in the deposit, access to which is obtained by drilling holes from the surface or by using shafts and underground workings. The underground part of a coal seam prepared for gasification is called a geo-reactor or gas generator. In the very simple configuration of a geo-reactor, two vertical holes are connected to each other with a fire channel in the coal seam, where one of them leads the gassing factor and the other transports the produced gas [1, 2, 3]. The simple idea of underground coal gasification process is shown in Figure 1.

![Figure 1. Underground coal gasification process scheme](image)

The assumptions of the underground coal gasification method and, above all, the need to maintain the tightness of the geo-reactor, preclude the possibility to observe the underground process. Moreover, the monitoring of the parameters of the gasification course, including temperature, is extremely limited and amounts to indirect methods of measurement. To solve these problems numerical methods for modelling the process course are used, whose basis is the tests carried out on the surface [4]. A similar research situation occurs in the assessment of the changes in mineralogical rock covers of the geo-reactor and the influence of the temperature on both rock and ash residues after the gasification of coal. So far, the assessment of the impact of temperature on rocks in the vicinity of the geo-reactor has been carried out on the basis of changes such as swelling, cracking, melting and phase transformation of heated samples in laboratory conditions [5, 6]. However, in this work, the mineralogical studies of rock and waste samples, taken directly from the geo-reactor after the end of the process and after its cooling, were performed.
2. Material and Methods

Examinations of changes in the mineralogy of rocks originating from the region of underground *in situ* coal gasification were carried out after the completion of the process executed as a part of a research project. The two month long experiment of underground coal gasification was performed in an operating mine, where 230 tons of hard coal were gasified, producing over 1,000,000 m³ of gas. Subsequently, a six month period of cooling by pumping nitrogen into the underground georeactor took place [7].

The transformed material and the reaction product (waste) was analyzed by means of X-ray powder diffraction using the Bragg-Brentano geometry. The analysis was performed by utilizing a Bruker D8 DISCOVER diffractometer, a CuKα tube, a Ni filter and a LYNXEYE_XE detector. Mineral composition was determined by use of the licensed databases of diffraction patterns: PDF-4+ 2015 RDB, ICDD and NIST.

3. Results and Discussion

3.1. Temperature impact on the changes in the process of underground coal gasification

In underground coal gasification, zones of oxidation, reduction, drying and pyrolysis (carbonization) are created around the fire channel of the geo-reactor. Natural and technological factors determine the course of the process. Among the natural factors a dominant role is played by geological parameters and the characteristics of the coal in the seam for gasification. Technological factors include the type of gasifying measures that affect the course of the reaction and the existence of these zones. The main product is the gas occurring, simultaneously, in the processes of degassing (pyrolysis) and gasification. While the temperature influence on the body of coal, heated with the heat of the progressive front fire, causes its degassing and drying, in places where coal directly reacts with an oxidizer, the gasification reaction takes place [2, 8].

Temperature and the oxidizing factor constitute two major factors in the process of mineralogical changes in rocks from the region of underground coal gasification. The oxidizing factor decreases along the region of gasification in the direction of the exhaust. As a result of gasification, the total volume of coal is reduced, which results in the occurrence of a hollowness filled with reaction products (waste) as well as transformed ceiling rocks: the main focus of this article.

The sample post-reaction material and metamorphosed rocks from the geo-reactor are shown in photographs (Figs. 2 & 3).

Figure 2. Fragments of the underground coal gasification process slag

Figure 3. Changes on the surface of burnt sandstone from the geo-reactor

3.2. Mineralogical research

Results of the mineralogical investigations of rock samples from the region of underground coal gasification presented three interesting systems of phases. The first concerns a temperature-based transformation of primary minerals such as: kaolinite, illite, chlorite, biotite, muscovite, pyrite, calcite, dolomite, siderite and mixed-layer minerals. The second system consists of coexistent phases of augite, forsterite, and magnetite (Figs. 4 & 5).

Figure 4. Augite and magnetite found in a sample of underground coal gasification process slag

[Graph showing Augite and Magnetite peaks]
Most probably such a system arose over a wide range of temperatures between 900°C and 1200°C. The third system of phases concerns the cooling period and is characterized by the presence of secondary and frequently hydrated minerals such as gypsum and hexahydrate (Fig 6).

In addition to the phases listed in the three mineral groups found in the rocks, the geo-reactor sediments and slag (waste) after the process of underground coal gasification, such phases occur that cannot be classified to these groups. The presence of the secondary hematite, with partially glazed SiO₂ phases as well as anhydrite and mullite (in rare cases) indicates that some samples of slag constitute glazed ash occurring in gasified coal. The chemical composition of coal ash treated with underground gasification is presented in Table 1. This statement shows that sulphide sulphur and sulphur from coal may be of about 20% of the SO₃ partially discharged from the manufactured gas. However, SO₃ constitutes the basis of the presence of thermally formed anhydrite, gypsum and hexahydrate which were found, and possibly of other secondary sulphate minerals. Research analysis of burning mining waste dumps shows that minerals such as jarosite, bassanite, rozenite, melanterite, alunite and sideronatrite should be expected. The issue of secondary sulphate mineralization together with hematite as phases related to the thermal transformation of ash from coal used in underground gasification, requires more mineralogical-
The third system of crystalline phases, in the underground coal gasification zone, concerns the cooling stage and the occurrence of secondary minerals, which are often hydrated, i.e. gypsum and hexahydrate. In addition, in slags generated from ash contained in the gasified coal of the geo-reactor zone as a secondary phase, hematite, anhydrite and mullite have been found. However, in this slag the system of phases: minerals SiO$_2$-mullite-hematite require much more profound research.

References


4. Conclusions

This work, using the results of mineralogical tests of samples of rocks extracted from an underground gasification zone, shows the changes in the mineralogy of rocks from the region of underground coal gasification in the Upper Silesian Coal Basin, Poland.

It can be concluded that thermal transformations associated with underground coal gasification consist primarily of clay minerals (kaolinite, illite, chlorite, biotite, muscovite and mixed-layer materials) occurring both in the rocks surrounding the geo-reactor and in the coal. These minerals are often accompanied by sulphides and carbonates (pyrite, calcite, dolomite and siderite) that are also subject to thermal changes in the first place.

The second group of minerals in thermally produced slag is the system of phases: augite, forsterite, and magnetite. This system is probably formed over a wide range of temperatures from above 900°C to about 1200°C. Temperature changes in the geo-reactor, resulting from the properties of the process, being subsequent to the application of a variety of gasifying factors including oxygen, may cause the temperature of the synthesis in the augite-forsterite to be greater than 1200°C.

Table 1 – Chemical composition of coal ash treated with underground gasification in %

<table>
<thead>
<tr>
<th>Components</th>
<th>Ash</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO$_2$</td>
<td>9.49</td>
</tr>
<tr>
<td>TiO$_2$</td>
<td>0.20</td>
</tr>
<tr>
<td>Al$_2$O$_3$</td>
<td>6.17</td>
</tr>
<tr>
<td>Fe$_2$O$_3$</td>
<td>11.42</td>
</tr>
<tr>
<td>Mn$_3$O$_4$</td>
<td>0.76</td>
</tr>
<tr>
<td>MgO</td>
<td>18.94</td>
</tr>
<tr>
<td>CaO</td>
<td>30.91</td>
</tr>
<tr>
<td>Na$_2$O</td>
<td>1.04</td>
</tr>
<tr>
<td>K$_2$O</td>
<td>0.14</td>
</tr>
<tr>
<td>SO$_3$</td>
<td>20.02</td>
</tr>
<tr>
<td>P$_2$O$_5$</td>
<td>0.01</td>
</tr>
</tbody>
</table>

The system of phases: minerals SiO$_2$-mullite-hematite in the slag remaining after this process requires much more thorough research.

4. Conclusions

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<td>SO$_3$</td>
<td>20.02</td>
</tr>
<tr>
<td>P$_2$O$_5$</td>
<td>0.01</td>
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</table>

The third system of crystalline phases, in the underground coal gasification zone, concerns the cooling stage and the occurrence of secondary minerals, which are often hydrated, i.e. gypsum and hexahydrate. In addition, in slags generated from ash contained in the gasified coal of the geo-reactor zone as a secondary phase, hematite, anhydrite and mullite have been found. However, in this slag the system of phases: minerals SiO$_2$-mullite-hematite require much more profound research.

References

Safety Compliance in a Rwandan Underground Gallery – A Practical Study

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Department of Mining Engineering and Survey, University of Johannesburg, Johannesburg, South Africa

Abstract—Mining in Rwanda is mainly practiced using artisanal mining techniques. The mining techniques are labor intensive, utilizing rudimentary tools to mine and with work practices that often do not adhere to safe mining standards as prescribed by local authorities. Over the past two years 80 Artisanal miners operating in Rwanda have been fatally injured, which is a concern for the Rwandan government. In response to the high number of deaths the Rwanda government has begun to restructure the mining regulations. However,

The mining industry, government departments and Non-Government Organizations (NGOs) recognize that the Rwandan mining industry requires improving its safety standards. Further it is recognized that modifying the mining regulations is not enough to reverse this trend and there is a real need to improve the knowledge and skill base of the mining industry. While the country’s five to six small scale mines have mining engineering staff who direct and coordinate safety measures as necessary it is believed that well over half to three quarters of Rwanda’s artisanal miners are working underground with no effective occupational health and safety program. Commonly, portals, tunnels, and working areas are unsupported with underground workings positioned in shallow and located often in colluvium or weathered schist.

In Rwanda, the Ministry of Natural Resources through the Geology and Mines Department of the Rwanda Natural Resources Authority is mandated to carry out mining inspections for all mining operations. Regular inspections of the mines are aimed at enforcing the country’s mining guidelines and standards, as well as to ascertain compliance with environment, safety and health regulations.

Despite these efforts, the local knowledge base and the availability of trained mining engineers is not sufficient to address the safety challenges of Rwanda’s artisanal small scale mining industry.

Hazard identification and risk assessments skills are critical attributes that miners, supervisors and owners must have in order to comply with the standards set for occupational health and safety of mine workers. Based on the compliance assessment it is apparent that these skills aren’t readily available in Rwanda. Further, review of the annual Mine Inspector Form would indicate that government mine inspectors also lack the necessary skills to identify many of the risk associated with artisanal small scale mining.

This paper documents the actual mining conditions found in an underground Rwandan mine. The paper highlights and discusses the requirements to improve the general skills and knowledge in terms of hazard identification and risk assessment, which should lead to improved occupational safety and health conditions in the small scale mining environment of Rwanda.

Keywords: artisanal small scale mining, safety audit, Rwanda mining

1. Introduction

Mining in Rwanda is mainly practiced using artisanal mining techniques. Artisanal mining is labor intensive, utilizing rudimentary tools to mine with work practices that often do not adhere to the mining standards as prescribed by local authorities. The main objective of any mine is to maximize production and grade whilst minimizing costs. The most obvious way for an artisanal mine to reduce costs is to cut-out money spent on safety or anything else that doesn’t directly add to the extraction and recovery of metal concentrate. In contrast to this approach, the mine site operator or mine owner is obligated to ensure workers and working conditions conform to occupational health and safety standards as required by the government.

The mining industry, government departments and Non-Government Organizations (NGOs) recognize that the Rwandan mining industry requires improving its safety standards. While the country’s five to six small scale mines
have mining engineering staff who direct and coordinate safety measures as necessary it is believed that well over half to three quarters of Rwanda’s artisanal miners are working underground with no effective occupational health and safety program. Commonly, portals tunnels, and working areas are unsupported with underground workings positioned in shallow and located often in colluvium or weathered schist.

Historically, there have been international organization and government initiatives to deliver technical support programs to the artisanal mining sector. For example, the EU-funded COPIMAR program provided engineering and geological advice to artisanal cooperatives organized in six geographical regions from 1988 to 1994. In 2010, the OGMR gave a 2-week course in underground mining methods to cooperatives, and the BGR developed an occupational health and safety guidance for small-scale (not artisanal-) mines (Metcalf, 2013). In 2013, the German Geological Survey (BGR) provided a short course to newly recruited government inspectors and in 2013, Mineral Supply Africa (MSA) provided a 2-day safety course for mine operators supplying mineral concentrate to the company.

In Rwanda, the Ministry of Natural Resources (MINIRENA), through the Geology and Mines Department (GMD) of the Rwanda Natural Resources Authority (RNRA), is mandated to carry out mining inspection in all individual, mining companies and cooperatives operating in the Rwandan mining industry (No.002/2012/MINIRENA of 28/03/2012). Regular inspection of operating mines is aimed at enforcing the already established national mine site inspection procedures, guidelines and standards, as well as ascertain compliance with environment, and safety and health regulations. The inspections ensure that mining activities are carried out with the appropriate techniques and equipment for the sustainable exploitation of mineral resources and to ensure that the mine sites have efficient water and waste management system to control pollution. During this process, inspectors must check that all mines sites have safe and secure working conditions.

Despite these efforts, the local knowledge base and the availability of trained mining engineers is insufficient to address the safety challenges found on the artisanal mining operations.

2. Mine Inspection Form

In Rwanda a mine inspection form is used to perform annual mine site inspections (Republic of Rwanda, 2012). As part of the inspection, the mine inspector is to examine the mine’s working conditions in order to confirm that mine operator (owner) is complying with Rwanda’s occupational health and safety requirements. A review of the Rwandan Mine Inspection Form reveals a very superficial safety checklist. The following key elements pertinent to health and safety are covered in the inspection list:

- Written policy for health and safety
- Occupational health and safety

Compliance in terms of occupational health and safety, as described in the mine inspection form is as follows:

4 = Occupational injuries and work-related health problems are minimal and when they occur the mine site operator takes immediate measures to assist in treating them.

3 = Occupational injuries and work related health problems occur occasionally and when they occur the mine site operator takes measures to assist in treating them.

2 = Occupational injuries and work related health problems occur occasionally and the mine site operator is willing to help but does not have a clear procedure for treating them.

1 = Occupational injuries and work-related health problems are frequent and the mine site operator takes no responsibility in treating them.

0 = The mine site operator takes no responsibility for the health and safety of workers.

The mine inspection form also requires that the mine operator provide appropriate protective safety equipment and training to all workers and is ranked based on the following criteria:

4 = The mine site operator provides essential protective safety equipment to workers. A dedicated training staff offers training courses regularly on safety and health.

3 = The mine site operator provides most of the essential protective safety equipment for workers. Mine site operator personnel offer training courses regularly on safety and health as part of their on-going responsibilities.
2 = The mine site operator provides very little protective safety equipment for workers. Mine site operator personnel offer training courses Occasionally on safety and health.

1 = The mine site operator does not provide any protective safety equipment for workers. Maintaining safe practices is the informal responsibility of foremen or other on-site operators and the mine site operator makes no special provisions to promote safe practices.

0 = The mine site operator obliges artisanal groups to work on their own and resists any intervention. The mine site operator has no intention of organizing training for workers.

As a guide, the mine inspector form includes a checklist for inspectors to use as a means for on-site verification. Figure 1 depicts the checklist used for personal protective equipment (PPE) and Figure 2 provides the checklist used for mine safety risk and hazards.

As can be seen from Figure 1 the PPE requirements are basic and reflect a mining culture of non-compliance as most of the PPE is seen as a luxury. Most artisanal operations do not comply with the above requirements and often PPE is viewed a further risk. For example, safety boots are viewed as a hindrance when working on slopes in deposits containing schist and mica.

The safety risk and hazard form is also very basic and fails to address many of the real risks and hazards associated with mining activities. Section 3 reports the finding of a compliance assessment conducted on a mining operations, highlighting a number of concerns that are not assessable utilizing an abbreviated inspection form as depicted in Figure 2.

The Mine Inspection Form sets out a minimum standard for health and safety, however the actual questions around mining itself are lacking. This would tend to suggest insufficient technical knowledge from the government agency issuing the checklist. As a means to demonstrate the gap between the Mine Inspection Form and a typical compliance assessment, the author, experienced mining engineers, conducted a compliance assessment on the G1 incline shaft and gallery. The compliance assessment was conducted by identifying all dangers or deficiencies present in the incline shaft. The findings or risks of the assessment were classified into low, medium or high risks based on the likelihood and the impact of the risk, the consequence of the substandard, and the proposed mitigatory actions.
3. Findings of Compliance Assessment
The following sections provide a brief description of the noncompliance found while conducting the compliance assessment.

3.1. Access Control
Access control is a fundamental part of every operating mine, as it is necessary to ascertain that every employee can be accounted for after every shift and prior to any blasting operations. The Gahengeri G1 incline shaft has no register or token system in place that can be used to identify whether persons are still underground before blasting commences. A recommendation was brought forward that a board be put at the incline entrance with the workers names. Green and red tokens would be used to identify whether a worker was still working underground or working on surface. The red tokens would be designated to show that a certain worker is still underground. An alternative to this method is the use of a register whereby workers sign in and out of the incline shaft. As there represents a real chance of blasting prior to all workers leaving the mining area a medium risk has been estimated for access control. A gate is used to lock the incline at the end of the shift; hence adequate measures have been taken to prevent unauthorized access to the gallery after hours but precautions are inadequate prior to blasting.

3.2. Treatment of Misfires
When a charged production drill hole fails to explode after a blast the misfire needs to be treated, as one cannot drill the face until all blasted drill holes are clear of explosives. The mine has no proper method to handle misfires i.e. a scraper wire isn’t available to remove tamping and cartridge explosives or a two-way (air and water) blowpipe to clean out the exploded drill holes (sockets). Currently, misfires are treated by extracting the explosives from the blast hole utilizing a hammering and chisel; i.e. to chisel the rock away from the blast hole to enable the explosive cartridge to be removed. This is an extremely dangerous practice because if the blasting cap is struck by the chisel the blasting cap and explosive could be initiated and resulting in an explosion, which could harm or fatally injure persons working in that area. The hazard associated with misfires needs to be communicated to all minersworkers and proper equipment made available to handle misfires. Further, the inspection form should include the treatment of misfires and face preparation prior to the commencement of drilling.

3.3. Secondary Access and Development to B1 Stope
A secondary means of egress must exist for any operating mine. Although there exists a holling from the V4 vein it is questionable if all person would be able to make use of this access noting that any escape way should be at least 1.8m high for the ease of travel. The mine is currently developing toward the B1 working areas to improve ventilation and create a secondary access way. Approximately 25m of development remains before this holling will take place. Until this holling takes place the mine is contravention of Rwanda’s occupational health and safety requirements.

In addition to the concerns raised by the mine only having a single access way, the development of the B1 access way also represents a risk to workers mining directly above the access way as blasting of the access way could initiate falls of rock. Thus, before blasting takes place either in the workings above the B1 access way or the B1 access raise itself proper communication must be adhered by the two work area supervisors to ensure no workers are in the vicinity of the blast.

3.4. Top Blocks and Braking Devices
The risk of a run away wagon down the incline shaft is a concern, as wagons are parked in close proximity of the incline shaft entrance without proper stopping mechanisms to prevent run-aways. A stop block should be placed at the top of the incline to prevent run away wagons from accidently entering the incline shaft. Similarly, T-sprags should be placed at the bottom of the decline to stop or derail wagons if they should somehow be uncoupled from the winder rope. Where wagons are stopped and loading commences a braking device should be used to prevent the wagons from “running away”.

3.5. Electrical Wiring and Fire Prevention
Having electrical cables adjacent to wood is poor practice and represents a fire risk. Although much of the timber is damp due to moisture, not all timber is moist and ignition of the timber sets could cause catastrophic consequences, as a fire could prevent workers from escaping through the single access way. The electric cable installation should be rectified with electric cables placed six inches from timber or other means taken to remove the cable from the timber.

3.6. Ventilation
The incline shaft is ventilated using a natural ventilation systems with fresh air being sourced from the V4 work area located 130m from the end of the incline. As the development ends advance further into the deposit the environmental conditions deteriorate and the risk of gassing increases. Best practice dictates that development ends should be ventilated once they advance beyond 10m of through ventilation. A ventilation fan and ventilation ducting should be sourced and installed as a matter of priority.

3.7. Unbarricaded Opening
A number of disused vertical shafts remain un-barricaded, posing a risk for workers to inadvertently fall into the disused shafts. To rectify this risk the mine should fill in or barricade off all old vertical shafts.

3.8. Making Safe
Large rocks are lodged above the V9 breakaway position in the 1400 level haulage and should be made safe to ensure no loose rocks fall into the haulage. This area should be monitored daily and workers not allowed to remain in the V9 area. Depending upon conditions it may become necessary to bring down the entire hang-up but if the key block is brought down the rest of the rocks behind it will come down as well.
3.9. Personal Protective Equipment
The personal protective equipment (PPE) used by mineworkers normally consists of an overall with no reflective vests, plastic boots and a hard hat. It was noted that contract workers are not supplied with boots. Earplugs are available but are seldom used. Dust masks are sometimes issued but are ineffective. The use of safety gloves and eye goggles are not used at all. It is recommended that gloves and goggles be issued to workers who need the protection the most. For example, every drill operator should be supplied with gloves because the hydraulic oil makes the machine slippery and he needs goggles to prevent fly rock from entering into the operator’s eyes.

3.10. First Aid Equipment
First aid equipment must be made available at all working sites. As a minimum, a stretcher and equipped first aid boxes should be kept at the shaft surface office, demarcated by signage and checked on a weekly basis. Key personnel should also be trained in first aid, which is refreshed on an annual basis with accreditation every three years. Currently, the first aid boxes are not properly equipped and many items, such as bandages, cotton wool, antiseptic and plasters not available. No means of reporting incidents or accidents is in place. The mine should introduce a reporting system as a means to actively manage health and safety, as well as prevent future accidents.

3.11. Rehabilitation of the Timber Sets
In a number of areas of the incline shaft the timber found in the sets is old, cracked, or beginning to rot. In several areas the timber sets actually provide no support and therefore should be removed so that the hanging wall can be sounded during safety inspections.

3.11. Rehabilitation of the Tracks
Track work, i.e. installation and design, is critical to the efficient hauling of blasted rock from the development end to the tipping point on surface. Poor tracks not only equate to poor cleaning times but also represent a hazard to the tramming crew, as derailments can sometimes cause injuries and even fatalities.

4. Comparison of Mine Inspection Form to Compliance Audit
In order for an operation to continue mining safely certain risks and hazards must be identified and rectified. The Mine Inspection Form is one method used to determine whether operating mines comply with the Rwandan standards.

The mine compliance assessment exposed a number of poor practices, which are emphasized in the Mine Inspection Form, and that were not adhered to by the mine. For example, PPE compliance, personnel movement control, timber support, mine access (two exits), blasting safety and rock fall safety were noncompliant areas. In other situations, the Mine Inspection Form fails to adequately address safety areas. For example, the area of material handling, such as track conditions and prevention of run-a-way wagons, is not included in the inspection form. Other key gaps are the barricading off of old working areas - especially vertical shafts, fire prevention and the control and distribution of explosives i.e. the treatment of misfires or handling of old explosives. Other issues include travelling in the incline shaft at the same time as rock hoisting, examination procedures - including barring and testing of hanging wall and side wall conditions, and emergency readiness.

The difficulty with using generic checklists, such as the Mine Inspectors Form, is that the checklist leaves much to the interpretation of the user. This is understandable, as every mine site is different and has its own peculiarities and requirements. Underground safety consists of many issues that cannot be possible covered by a generic checklist and therefore the inspection form is reliant on the competency of the mine operator/owner to adequately address all hazards and risks associated with underground mining. Critical hazards will always be gases and ventilation, hanging wall and side wall support, mining spans, prevention of FOCS, access control, safe egress in and out of the mine, adequate lighting, PPE, fire prevention, blasting controls and water management.

5. Conclusions and Recommendations
This paper focuses on artisanal and small scale mining in Africa. The aim of the paper is to demonstrate the level of mine, health and safety practice in Africa and for young countries such as Rwanda that are sophomoric in their approach to regulate mining operations. Between 2015 and 2016 over 86 miners lost their lives to mine accidents.

Simple and safe mining practices are often disregarded in artisanal mines. The compliance assessment exposed a number of unsafe systems and activities at the G1 incline shaft and gallery. One of the main finding is that medium risks are more prevalent at the operation than any other type of risk. These risks can lead to serious injuries or fatalities, therefore substandards should be prioritized and an action plan drawn up with realistic due dates.

Hazard identification and risk assessments skills are critical attributes that mine operators, supervisors, and owners must have in order to comply with the standards set for occupational health and safety of mine workers. Based on the compliance assessment it is apparent that these skills aren’t readily available in Rwanda. Further, review of the annual Mine Inspector Form would indicate that government mine inspector also lack the skills to identify many of the risk associated with mining.

It is recommended that governmental departments address this shortcoming in terms of identifying and evaluating hazards. Governmental departments should also embark on training to improve the inspectors experience and ability to identify deficiencies in operating mines occupational health and safety programs. There is a real need to improve the general skills in
terms of hazard identification and risk assessment to improve occupational safety and health conditions. By improving the basic knowledge and skills of the mining process, mine inspectors, mine owners, supervisors and mine operators should be able to improve in the identification of risks and hazards, as well as implement actions to mitigate the identified risks and hazards.

Acknowledgement
The author would like to acknowledge and thank the following University of Johannesburg Bachelor of Technology (BTECH) students: T.A. Sethu, T.P. Letsebe, and L.N. Magwaza for their assistance in the compliance assessment of the G1 Incline Shaft and Gallery

References

The influence of natural aging processes on the strength parameters of steel-cord conveyor belts

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The article presents an analysis of the influence of the natural aging time on the strength parameters of steel-cord conveyor belts. The investigated phenomena include the influence of temperatures both above and below 0°C, as well as exposure to sun, rain and snow. The tests were performed on conveyor belts which have been operated in the Belchatów lignite mine for 30 years, transporting ROM material. The tests included adhesion of the steel cords to the belt’s core rubber and the tensile strength of both the carry and the pulley covers. The obtained results allowed determining the number of years after which belts operated in mines lose their properties. The results also indicate how long a conveyor belt, after launching it into service, retains such strength properties. The results also indicate how long a conveyor belt, after launching it into service, retains such strength parameters that enable its further usage after revulcanization. This information is of significant importance for the splicing and regeneration processes.

Keywords: conveyor belts, mine transport, research.

1. Background

The function of a belt conveyor is clear – it must carry the excavated overburden or coal to another location – to the dumping ground or the power plant. Its construction is simple and so is its principle of operation. However, since exploitation started in the Belchatów mine, generations of engineers, designers and researchers have coped with a number of unsolved issues related to the operation of belt conveyors. The belt conveyors designed and constructed mostly in the 1970s and 80s were technically sophisticated by the then standards. Yet, the currently operated conveyors are significantly different than the ones used 30 years ago. They are equipped with modern control and diagnostic systems, optical fiber links and central control systems. These solutions allow operating belt conveyors without constant human supervision [3].

Presently, increased attention is paid to lowering the operating costs of belt conveyors, mainly by lowering their energy consumption [6, 10]. This process is becoming crucial due to the rapidly growing prices of electric energy and ecological awareness. The transportation of the ROM material in the Belchatów mine, using belt conveyors with a total length exceeding 120 km, accounts for approx. 50% of the mine’s electricity consumption. This fact shows clearly how important a role belt conveyor transportation plays in terms of economy and ecology (significant CO2 emissions). This observation, as well as the urge to increase belt conveyor reliability, served as a departure point for much research undertaken by mining companies in cooperation with Wrocław University of Science and Technology [4, 9]. The key factor behind the energy consumption of belt conveyors is main resistance coefficient $f$, also referred to as the artificial or fictive friction coefficient. The lowering of this coefficient has become one of the priorities in research, and entailed the necessity to carefully examine its components. Lowering artificial friction coefficient is the aim of a number of currently performed research studies, which focus inter alia on the rotational resistances of idlers under load, on the resistance to motion of a single idler set in industrial conditions, or on the selection of idler sets [4, 9].

Significant cost of a conveyor belt, which may constitute as much as 50% of the conveyor’s value, necessitates its careful selection. One of the priorities in research on conveyor belts is to determine their life span, i.e. the time span over which they work failure-free and retain their physical and mechanical properties. The span guaranteed by the manufacturers is normally between 3 and 6 years. Extending this span to more than ten years is possible, on condition that the belts are properly maintained, serviced and inspected, and that the splices have proper tensile strength. Frequent visual inspection of belt condition is impractical, as the length of belt conveyors is increasing while some types of damage are difficult to recognize. In response to user demand, magnetic methods are becoming increasingly popular. These methods allow continuous control and diagnostics of the phenomena which occur in belt core [1, 2, 11].

Steel-cord belts have been regenerated in the Belchatów mine since 1981 in order to limit the import of belts or of the components needed for belt manufacturing. By using the same rubber core of a steel-cord belt twice or even three times, mining companies achieved significant cost reductions. Despite price reductions, regenerating belts is still motivated economically. Regenerated belts are commonly believed to have similar life span to new conveyor belts. However, research performed in the Laboratorium Transportu

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Continuous operation of a belt conveyor, consisting mostly in the transportation of the ROM material, causes extensive belt damage [5, 7]. The damaged belt section is typically replaced with a new belt section of identical core structure and nominal strength. Various belt sections are typically stored in mines. These belt sections have been removed from conveyors, e.g. when a conveyor was shortened, and can be still used to replace the damaged sections. Some of these sections are above 30 years of age and have been in continuous operation. Therefore, the key question is whether such sections can be still used. In order to answer this question, one must investigate how the time and the process of natural aging, covering the annual range of air temperatures, as well as precipitation and weather conditions, influence the strength parameters of conveyor belts. To this end, tests of adhesion between steel cord and rubber core were performed [14]. The tests also involved investigations into how the strength parameters of belt covers change over time.

### 2. The tested object

The object of the tests comprised four ST 3150 14+7 steel-cord conveyor belts with nominal tensile strength of 3150 kN/m. Between 1987 and 2017, these belts were – and in fact some of these belts have still been – used to transport the ROM material in the Belchatów mine. All of the belts have been manufactured by the same producer. The belts used in the tests have the following symbols:

1. The 6545/16 belt – the belt was manufactured in 2016; it is new, never operated. The structures of both the carrying cover and the pulley cover have a textile reinforcement in the form of strings 2 mm in diameter and at a 9 mm pitch.

2. The 3020/07 belt – the belt was manufactured in 2007; it has been operated since 2007 on an overburden conveyor. Its carrying side shows extensive damage (multiple cracks and cuts), while the pulley side remains undamaged.

3. The 1024/98 belt – the belt was manufactured in 1998; it was installed on a conveyor in the same year. However, a section of this belt, which is the object of this research, has been in storage for 19 years. Neither the carrying side nor the pulley side of the belt show traces of use or damage.

4. The 2475/87 belt – the belt was manufactured and installed on a conveyor in 1987. The belt was operated for 5 years and than stored for the following 25 years. Its carrying side shows some minor cuts and the pulley side remains undamaged.

### 3. Experimental part

#### 3.1. Tests of adhesion between steel cord and rubber core

Tests of adhesion between steel cord and rubber core were performed in accordance with PN-EN ISO 7623 [12]. The tests were performed on belts which were not subjected to thermal aging, as well as on belts subjected to one and two thermal aging processes. The thermal aging procedure consisted in placing the samples for 150 minutes between two plates heated to 145±5 °C. Each of the belts was represented by 9 samples subsequently divided into 3 groups: not aged, aged once, and aged twice. The non-aged samples were designated as A1,B1,C1; the samples aged once were designated as A2,B2,C2; and the samples aged twice were designated as A3,B3,C3. The samples were taken from locations showed in Fig. 1.

**Fig. 1. Sample collection locations**

Table 2 shows the summary of test results.
Table 2. Test results for the adhesion between belt cords and the rubber core

<table>
<thead>
<tr>
<th>Belt No.</th>
<th>Sample No.</th>
<th>Steel cord pull-out strength N/mm</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>no aging</td>
</tr>
<tr>
<td>1</td>
<td>A</td>
<td>211.0</td>
</tr>
<tr>
<td></td>
<td>B</td>
<td>212.2</td>
</tr>
<tr>
<td></td>
<td>C</td>
<td>209.0</td>
</tr>
<tr>
<td></td>
<td>mean</td>
<td>211</td>
</tr>
<tr>
<td>2</td>
<td>A</td>
<td>203.4</td>
</tr>
<tr>
<td></td>
<td>B</td>
<td>197.0</td>
</tr>
<tr>
<td></td>
<td>C</td>
<td>204.5</td>
</tr>
<tr>
<td></td>
<td>mean</td>
<td>202</td>
</tr>
<tr>
<td>3</td>
<td>A</td>
<td>143.0</td>
</tr>
<tr>
<td></td>
<td>B</td>
<td>161.7</td>
</tr>
<tr>
<td></td>
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<td></td>
<td>mean</td>
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<td></td>
<td>B</td>
<td>113.3</td>
</tr>
<tr>
<td></td>
<td>C</td>
<td>121.9</td>
</tr>
<tr>
<td></td>
<td>mean</td>
<td>119</td>
</tr>
</tbody>
</table>

3.3. Result analysis

The tests of adhesion between steel cords and the rubber core were performed on belt samples prior to thermal aging, as well as on samples subjected to one and two thermal aging processes. The results are presented in Table 2. Based on Table 2, diagrams have been plotted (Fig. 2), which illustrate the relationship between the adhesive strength of steel cords to the rubber core and the belt operating time.

![Fig. 2](image_url)

The tests of adhesion between steel cords and the rubber core were performed on belt samples prior to thermal aging, as well as on samples subjected to one and two thermal aging processes. The results are presented in Table 2. Based on Table 2, diagrams have been plotted (Fig. 2), which illustrate the relationship between the adhesive strength of steel cords to the rubber core and the belt operating time.

The analysis of the diagrams shown in Fig. 2 allowed an observation that in each of the three scenarios (prior to thermal aging, after first aging and after second aging) the adhesion between the cords and the rubber core decreases as belt operating time increases. In the ST 3150 belts operated in open pit lignite mines, the adhesion between the steel cords and the rubber core layer is expected to exceed 126 N/mm. Prior to thermal aging, the tested belts exceeded this adhesion value except for the belt operated for 30 years, whose adhesive strength was measured at 119 N/mm.

Neither first nor second thermal aging process influenced adhesion values in the new belt (manufactured in 2016). As compared to the values measured prior to aging, this belt showed adhesion reduced by only 3.3%.

After first aging process, adhesion value in the belt operated for 10 years (manufactured in 2007) decreased by 8% as compared to the values measured prior to aging. After second aging process, the belt’s adhesion decreased by a further 9%. The adhesion decrease for this belt (after two aging processes) totaled at 17% as compared to the values prior to aging.

Adhesion values observed after first aging process for the belt operated for 19 years (manufactured in 1998) decreased by 32% as compared to the values prior to aging. No further decrease was observed after second aging process.

The belt which had been operated in the mine for the longest period of time (30 years) showed 31% decrease in adhesion values after first aging. After second aging, the adhesion level remained at the same level.

The analysis of the influence that the belt’s operating time has on the adhesion between the cords and the rubber core leads to a conclusion that after 19 years in operation, the decrease in adhesion values, as compared to new belt, is stabilized at approx. 32%.

Table 3. Results of tensile strength and elongation tests for carrying and pulley covers

<table>
<thead>
<tr>
<th>Belt No.</th>
<th>Production year</th>
<th>Belt operating time, years</th>
<th>Steel cord pull-out strength N/mm</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>no aging</td>
</tr>
<tr>
<td>1</td>
<td>2016</td>
<td>0</td>
<td>25.5/24.9</td>
</tr>
<tr>
<td>2</td>
<td>2007</td>
<td>10</td>
<td>25.3/23.6</td>
</tr>
<tr>
<td>3</td>
<td>1998</td>
<td>19</td>
<td>24.5/22.1</td>
</tr>
<tr>
<td>4</td>
<td>1987</td>
<td>30</td>
<td>22.0/20.6</td>
</tr>
</tbody>
</table>

3.2. Tensile strength tests of carrying and pulley covers

Tensile strength tests of carrying and pulley covers were performed in compliance with PN-ISO 37 [13]. Tension and elongation tests for both the carrying cover and the pulley cover were performed on belts prior to thermal aging and after the first aging process. Thermal aging of rubber samples consisted in placing them for 7 days in a thermal chamber heated to 70°C. Table 3 shows the test results for the tensile strength and the elongation of both the carrying cover and the pulley cover.

![Table 3](image_url)
Table 3 served as a basis to plot diagrams illustrating the decrease in tensile strength for rubber covers, both before and after aging processes, in relation to belt operating time (Fig. 3). In accordance with the technical specifications required by the mine, the newly purchased belts had carrying cover tensile strength at a minimum of 24 MPa, and pulley cover tensile strength at a minimum of 20 MPa.

Prior to aging, the tensile strength for the carrying covers of the belts manufactured in 2016, 2007 and 1998 is still in accordance with the mine’s technical requirements, while the belt manufactured in 1987 no longer complies to these requirements. After first aging, the tensile strength values for the carrying covers of the new belt and the 2007 belt decreased, but remained above the required minimum. The two other belts had tensile strength below the minimum values required by the mine. The strengths of the pulley covers, both prior and after aging, meet the mine’s technical requirements. The only exception is the 1987 belt, which showed tensile strength of 19 MPa after aging.

The strength of the carrying cover subjected to natural aging for 10 years was not lowered, while over the next 9 years this value decreased by 3%. On the contrary, natural aging processes over a period of 30 years caused a decrease in the tensile strength of the carrying cover by 14% as compared to the values observed for the new belt.

Figure 4 shows the decrease in the elongation of both the carrying and the pulley covers in relation to belt operating time. In accordance with the mine’s technical requirements, the covers in the newly purchased belts had a minimal elongation at tear of 460%.

The analysis of the diagrams showed in Fig. 4 allowed a conclusion that elongation values for the rubber covers of the investigated belts decrease both before and after thermal aging, in relation to belt operating time. Still, all of the rubber cover elongation values, both before and after thermal aging, remain above the minimum required by the mine.

After thermal aging, the carrying cover in the new belt had elongation value reduced by 4.4% as compared to the values obtained prior to the aging process. As compared to the values obtained prior to the aging process, the thermally aged carrying covers in the 2007, 1998 and 1987 belts had elongation values reduced by approx. 5.1%, 3%, and 6%, respectively.

4. Conclusions

Laboratory tests of adhesion between the cords and the rubber core and the strength parameters of belt covers allowed a conclusion that with time the values of these parameters decrease. This phenomenon is greatly influenced by atmospheric conditions in which the belts have been operated on the conveyor, as well as by the storage methods. The tests show that:

- after 30 years of operation on the conveyor, the value of adhesion between the steel cords and the rubber core decreases by approx. 1/3 of the initial value,
- belts installed 30 years ago may be still operated on condition that the length of the conveyor was neither reduced nor increased. If the conveyor’s length is changed, the belt will need new splices, which will not reach the required tensile strength. Therefore, achieving required splice strength necessitates using belts which were operated for a period not longer than 16 years (Fig. 2),
- after 27 years in operation, belts display the adhesion between the cords and the rubber core below the minimum required by the mine. Further operation of the belt poses a risk of it breaking. In order to prevent that from happening, the belt’s condition should be continuously monitored,
- regeneration is possible for belts which have operated not longer than for approx. 16 years. However, care should be taken to prevent the belt core from fatigue-related degradation, damage, cuts of both the cords and rubber, as well as from corrosion. Otherwise the belt may be disqualified from regeneration even if its age did not exceed 16 years,
- belts which have been in operation for more than 16 years, when disassembled from the conveyor, should be utilized, as the research results indicate that they cannot be reused,
- after 30 years, the decrease in the strength of both the carrying cover and the pulley cover is 14% and 17%, and in the elongation at tear, it is 22% and 18%, respectively. Over a 30-year period, the changes in both the strength and the elongation of belt covers do not result in reaching any disqualifying levels.

Although the above conclusions have been based on the results of tests performed on four conveyor belts, they remain in connection to actual conveyor belt operating conditions. Therefore, the results of research into the adhesion between the steel cords and the rubber core of the belts operated in actual conditions introduce novel information about the nature of these adhesion changes.

The analysis of the above presented test results also reveals how important it is for conveyor belt manufacturers to leave a “margin” of adhesion between the steel cords and the
rubber core. The tested belts were dedicated to open pit mining applications. Such belts are especially prone to various weather conditions, which increase their aging processes. Belts manufactured with parameters higher than the required minimum offer a guarantee that even despite adverse weather conditions, the adhesion between the cords and the core will remain above the expected minimum, thus extending the belt’s life span.

Acknowledgement
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Augmented Reality Based Approach for Interpretation and Visualization of Methane Flow Behavior

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Abstract— In this paper the possibilities of alternative approach to the visual presentation of computer fluid dynamics modelling results are discussed. Related work and physical background are mentioned first, followed by the experimental results. The paper is focused mainly on the interpretation of the data obtained from the models simulation process with the help of Augmented Reality. The experimental deployment offers promising results and presents the way for further development in this field.

Keywords: Augmented Reality, CFD, visualization, methane flow, modeling and simulation.

1. Introduction

In the beginning, the Computational Fluid Dynamics (CFD) was only known in the high-tech engineering areas such as aeronautics and astronautics, but now, with the rapid advancement of computers and information technologies, it is becoming a rapidly adopted methodology for solving complex problems and CFD is also finding its way into important uncharted areas, especially in chemical, civil and environmental engineering. Among the benefits of CFD belong detailed visualization and comprehensive information about studied phenomenon.

Many authors already began to realize the strength and benefits of using CFD tools, and so many applications can be found in the field of mining engineering. Among those, Xu et al. [1] could be mentioned. Their paper provides a review of computational fluid dynamics applications in mining engineering, with particular focus on mine ventilation-related flow problems. CFD applications in mining engineering research and design areas are reviewed, which illustrate the success of CFD and highlight challenging issues. According Xu et al., it is expected that more CFD research has to be carried out to solve problems in mining engineering. The potential benefits from the simulations are enormous if proper modelling procedures are followed and modern computational approaches are implemented.

The CFD tool may not only be used to solve problems associated with ventilation and gas flow. Yin et al. [2] presented a study where a computational fluid dynamics was conducted to investigate the effect of several important factors on the dust control performance of a down-the-hole air hammer drill bit for reverse circulation drilling in mines. A series of experimental tests in the laboratory were carried out to validate the results of CFD modelling. It was shown that the CFD simulation results were in good match with experimental data. Field test results indicate that the modified drill bit could control drilling dust more effectively. No cuttings escaped from the borehole when it was used for downward drilling.

Visualization of results is an equally important part of the whole work of modelling and simulation. Presentation of the results is particularly difficult for the 3D tasks. Sometimes, it is difficult to find the right look to the examined phenomenon so it is not easy to understand and interpret how model behaves. For these purposes and needs, there are other products and tools such as ANSYS CFD-post or the possibility to export results of modelling and simulation to the environment of Virtual and Augmented Reality.

Figure 1. The evolution of visualization methodology. Development of visualization methods originates in the 17th century and it is rapidly improving today due to technical progress

Efficient visualization tools should respect cognitive and perceptual properties of the human brain. Visualization aims to improve the clarity and aesthetic appeal of the displayed...
information and allows a person to understand large amount of data and interact with it in more natural way. Visualization methods have evolved much over the last decades (see Figure 1) and the only limit for novel techniques is human imagination. To anticipate the next steps of data visualization development, it is necessary to take into account the successes of the past. It is considered that quantitative data visualization appeared in the field of statistics and analytics quite recently [3].

2. Related work

The modelling process does not end by obtaining output data from solvers in the field of CFD numerical simulation. To be able to leverage the benefits of the solution, it is needed a sophisticated tool that will give us a complex view of simulation results.

There are many various approaches how to visualize obtained results in the form of graphics or animation.

One of the tools that can be used for this purposes is ANSYS CFD-Post. It is a universal tool for processing outputs from the CFD solver that provides everything needed for visualization and analysis of results. It gives to users a way to get detailed information about the investigated phenomenon in the form of data, graphs, images and animations.

Another option how to visualize and interpret results obtained from CFD analysis is to use Virtual Reality (VR). VR allows user to interact with the simulated environment. It creates the illusion of the real world and that’s why this technique is well suitable for training purposes. Cha et al. in theirs study [4] propose a series of data conversion techniques and a real-time processing framework to develop a fire training simulator on the basis of a precise CFD simulation that is capable of calculating various invisible physical quantities such as concentration of toxic gases and heat distribution as well as visible factors such as smoke and flame.

An immersive visualization environment (IVE) with 3D stereoscopic capability can mitigate some shortcomings of 2D displays via improved depth cues and active movement to further appreciate the spatial localization of imaging data with temporal computational fluid dynamics (CFD) results. Quam et al. [5] presented a semi-automatic workflow for the import, processing, rendering, and stereoscopic visualization of high resolution, patient-specific imaging data, and CFD results in an IVE.

The paper [6] summarizes the developments and applications of blast furnace CFD models and the virtual blast furnace. The multiple computational fluid dynamics (CFD) models have been developed to simulate the complex multiphase reacting flow in the three regions of the furnace, the shaft, the raceway, and the hearth. The models have been used effectively to troubleshoot and optimize blast furnace operations. In addition, the CFD models have been integrated with Virtual Reality. Described interactive virtual blast furnace has been developed for training purpose.

Malkawi et al. [7], [8] presented an interactive gesture recognition-based, immersive Augmented Reality (AR) system visualizing CFD datasets of indoor environments. CFD simulation is used to predict the indoor environments and assess their response to specific internal and external conditions. To enable efficient visualization of CFD datasets in actual-space, an Augmented Reality system was integrated with a CFD simulation engine. To facilitate efficient data manipulation of the simulated post-processed CFD data and to increase the user control of the immersive environment, a new intuitive method of Human-Computer Interaction (HCI) has been incorporated. A gesture recognition system was integrated with the Augmented Reality-CFD structure to transform hand postural data into a general description of hand-shape, through forward kinematics and computation of hand segment positions and their joint angles. This enabled real-time interactions between users and simulated CFD results in actual space.

Heuveline et al. [9] presented a visualization method for large-scale scientific computing illustrated by the example of urban air flow simulation. Results of numerical simulations will be available to decision-makers and citizens, raising the impact and improving the communication of scientific results. The visualization method is based on the accurate alignment of the viewer’s position and the orientation of his camera view with the three-dimensional city model and the numerical simulation. The case of urban wind flow simulations proves the benefits of mobile Augmented Reality visualizations, both in terms of selection of data relevant to the user and facilitated comprehensible access to simulation results. Their methods are backed by a client-server framework and offer business models for simulation and visualization on demand in a cloud-based setup.

![Figure 2. Example of flow visualization through Augmented Reality; [9]](image)

Moreland et al. [10] presented a technical approach for combining Augmented Reality with Computational Fluid Dynamics to develop materials for training on a large boiler of a coal-fired power plant.

Maptek mining company [11] is among the first companies using visualization via Augmented Reality during mining operations, such as support in real-time decision making process in the opencast mine. The ground surface is recorded using the I-Site 8810 laser scanner. The planned model of the daily work schedule is projected on the recorded scan. Whole system is paperless and results can be viewed on
smartphone, tablet or over organization network. The operator of mining machinery accurately observes the daily plan and has immediate control that he digs in a proper place.

Laser scanner then reveals that the wall has been excavated under a 35-degree angle while it should be 45-degree and may remove the error immediately. This leads to increased safety of highwalls through improved wall stability. Another benefit is increased productivity by adjusting excavation before severe mistakes are made which leads to increased mining machinery effectiveness. Early detection of errors in material movement enables also excavation cost savings. Laser survey data are linked to mine planning system and enable feedback on historical mining data.

This year the Leapfrog company which specializes in geological software for mining presented its new module for Augmented Reality in mining industry called Aspect [12]. Compared to PerfectDig this software is still in testing, but yields promising results. It allows users to visualize geological data and models in the real world, to deliver improved physical validation, collaboration and communication. The geologists viewing the model can use their knowledge of the deposit and the terrain properties to give a contextual visual check and collaborate in the development of the very best model solution. This also gives field geologists the opportunity to see what the resource geologist deems important, and provides a basis for further discussion.

Collectively, we can conclude that Augmented Reality and Virtual Reality slowly finds its way into the field of mining. The application potential of these emerging technologies is limited only by human imagination. The first application we discuss.

3. Technological and Methodological Background

3.1. Computational Fluid Dynamics

Fluid Dynamics is the study of fluids in motion. The basic equations governing fluid motion have been known for more than 150 years and are called the Navier-Stokes equations which govern the motion of a viscous, heat conducting fluid. These equations are based on the Euler’s equations of hydrodynamics and hydrostatics. In its essence, these equations describe motion of real fluids, reflect the balance of forces in the fluid flow. In addition to pressure, external and inertial forces the Navier-Stokes equations include and the actual friction force caused by fluid viscosity. Together with the continuity equation the Navier-Stokes equation form system of partial differential equations.

\[
F_u = F_{uo} + F_{e} + F_t \tag{1}
\]

\[
\frac{\partial u}{\partial t} + \frac{\partial (uu)}{\partial x} + \frac{\partial (uv)}{\partial y} + \frac{\partial (wu)}{\partial z} = -\frac{\partial p}{\partial x} + \nu \left( \frac{\partial^2 u}{\partial x^2} + \frac{\partial^2 u}{\partial y^2} + \frac{\partial^2 u}{\partial z^2} \right) + f_e \tag{2}
\]

\[
\frac{\partial u}{\partial t} + \frac{\partial (uu)}{\partial x} + \frac{\partial (uv)}{\partial y} + \frac{\partial (wu)}{\partial z} = -\frac{\partial p}{\partial y} + \nu \left( \frac{\partial^2 u}{\partial x^2} + \frac{\partial^2 u}{\partial y^2} + \frac{\partial^2 u}{\partial z^2} \right) + f_e \tag{3}
\]

\[
\frac{\partial u}{\partial t} + \frac{\partial (uu)}{\partial x} + \frac{\partial (uv)}{\partial y} + \frac{\partial (wu)}{\partial z} = -\frac{\partial p}{\partial z} + \nu \left( \frac{\partial^2 u}{\partial x^2} + \frac{\partial^2 u}{\partial y^2} + \frac{\partial^2 u}{\partial z^2} \right) + f_e \tag{4}
\]

These equations differ in last element on the right side from the Euler’s equations of hydrodynamics. This element is the force needed to overcome the viscous friction of the fluid. The differential operator is applied to three velocity coordinates:

\[
\Delta = \frac{\partial^2}{\partial x^2} + \frac{\partial^2}{\partial y^2} + \frac{\partial^2}{\partial z^2} \tag{5}
\]

CFD is a computer-based mathematical modeling tool that incorporates the solution of the fundamental equations of fluid flow.

The general procedure of modeling, see Figure 3, can be divided into three basic steps.

In any CFD analysis, the first step is definition and creation of geometry, i.e. computational domain, where the studied phenomenon takes place.

The second step is mesh creation. The term mesh or network and its significance in mathematical modeling tasks represent a division of the computational area which was developed under the previous step into successive component cells. Development of a quality network is a fundamental precondition to the mathematical modeling.

Appropriately defining the model’s boundary conditions is an integral part of the entire modeling and simulation process using the CFD Fluent software. The definition is of two kinds. At the first stage, the type of boundary conditions is defined, i.e. we define, e.g., “this” as a wall, “that” as a velocity input, “that” as a pressure input, etc. The process still takes place in the software environment where the model of the area and network was previously developed. All of this was done under the step called pre-processing.

Another stage of setting the boundary condition involves the situation where specific physical and chemical properties, specific values, are assigned to individual types of boundary conditions. That already takes place in the actual CFD environment, such as in Fluent and it could be said that this is a part of solving, if we do not understand the term solving in in a very limited form of calculation itself.

A great emphasis is put on the actual setting of the boundary conditions as an appropriate setting is the key stone of the entire computation. In principle, that means that the more accurate boundary conditions are set, the more accurate result may be expected.

The step called solving also requires the definition of fluid properties, solver properties, materials and many other variables depending on the defined physical flow and chemical processes.
In CFD analysis, the numerical solution of before mentioned governing equations can provide entire the fluid properties in space and time. This stunning amount of information must be displayed in a meaningful form. Thus flow visualization is equally important in computational as in experimental fluid dynamics.

3.2. Augmented Reality Approach

Augmented reality is generally characterized as a technology that adds graphical, audio and other virtual elements into perceived reality in real time. To be able to use the Augmented Reality, we always need at least two or three devices, namely a scanning camera, a computing unit and a video display unit. The ideal choice, for their compact size, are mobile devices (smartphones, tablets) which include not only the camera and display but also the necessary computing power or modern wearable electronics such as smart glasses. Many of those devices also benefit from additional sensors that can be utilized for the needs of Augmented Reality as an electronic compass, gyroscope, accelerometer, GPS (Global Positioning System) and wireless Internet access. Marker is an important element in the entire concept of Augmented Reality. Marker is a “symbol” that is recognized in the real scene and applications respond to this “symbol” in a certain way.

As can be seen on Figure 4, detection of the marker itself takes place in a way that starts with an arbitrary point \( x \) on the outline of a square. A point that has maximum distance from the point \( x \) is the corner point \( C_0 \). This creates a diagonal that runs through the center of the square. After creating of this diagonal, we find the corner points \( C_1 \) and \( C_2 \) which have the greatest distance to the right and left side of the diagonal.

Using the diagonal from point \( C_1 \) and \( C_2 \), we find the point \( C_3 \) that is the most distant point on the right side.

The detection takes place in a way that converts the raster image to a grayscale. Next operation is called thresholding. The thresholding is based on the evaluation of the brightness of each pixel. If the brightness of the pixel is below the threshold value, a black color is assigned to the pixel, if the brightness is above the threshold value, a white color is assigned. The threshold value is typically obtained from a histogram which often includes two maximums. One maximum for the background brightness and one for the foreground brightness. The optimal threshold is then located between these maximums. Edge detection is a process that automatically detects locations in the image with the most changing brightness values.

Marker generally consists of a square in which an additional identification mark is placed. The identification mark can have a simple geometric shape but it can also cope with more complex shapes. It is also possible to use a marker in the shape of a text. Each marker is stored in the application and there is a certain object assigned to it.

4. Case studies

4.1. Methane flow behavior at coalface area in underground coal mines

One of the interesting possibility of using CFD codes is simulation of vent gas from the coalface area. A very important activity in mining in underground mines is opening new mines that are not necessarily associated with separate ventilation. Separate ventilation system provides ventilation
of non-excavated mines. This allows us to dilute and divert dangerous mine gases, and so fulfill the strict safety limits imposed on permissible concentrations of hazardous mine gases. More detailed information about this study with evaluation is mentioned in our studies [14] and [15].

Subsequent modeling and simulation is carried out in the CFD program Fluent. Specifically, the turbulent model, standard k-ε, was used for the flow simulation.

Turbulent viscosity is as defined in [16]:

$$\mu_t = C_v \frac{k^2}{\varepsilon}$$  \hspace{1cm} (6)

The final form of the equations for turbulent energy and dissipation rate ε is defined as follows [16]:

$$\frac{\partial}{\partial t} \left( \frac{k}{\rho} \right) + \frac{\partial}{\partial x_j} \left( \rho u_j \frac{k}{\rho} \right) = \frac{\partial}{\partial x_j} \left( \nu_t \sigma_k \frac{\partial k}{\partial x_j} \right) + \nu_t \left( \frac{\partial u_j}{\partial x_j} + \frac{\partial u_l}{\partial x_l} \right) \frac{\partial k}{\partial x_j} - C_D \frac{k^3}{\varepsilon} \frac{2}{\varepsilon}$$  \hspace{1cm} (7)

$$\frac{\partial}{\partial t} \left( \frac{\varepsilon}{\rho} \right) + \frac{\partial}{\partial x_j} \left( \rho u_j \frac{\varepsilon}{\rho} \right) = \frac{\partial}{\partial x_j} \left( \nu_t \sigma_\varepsilon \frac{\partial \varepsilon}{\partial x_j} \right) + \nu_t \left( \frac{\partial u_j}{\partial x_j} + \frac{\partial u_l}{\partial x_l} \right) \frac{\partial \varepsilon}{\partial x_j} - C_D \frac{k^3}{\varepsilon} \frac{2}{\varepsilon}$$  \hspace{1cm} (8)

The model involves five empirical constants: $C_v$, $\sigma_k$, $\sigma_\varepsilon$, $C_{1\varepsilon}$, and $C_{2\varepsilon}$, which can be identified using simple cases of flow or by numerical optimization. According to Patel, Rodi, and Scheurerer, different variants of the k-ε model typically use the following values [17]:

- $C_v = 0.09$
- $\sigma_k = 1.0$
- $\sigma_\varepsilon = 1.3$
- $C_{1\varepsilon} = 1.44$
- $C_{2\varepsilon} = 1.92$

The following figure (Figure 6) shows a schematic diagram of the modeled area.

![Schematic diagram of the modeled area](image)

Figure 6. Schematic diagram of the modeled area with a blowing air duct

At the corridor inlet, Figure 6 - a), the methane-air mixture is flowing at the speed of 1 m·s$^{-1}$. The composition of the flowing mixture is as follows: $O_2 = 20.5\%$, $CO_2 = 1\%$, $CH_4 = 0.5\%$ a $N_2 = 78\%$.

The coal bed at the face area, Figure 6 - b), releases 5% concentration of methane at the speed of 0.5 m·s$^{-1}$.

This variant represents a situation where the auxiliary ventilation is secured with a blowing ventilation system.
5. Experimental Results

The whole process of visualizing results in Augmented Reality can be seen on Figure 9. Application starts scanning the scene and search for the marker in the scanned image so that it could respond to it by rendering the 3D object in a real scene which is assigned to the marker. Once the object is recognized, the database is queried to provide adequate data to that particular marker. The resulting image is then the combination of perceived reality and augmentation. Composite image is then sent to the display of a smartphone or smart head-mounted display the user is wearing.

Figure 9. Framework of results visualizing through AR

To verify Augmented Reality-Based Approach the described examples were chosen and placed them in the context of real environments.

To view the result, it was necessary to equip chimney with marker and assign an appropriate model to display.

Figure 10. Demonstration of AR model in the case of degassing chimney

On the Figure 10, the linking of the real world extended by measured data visualization can be seen. The displayed data may not only be static, but allows user interaction adjusting the wind speed and atmospheric pressure which changes model visualization.

Thanks to this experiment, it was verified that the data from CFD codes are usable and viewable in practice. For visualization it was used a smartphone and available software and hardware resources. The limiting factor in this case is the size of the marker, the quality of the capturing sensor in the camera and lighting conditions.

Figure 11. Demonstration of AR model in the case of ventilation of mine corridor

In a similar way, it was performed an example of the flow of methane in the mine corridor. Here was the greatest obstacle lighting conditions which led to the difficult detection of maker.

6. Conclusion

The article described the relatively new possibility of using Augmented Reality to visualize the results of CFD analysis in real environments. Models of methane flow have been developed in the ANSYS program based on measured data. Visualizations of these models were processed using software tools for Augmented Reality and then interpreted using a conventional smartphone.

The use of smart devices opens the way to intuitive access to and interaction with numerical simulations that are highly comprehensible due to the embedding into the real-life camera view as Augmented Reality visualizations.

Exclusivity of this approach is the effort to visualize and demonstrate how the gas behaves in the model examples that were tested. Without the linking of numerical flow technology, commonly referred to as CFD programs, and Augmented Reality we can practically never visualize and comprehend the situation completely. Thanks to the use of the Fluent software and visualization through Augmented Reality, we are theoretically able to simulate and subsequently learn how ambient influences may alter and act upon the flow. Thanks to this approach we can clearly imagine and interpret the obtained results which could be visible by other means.

In the future technological developments lead to overall integration with smart head-mount display. Even now we see the integration of these technologies thanks to the rising trend of Industry 4.0 and the Internet of Things. Encouraged by our
existing achievements in the field of visualization, we will continue in this direction in the future.

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Experimental Study on Compact Underground Coal Gasification System with A Horizontal Well

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Abstract — Underground coal gasification (UCG) is a technique to recover coal energy by the in-situ conversion of coal into gaseous products. We suggest a co-axial UCG system with compact and flexible to adopt under the complicated geological conditions because the geological conditions of coal seam are complicated as the existence of faults and folds in Japan. In this study, an application of co-axial UCG system with a horizontal well is discussed by means of the model UCG experiment with a large-scale simulated coal seam which the size is 550 × 600 × 2,740 mm. A horizontal well which had 45 mm diameter and 2,600 mm length is used as an injection/production well. During the experiment, the changes of temperature field and product gas compositions were observed by changing the position of an injection pipe. The average calorific value of product gas is 6.85 MJ/Nm$^3$ and the gasification efficiency which means the conversion efficiency of the gasified coal to the syngas shows 65.43% in the whole experimental process. The study results suggest that the recovered coal energy from a co-axial UCG system is comparable with that of a conventional UCG system. Therefore, a co-axial UCG system may be a feasible option to utilize the coal resources abandoned in underground without mining.

Keywords: Underground coal gasification, Horizontal well, Co-axial system, Laser ignition

1. Introduction

Underground coal gasification (UCG) is a technique to extract energy from coal in the form of heat energy and combustible gases through the chemical reactions in the underground gasifier. A conventional UCG systems is gasification of the coal in a gasification channel made by a well linking the injection and production wells \cite{1, 2} as shown Figure 1. This system is suitable for thin, flat, and deep coal seams. An alternative UCG system must therefore be developed in Japan because geological conditions are
complicated, with the existence of faults and inclined coal seams. Given this background, we are developing a co-axial UCG system that is compact, safe, and highly efficient as shown Figure 2. The co-axial UCG system uses only well drilling and a double pipe. Gasification agents are injected from the inner pipe to expand the combustion zone. The production gas is recovered from the outer pipe.

Until now, various UCG model experiments have been carried out to develop the co-axial UCG system [3-6]. However, the recovered energy from the coal is relatively low because the gasification area in a co-axial system is limited around a well [7, 8]. Therefore, an application of co-axial UCG system with a horizontal well is discussed in order to improve the total efficiency of gasification process in this study (Figure 3).

### Table 1 – Proximate and ultimate analyses of the coal

<table>
<thead>
<tr>
<th>Calorific value (MJ/kg)</th>
<th>Proximate analysis (wt%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Moisture</td>
</tr>
<tr>
<td>32.1</td>
<td>2.1</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Ultimate analysis (wt%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>C</td>
</tr>
<tr>
<td>78.4</td>
</tr>
</tbody>
</table>

Ignition is an important process to rise coal temperature to start UCG process since product gas can be generated due to the promotion of chemical reactions around the high temperature area. In this study, the laser ignition system with oxygen supply was adopted to ignite the coal with easy, safety, and quickly (Figure 5). We used semiconductor laser equipment (M710A45; Omron Laserfront Inc.) which laser emission wavelength was 808 nm and rated output was 45 W. The ignition process has succeeded by emitting the laser to the bottom of co-axial well from 150 mm away with oxygen supply.

After the ignition stage, a mixture of air and oxygen was injected continuously with 30~40 L/min while the oxygen concentration was kept as stable (50%) based on the previous experiments [8]. During the experiment, temperature, the flow rate, and the compositions of the product gas were measured. Temperature was monitored to visualize inner part of the coal seam by using type K thermocouples (SUS310S; Chino Corp.) and data logger (GL220; GRAPHTEC Corp.). Figure 6 shows the distributions of sensors. The flow rate of the product gas was measured using an ultrasonic flowmeter (DigitalFlow™ GM868). The compositions of product gas (O₂, N₂, CO₂, H₂, CO, CH₄, C₂H₆, C₃H₆, and C₃H₈) were monitored every hour using a gas chromatograph (Micro GC 3000A; Inficon Co. Ltd.).

Figure 3. Co-axial UCG system with a horizontal well

Figure 4. Diagram of UCG model experiment
Additionally, the position of an injection pipe was changed periodically every 300 mm toward the inlet of oxidant to move the gasification area when the gasification reactions were not active. The gasification period was 72 hours because this experiment was stopped 72 hours elapsed due to the trouble of the experiment equipment. After the process, a mixture of white cement and gypsum was filled into the post-gasification cavity to investigate a cross-section study of the combustion zone.

3. Results and Discussion

3.1. Temperature and cross-section study

Based on the temperature data, the two-dimensional maximum temperature profiles are plotted for several experimental periods in Figure 7, representing the maximum temperature distribution in a cross-section of a horizontal well. Each figure shows results in the different position of an injection pipe. From the results in Figure 7, the maximum temperature of simulated coal seam exceeds more than 1,200 degree. Considering the gasification reactions are promoted under the high temperature, it is assumed that the gasification process is activated around the high temperature area. Additionally, the high temperature area is moved when the injection position is changed, meaning that the gasification area is moved. This fact means that the gasification reactions are activated around an injection pipe. Gasification reactions are promoted under the high temperature as a result of oxidation reaction. Therefore, the gasification area is expanded around an injection pipe because the most of oxidant is consumed near the injection pipe.

From a different perspective, it is possible to control the gasification area with changing the position of the injection agents. Usually, molten slag formation during the gasification process due to the ash contents of the coal prevent the promotion of the gasification reaction because of the limitation of the gas-solid contact. Therefore, the control of gasification area can be useful to promote the gasification reaction continuously because it is possible to move the gasification area to the unreacted part of coal seam that has no slag formation.

Figure 8 presents an estimation of the post-gasification cavity based on a cross-section study every 100 mm. The cross-section consists of cavity, char, unreacted coal, and shale. Unfortunately, there are shale seam in the upper part of simulated coal seam because we obtained the simulated coal seam in an active open-cut coal mine in-situ. From the results of Figure 8, the shape of char reflected gasification area agree with those obtained from temperature profiles. Additionally, it can be said that wide range of simulated coal seam in the lower part is gasified. This fact confirms the effectiveness of changing the position of the injection agents to expand the gasification area to the horizontal direction. The gasification area is limited around the middle part of the coal seam due to the effect of refractory cement. This is the reason why the refractory cement is filled into the void of coal seam near the gasification channel.

3.2. Compositions of product gas and calorific value

Monitoring results of the main compositions and the calorific value of a product gas are presented in Figures 9. There are no results from 38~47 hours and 50~51 hours elapsed due to the trouble of monitoring system. The calorific value of the product gas can be calculated with the concentration of the combustible gas contents such as CO, H₂, CH₄, and other hydrocarbons [9].

During the initial period, the CH₄ increases continuously and causes the first peak of the calorific value (9.79 MJ/m³).
The production of combustible gases such as CO, H₂, and CH₄ are decreased slowly until injection pipe is moved. The composition of product gas are dramatically changed and the calorific value reaches to second peak (10.27 MJ/m³) when the position of injection pipe is moved at about 23 hours, meaning that the combustible contents such as CO, H₂, CH₄ are much increased and the CO₂ is much decreased. Considering CO₂ is a source of reduction reaction which is a main chemical reaction to produce the combustible contents in UCG process, it can be considered that the gasification process is activated again after the injection pipe position is moved to the unreacted part of coal seam. Latter stages also show the same tendency: the calorific value reaches to third peak (11.60 MJ/m³) at about 46 hours and fourth peak (9.73 MJ/m³) at 63 hours. This fact shows the possibility to control the quality of product gas by arranging the injection position of oxidants though the slag formation usually inhibits the gasification reaction.

![Gas concentration and calorific value of product gas](image)

Figure 9. Main compositions and the calorific value of product gas

### 3.3. Comparison with the results of previous study

In order to evaluate co-axial UCG system with a horizontal well, the comparison of results obtained from this study and previous study [8] are conducted. Three type of model experiments were carried out in previous study: co-axial 1 (injection ratio is 26 L/min and oxygen concentration is 58%), co-axial 2 (injection ratio is 20~50 L/min and oxygen concentration is 50%), and linking (injection ratio is 20~50 L/min and oxygen concentration is 58%) which is a conventional system. Experiments of co-axial 1 and 2 simulate the co-axial UCG system with a vertical well whose the length is 400 mm. As a parameter to compare each UCG experiment, we calculate the gasification efficiency, which means the conversion efficiency of the gasified coal (chemical energy of product gas/chemical energy of gasified coal), as shown equation (1).

\[ R_g = \frac{\frac{\dot{m}_g}{\dot{m}_c}}{\dot{Q}_c} \]  

(1)

where \( R_g \) is the gasification efficiency (%), \( E_g \) means the total energy of product gas (MJ), \( W_p \) represents the gasified coal (kg), and \( Q_c \) stands for the coal calorific value (MJ/kg).

Total energy of product gas is calculated by using the results of product gas composition and product gas flow rate. Additionally, the gasified coal is calculated based on balance computation of C element [10]. Table 2 presents the calculation results for gasification efficiency. In previous study, the values of gasification efficiency in co-axial 1 and 2, which simulate a co-axial UCG system, were 45.93% and 43.19%. By contrast, that in linking which simulate a conventional UCG system was 63.15%. From these results, it can be understood that a co-axial UCG system with a vertical well has lower efficiency for energy recovery from coal than that of a conventional UCG system because of the low quality of product gas: 4.68 MJ/Nm³ for co-axial 1 and 4.75 MJ/Nm³ for co-axial 2 while 7.78 MJ/Nm³ for linking. On the other hands, the results of the current study about a co-axial system shows that the gasification efficiency achieves 65.43% with high quality of product gas (6.85 MJ/Nm³), meaning that the value is dramatically improved and comparable with that of a conventional system. This finding suggests that the recovery energy from co-axial UCG system can be improved with the control of gasification area by changing the position of injection agents when the co-axial well can be prepared along coal seam dip.

<table>
<thead>
<tr>
<th></th>
<th>Co-axial 1</th>
<th>Co-axial 2</th>
<th>Linking</th>
<th>Co-axial with a horizontal well</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gasification period (hour)</td>
<td>24</td>
<td>51</td>
<td>111</td>
<td>72</td>
</tr>
<tr>
<td>Average calorific value (MJ/Nm³)</td>
<td>4.68</td>
<td>4.75</td>
<td>7.78</td>
<td>6.85</td>
</tr>
<tr>
<td>Total energy (MJ)</td>
<td>383.24</td>
<td>673.43</td>
<td>2804.82</td>
<td>1820.81</td>
</tr>
<tr>
<td>Gasified coal (kg)</td>
<td>25.91</td>
<td>48.54</td>
<td>138.28</td>
<td>86.63</td>
</tr>
<tr>
<td>Gasification efficiency (%)</td>
<td>45.93</td>
<td>43.19</td>
<td>63.15</td>
<td>65.43</td>
</tr>
</tbody>
</table>

### 4. Conclusions

This study focuses on the co-axial UCG system with a horizontal well that can be moved the injection position in order to improve the total efficiency of gasification process. Results show that it is possible to control gasification area by arranging the position of injection pipe; gasification reactions are activated around an injection pipe because the most of oxidant is consumed near the injection pipe. Additionally, the recovered coal energy from a co-axial UCG system with a horizontal well is comparable with that of a conventional UCG system in terms of gasification efficiency due to the high improvement of product gas quality. Therefore, a co-axial UCG system may be a feasible option to utilize the coal.
resources abandoned in underground by controlling the injection position and designing a co-axial well along coal seam dip.

Acknowledgement
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References
MPES2017

Session 9 – Underground Mining

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Mitigation of fugitive dust impact arising from BR dry disposal

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Abstract: The development of international policies about environmental defense has enforced some major modifications in the management of industrial tailings. As regards the disposal of the residue deriving from the bauxite processing (BR) in the alumina industry, national and international regulations have encouraged the evolution from wet to dry disposal techniques. However, such a change in the storage practices poses a major concern due to the potential increase of the atmospheric impact in the surrounding areas, due to the emissions of Particulate Matter (PM) generated both by the BR disposal activities and by the wind erosion over the dried surfaces of the landfill. The article analyses the effect of the impact mitigation measures typically adopted to control PM emissions with reference to a major red mud basin located in the southwest of Sardinia (Italy). The PM dispersion models performed with the CALPUFF code (US EPA) allowed, for the case study under consideration, the estimate of the improvement provided by moistening the dry surfaces and reducing the total length travelled per year by the machinery involved in material handling, transportation and disposal.

Keywords: bauxite residue (BR), particulate matter (PM), fugitive dust, dust air dispersion, atmospheric impact, air pollutants.

1. Introduction
Before the seventies only seawater discharge and lagooning were in use worldwide for the disposal of the residue deriving from the Bayer process (purification of aluminum hydroxide) for the production of alumina (aluminum hydroxide). In fact, until that time the alumina industry, the governmental authorities and the human communities did not consider the bauxite treatment and the residue disposal as a critical issue.

From the seventies, the rapid growth of the alumina industry, the evolution of the environmental protection regulation and the ever-increasing social awareness about environmental impact issues have enforced the adoption of dry storage practices, such as dry stacking and dry disposal, aimed at reducing the environment impact on soil and groundwater. Nowadays 30% of the alumina refineries worldwide practice lagooning and marine discharge, while the remaining 70% have converted their practice into dry disposal [1].

However, such a change in the storage practice has posed a major concern with respect to the potential increase of fugitive dust impact, due to the work activities involving material moving and placing within the basin and to the wind erosion over the dried surfaces, both during the basin operation and afterwards, in the post-closure stage.

The article analyses the effect of the impact mitigation measures typically adopted to control fugitive dust emissions with reference to a major BR (Bauxite Residue) basin located in the southwest of Sardinia (Italy). It is worth mentioning that the same case study has been taken into consideration in previous articles [1, 2, 3]. The present paper includes and discusses the results of the PM impact analysis associated with the latest revision of the basin conversion project, according to the indications given by the governmental authority in the frame of the ongoing Environmental Impact Assessment (EIA) procedure [4, 5].

2. Bauxite residue storage practices

2.1. Marine discharge and lagooning

Marine discharge consists in the direct disposal of the residue into the ocean, by means of a pipeline that releases the slurry offshore. Nowadays, only a small percentage of red mud is disposed into the sea (around 2 – 3%).

The lagooning method consists in pumping a residue with a 25 – 30% solid content into ponds formed within natural depressions or into constructed basins. Ponds are usually lined to minimize underground liquor leakage. The reliability of the seal increases with the reduction of the hydrostatic pressure inside the basin, attainable by removing the liquor or draining the deposit. In such basins the liquor content is reduced to 35 – 30% either by evaporation from the lagoon surface and because of the mud consolidation process.

The potential environmental risk due to BR disposal is related to the physical and chemical properties of the residue, which essentially depend on the type of raw material in use (bauxite), on the processing parameters and on the residue treatment before disposal. When considering the potential impact on the soil and the aquifer, the key points to be taken into consideration are the residue water content, the water pH and the potential capacity of the solid particles to release alkaline ions into rainwaters [2].

Because marine discharge implies direct contact between the residue and the seawater, it represents the disposal practice with the highest environmental risk. Whereas the potential environmental hazard posed by land disposal depends on the amount and quality of fluid in the mud and by the effectiveness
of the geological barrier (natural or artificial) in sealing the deposit and avoiding the contaminant migration into soil and groundwater. In that sense, lagooning certainly represents the most precarious option among land disposal practices [2].

2.2. Dry stacking and dry disposal
Dry stacking requires the residue to be thickened before discharge, until a solid content between 48-55% is reached. Within that range, the mud is thixotropic and can be pumped and transported by means of suitable conduits.

After discharging, the residue descends along a slope with a resting angle between 2° and 6°. The discharged layers are left to drain and dry in the air before overlaying the next stratum. This way the mud consolidates to a solid content of about 62-65%, while the liquid phase is removed because of surface decantation and evaporation.

The dry mud is self-supporting and the deposit can be developed to reach significant heights without the need for containment structures and, therefore, with a significant reduction in the construction and maintenance costs. In relation to the relatively low residual water content in the mud, dry stacking provides some significant results in terms of soil occupancy and contamination risk, as it minimizes the use of land and the potential for leakage. On the other hand, it requires a further level of treatment before discharge (thickening or filtration) and additional technical measures to control dust emission from the deposit surfaces.

Dry disposal requires the process residue to be filtered to a solid content of about 65-70% and washed with water or steam to recover the soda and reduce the alkalinity. The dry mud is transported to the disposal area by trucks or conveyors. This method improves the positive attributes already mentioned for dry stacking, as it requires even smaller extent of land and no containment structure, and avoids the environmental and health hazards associated with the presence of open caustic lakes, whereas the potential for leakage to the ground and groundwater is minimised. On the other hand, it requires the installation of large filtration plants and washing devices, as well as the implementation of specific technical measures to mitigate the dust lifting from the deposit surfaces.

3. The case study
The red mud basin under consideration is located in the southwestern coast of Sardinia (Italy), within the industrial area of Portovesme (Figure 1). Since the seventies, the bauxite residue has been disposed by the lagoon method.

Considering the geographic location of the basin and its relative meteorological variables (rain and evaporation rate), the prospect of adding new embankments over the existing basin would imply a constraint on the vertical growth rate of the basin up to a limit of 1 m/y.

In fact, only by slowing the basin vertical development, the red mud already disposed in the basin would reach by consolidation the required solid content (about 65%).

The currently available evaporation surface (84.8 ha) combined with the limit in the raise velocity (1 m/s) would consent a maximum discharge volume of 850,000 m³/year, corresponding to a maximum production of alumina of 1,580,000 t/y [3]. In four years time the total evaporation area would be reduced to 66,4 ha, the maximum dischargeable volume would be 664,000 m³ and the alumina production rate could not exceed 1,230,000 t/a, with potential adverse effects for the alumina company in terms of economic outcomes.

From this perspective, also considering the unavailability of land and the environmental impacts associated with the hypothesis of a new basin in a different site, a conversion project has been developed to change the disposal practices from dry lagooning into dry disposal and use the existing basin [3].

3.1. The basin conversion project
Since a few year ago the red mud has been disposed in the three existing sectors of the basin A, B and C, represented in Figure 2. The two main sectors A and B are 26 m high and cover 114 ha of land; they have been developed according to the up-stream method and presently consist of a 10 m high lower embankment and 9 secondary embankments. Sector C is composed of a base embankment and only one secondary embankment, it covers 44 ha of land and is 11,5 m high.

The basin conversion project includes a variety of preliminary operations necessary to adapt the existing three sectors of the basin to the requirements of the EU Directive on the landfill of waste [6] and prepare an additional disposal area towards the north for the enlargement of the basin (new sector D). Figure 3 represents the future intermediate configuration of...
the basin (+18.5 m slm), with the three existing sectors (A, B and C) and the new sector D.

The modification of the disposal practice from lagooning to dry disposal also implies the construction of a filtration plant to allow the dehydration of the bauxite residue to a solid content of about 70%. Once dried, the residue is loaded onto the dump trucks and transferred to the basin summit, where is discharged into few main piles from where is reloaded, spread and rolled by means of traditional earth-moving machinery (wheeled loaders, dozers, compactor rollers).

On the other hand, the conversion project requires the modification of the basin characteristics to allow the implementation of the new disposal procedures (loading, transport and location of the dried residue), which also implies the formation of new large dried surfaces exposed to wind erosion, with consequent increase of fugitive dust emission. In order to reduce dust emission at the source, the following technical measure have been included in the conversion project under consideration: moistening or chemical treatment of dry surfaces, revision of work organization and wind barriers.

As regards the wind barrier, in particular, an experimental research is presently in progress to estimate the threshold velocity that activates the dust-lifting phenomenon. That implies the use of specific instrumentation, equipped with a laser scattering system for the real time detection of air dust concentration (TSP, PM10 and PM2.5). The simultaneous measurement of the wind velocity and direction near the basin surfaces exposed to wind, at few centimeters from the ground, will be necessary to correlate the threshold velocity to the lifting phenomenon. The results of the research will permit the design of a barrier system calibrated for the case study under consideration.

In the following part of the article, the results of the dust dispersion simulation are reported with reference to the effect provided by revising the work organization and moistening the dry surfaces of the basin, with mobile or fixed equipment. The first point, in particular, implies the reduction of the number of earth-moving machinery to be used for material disposal, according to the new storage practise.

4. The PM dispersion modelling

4.1. The CALPUFF code

The dust dispersion simulation has been carried out with the CALPUFF model system, developed by Sigma Research Corporation (currently part of Earth Tech, Inc.), with the contribution of the California Air Resources Board [7]. The modelling domain for the case study under consideration is a square with sides of 20 km, centered in the red mud basin (within the Industrial Area of Portovesme), and includes the two nearest villages of Paringianu and Portoscuso, respectively at 800 m and 3 km, which represents the main receptors in the surrounding territory.

4.2. Characterization of fugitive dust sources

Tables 1 reports all fugitive dust emission sources (from S1 to S8) included in the lasted revision of the basin conversion project (construction activities, disposal operations and wind erosion) and the applicable EPA AP42 codes [8]. For each of the eight sources under exam, Table 2 indicates the specific algorithm for the valuation of the PM10 Emission Factors (PM10 EF), as well as the equation parameters needed for the calculation. Table 3 reports the operating parameters used to calculate the Emission values (E): input data of the impact prevision models. For the activity S3 (Dry mud), two cases were considered: Case A (dumper capacity of 16.3 m$^3$) and Case B (dumper capacity of 20.0 m$^3$).
### Table 1 – Fugitive dust sources and selected EPA AP42 codes

<table>
<thead>
<tr>
<th>Code</th>
<th>Fugitive dust source</th>
<th>EPA AP42 Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>S1</td>
<td>Dry mud unloading from filter press</td>
<td>11.19.2 Conveyor Transfer Point</td>
</tr>
<tr>
<td>S2</td>
<td>Dry mud loading into truck</td>
<td>11.19.2 Truck</td>
</tr>
<tr>
<td>S3*</td>
<td>Material transport with dumpers</td>
<td>13.2.2 Unpaved road</td>
</tr>
<tr>
<td>S4</td>
<td>Heap formation &amp; material handling with loaders</td>
<td>13.2.4 Aggregate Handling and Storage Piles</td>
</tr>
<tr>
<td>S5</td>
<td>Stockpile handling</td>
<td>13.2.5 Industrial wind erosion</td>
</tr>
<tr>
<td>S6</td>
<td>Material placing with motor graders</td>
<td>13.2.3 Heavy Construction Operations - Grading equation Tables 11.9-2</td>
</tr>
<tr>
<td>S7</td>
<td>Material rolling with dozers</td>
<td>13.2.3 Heavy Construction Operations- Dozer equation in Tables 11.9-2</td>
</tr>
<tr>
<td>S8</td>
<td>Wind erosion from exposed surface</td>
<td>SPPC 1983 - Appendix A Section 1.1.17 to 1.1.18</td>
</tr>
</tbody>
</table>

*S* Includes: Dry mud, bottom and fly ash, lateral capping building materials

### Table 2 – PM10 Emission Factors (EF)

<table>
<thead>
<tr>
<th>Code</th>
<th>EF PM10</th>
<th>Equation parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td>S1</td>
<td>5.50 $10^{-4}$ [kg/Mg]</td>
<td>$k, a, b = \text{particle size coefficient}$ $s = \text{surface material silt content (%)}$ $W = \text{mean vehicle weight (tons)}$</td>
</tr>
<tr>
<td>S2</td>
<td>5.50 $10^{-4}$ [kg/Mg]</td>
<td>$k = \text{particle size multiplier}$ $U = \text{mean wind speed (m/s)}$ $M = \text{moisture content (%)}$</td>
</tr>
<tr>
<td>S3*</td>
<td>$k \cdot \frac{s}{12} \cdot (W^b \cdot 291.8)$ [kg/km]</td>
<td>$a = \text{area of moved surface (m$^2$)}$ $a_{\text{movh}} = \text{number of movements per hour (1/h)}$</td>
</tr>
<tr>
<td>S4</td>
<td>$k \cdot 0.0058 \cdot \frac{1}{M^4} [\text{kg/Mg}]$</td>
<td>$S: \text{mean vehicle speed (km/h)}$</td>
</tr>
<tr>
<td>S5</td>
<td>$7.9 \times 10^{-5} a_{\text{movh}}$ [kg/h]</td>
<td>$S: \text{mean vehicle speed (km/h)}$</td>
</tr>
<tr>
<td>S6</td>
<td>$0.0056 \cdot 0.6 \cdot (S)^2$ [kg/km]</td>
<td>$s: \text{silt content (%)}$ $M: \text{moisture content (%)}$</td>
</tr>
<tr>
<td>S7</td>
<td>$0.45 \cdot (S)^{1.5} \cdot 0.75 (M)^{1.4}$ [kg/h]</td>
<td>$S: \text{mean vehicle speed (km/h)}$</td>
</tr>
<tr>
<td>S8</td>
<td>$0.2$ [kg/ha/h]</td>
<td></td>
</tr>
</tbody>
</table>

*Includes: Dry mud, bottom and fly ash, lateral capping building materials

### Table 3 – Estimate of Emission (E): operating parameters

<table>
<thead>
<tr>
<th>Code</th>
<th>Operating parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td>S3 (Dry mud)</td>
<td>Case A</td>
</tr>
<tr>
<td>Case A - DAY</td>
<td>$a = 0.9$ $b = 0.45$ $k = 1.5$ $s = 10%$ $W = 55$ tons</td>
</tr>
<tr>
<td>Case A - NIGHT</td>
<td>$4.0$ $s = 10%$ $W = 59$ tons</td>
</tr>
<tr>
<td>S3 (Dry mud)</td>
<td>Case B</td>
</tr>
<tr>
<td>Case B - DAY</td>
<td>$a = 0.9$ $b = 0.45$ $k = 1.5$ $s = 10%$ $W = 55$ tons</td>
</tr>
<tr>
<td>Case B - NIGHT</td>
<td>Daily kilometer traveled: 37.5 km</td>
</tr>
<tr>
<td>S3 (Bottom and fly ash)</td>
<td></td>
</tr>
<tr>
<td>Daily kilometer traveled: 453 km</td>
<td></td>
</tr>
<tr>
<td>S3 (Lateral capping building materials)</td>
<td></td>
</tr>
<tr>
<td>Daily kilometer traveled: 77.5 km</td>
<td></td>
</tr>
</tbody>
</table>

### Table 4 – Emission values (E) for the two modelling scenarios

<table>
<thead>
<tr>
<th>Code</th>
<th>Scenarios</th>
<th>Notes</th>
</tr>
</thead>
<tbody>
<tr>
<td>S3 (Dry mud)</td>
<td>Case A - DAY</td>
<td>46.15</td>
</tr>
<tr>
<td>S3 (Dry mud)</td>
<td>Case A - NIGHT</td>
<td>30.05</td>
</tr>
<tr>
<td>S3 (Dry mud)</td>
<td>Case B - DAY</td>
<td>38.82</td>
</tr>
<tr>
<td>S3 (Dry mud)</td>
<td>Case B - NIGHT</td>
<td>25.25</td>
</tr>
<tr>
<td>S3 (Bottom and fly ash)</td>
<td></td>
<td>12.87</td>
</tr>
<tr>
<td>S8</td>
<td></td>
<td>3.57</td>
</tr>
</tbody>
</table>

The Emission values (E) in Table 4, used to run the air dispersion models, refer to the two scenarios: Scenario 1 (emission without mitigation) and Scenario 2 (emission with mitigation). Scenario 2 represents, in fact, the additional effect provided by wetting the basin surfaces: unpaved roads, top and lateral surfaces of the basin. An abatement efficiency of 85% has been assumed to run the simulation of Scenario 2.
5. Results and discussion

As mentioned above, the prevision models of dust dispersion were performed with reference to two hypothesis of work organization (Case A and Case B) and two Scenarios (Scenario 1 and Scenario 2). The modelling results refer to the following configurations:

1. Case A – Scenario 1
2. Case B – Scenario 1
3. Case A – Scenario 2
4. Case B – Scenario 2

For each configuration, the results are reported in a numerical format for the four points (CENPS 2, CENPS 4, CENPS 6 and CENPS 7) corresponding to the locations of sampling stations set up by the Environmental Protection Agency of Sardinia (ARPAS) [9] shown in Figure 3.

The incremental impact of the basin to the PM concentration values recorded by the ARPAS monitoring system allows the comparison with the limit values established by Directive 2008/50/EC on ambient air quality and cleaner air in Europe [10]. It is worth mentioning that the Directive establishes the limit values of PM10 for one day (50 µg/m³, not to be exceeded more than 35 times a calendar year) and for a calendar year (40 µg/m³); both limits have been in force since January 2005.

For each of the four simulated configurations, the contribution of the red mud basin resulting from the prevision models is given in terms of dust concentration for the 36th worst day (mean daily value) and mean annual value. Only for the first and last configuration (1 and 4) the results are reported also graphically (iso-concentration maps).

5.1. Simulation numerical results

Table 5 and Table 6 report the numerical results of the simulations for the configurations under exam. As shown in Figure 3, the sampling stations CENPS6 and CENPS7 represent respectively Paringianu and Portoscuso, the two villages nearest to the impact source.

The PM10 concentration values in Tables 5 and 6 refer to the 36th worst day of the year (mean daily value) and to the mean yearly value (Y). All the simulated concentration values are far below the limits established by Directive 2008/50/EC, yet they solely represents the contribution of the red mud basin, while the impact evaluation procedure (i.e.: confrontation with the limit values) requires the additional contribution of the PM10 concentration registered at the sampling stations.

The significant difference between the concentration values at the North side of the BR basin (Portoscuso) and those at the South (paringianu) depends of the Sardinian prevailing wind from North-West (maestrale).

Table 5 – PM10 concentration at the sampling stations

<table>
<thead>
<tr>
<th>Case A</th>
<th>Scenario 1</th>
<th>Scenario 2</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>36th day</td>
<td>Y</td>
</tr>
<tr>
<td>CENPS2</td>
<td>2.57</td>
<td>0.71</td>
</tr>
<tr>
<td>CENPS4</td>
<td>2.91</td>
<td>0.84</td>
</tr>
<tr>
<td>CENPS6</td>
<td>8.70</td>
<td>3.02</td>
</tr>
<tr>
<td>CENPS7</td>
<td>3.65</td>
<td>1.16</td>
</tr>
</tbody>
</table>

Table 6 – PM10 concentration at the sampling stations

<table>
<thead>
<tr>
<th>Case B</th>
<th>Scenario 1</th>
<th>Scenario 2</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>36th day</td>
<td>Y</td>
</tr>
<tr>
<td>CENPS2</td>
<td>2.20</td>
<td>0.61</td>
</tr>
<tr>
<td>CENPS4</td>
<td>2.47</td>
<td>0.72</td>
</tr>
<tr>
<td>CENPS6</td>
<td>7.32</td>
<td>2.58</td>
</tr>
<tr>
<td>CENPS7</td>
<td>3.17</td>
<td>0.99</td>
</tr>
</tbody>
</table>

As regards the confrontation between Scenario 1 and Scenario 2, the reduction in terms of air dust concentration is necessarily correspondent to the moistening abatement efficiency assumed as input data (85%). While the mitigation offered by using dumpers of greater capacity is in the range between 13 and 16%, when considering the daily mean values, and around 15% when considering the annual mean values.

The global reduction of PM10 concentration at the receptors given by both moistening the dry surfaces and reducing the number of earth-moving vehicles involved in the construction and disposal activities is around 87%.

5.2. Simulation Graphical results

Figures from 4 to 7 represent the resulting isocentre concentration curves for the worst (Case A – Scenario 1) and the best case (Case B – Scenario 2) under consideration.
1. Conclusions

The recent change in the red mud storage practices has posed a major concern due to the potential increase of PM atmospheric impact due to the disposal activities and the wind erosion. The article analyses the effect of the impact mitigation measures to control PM emissions with reference to a major red mud basin located in the southwest of Sardinia (Italy).

The PM dispersion models performed with the CALPUFF code (US EPA) allowed the estimation of the improvement provided by moistening the dry surfaces and by reducing the total length travelled par year by the machinery involved in the construction and disposal activities. The results showed that the global reduction of PM10 concentration at the nearest receptors (Paringianu and Portoscuso) is around 87%.

Apart from the potential improvement provided by the reduction of the total length travelled par year (i.e.: use of bigger machinery) and by moistening the dried surfaces, an additional mitigation measure is represented by the erection of wind barriers. In fact, an experimental research is presently in...
progress to estimate the threshold velocity that activates the dust-lifting phenomenon. That implies the use of specific instrumentation, equipped with a laser scattering system for the real time detection of air dust concentration (TSP, PM10 and PM2.5) and the simultaneous measurement of the wind velocity and direction near the basin surfaces exposed to wind, to correlate the threshold velocity to the lifting phenomenon. The results of that research will permit the design of a barrier system calibrated for the case study under consideration, which will be the object of a future article.

Acknowledgement
The study was carried out in the framework of projects conducted by Cagliari, Italy, and by CINIGeo (National Inter-university Consortium for Georesources Engineering, Rome, Italy).

References
Prediction of Acid Mine Drainage in coal mine waste based on Artificial Intelligence Method

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Abstract— With regards to unprecedented rate of mining in recent decades, global concern about mining waste, environmental contaminants, dust production, leakage of toxic materials from the tailings dam, directly or indirectly disposal of waste in waterways and tailings dam failure is increased significantly. According to the above conditions and diminish of the resources, reduction of grade reserves, and the climate change, managing the disposal of mining waste will be challenging issue in the future. One of the important issues in the management of waste disposal is the control of AMD. In this study, alongside on the exploring and understanding the sources of AMD generation, an ANFIS model based on artificial intelligence methods is developed to predict the effects of behavior properties of jig tailing waste of coal mine on generation of AMD. To establish this model some particular data including temperature, amount of precipitation, and amount of used pyrite were measured from jig waste dump of Tazareh coal mine. Comparison of the predicted result and actual values shows that the ANFIS model is capable of estimating the amount of AMD generation within the ±10 percent error range which is a reasonable range.

Keywords: acid mine drainage (AMD), ANFIS, coal mine waste.

1. Introduction

Acid Mine Drainage is one of the sources of environmental pollution associated with sulfide mineral oxidation that occurs in tailing dam, mine sites and mine bottom pit and walls [1]. Most of sulphide mine tailing is containing a collection of minerals or aggregation of polyminerel. The whole range of minerals including valueless minerals may come bundled with sulphide minerals that may be present as a useful or valueless mineral. When sulphide minerals are exposed to air or proximity to water (source of oxygen); an oxidation take place. As a result of these interactions, oxide water is produced which is containing heavy metals [2,3]. Progressive entry of AMD waters; as a result of mining operations; into the rivers and lakes, can cause irrecoverable pollution that its blocking is difficult and may continue for consecutive years. Main acid production sources, especially sulphides are including: Pyrite (FeS2); Marcasite (FeS2) and Pyrrhootite (Fe8S11) that all of which are iron sulfides. Although the Oxidation rate of marcasite and pyrrhootite is faster rather than pyrite, but because of the abundance of pyrite in nature in compare with other ones, it is recognized as a main source of AMD generation. Moreover; under the following conditions AMD water is formed [4]:

1- Underground spaces which are located at the top of water table level are affected by groundwater.
2- In open pit mines, groundwater through the floor or wall of mines is leaking to working area.
3- By monsoon and thus permeating of resulting water into bottom and wall of mine.
4- By monsoon and thus permeating of resulting water into dump of coal and metallic minerals, waste dump or heap leach piles.
5- Leaking of rain water into the tailing dam.
6- Contact of rainwater and its interaction mining operation.
7- Discharge of processed water from tailing dam, water tanks and heap leach piles.

AMD waters are formed very fast and are recognizable with possible signs such as; low pH and appearing of yellow-red bound. A classification of acidic waters is shown in table 1.

Table 1- Acidic waters classification based on acidity and it’s mineral

<table>
<thead>
<tr>
<th>pH</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>pH&lt;1</td>
<td>Extremely acidity (rocks are extremely rich with pyrite)</td>
</tr>
<tr>
<td>pH&lt;5.5</td>
<td>Acidic (rocks are rich with pyrite)</td>
</tr>
<tr>
<td>pH&lt;5.5 (AMD)</td>
<td>Acidic (heavy metals such as Fe, Cu, Pb, Zn; metalloid such as: As and other elements such as: Si, K, Ba)</td>
</tr>
</tbody>
</table>

Moreover; waste dump of coal mining and washing operation from coal plants, is among the potential environmental place for AMD generation [2].
Therefore preventing of associated risk in the field of intoxication and contamination of water bodies and wildlife is deeply an important issue. Early forecasting and thus counting the cost of preventing the aspect of arising AMD generation, in the preliminary stages of mine design and its various segments such as production planning leads to more realistic cost analysis[5,6,7,8]. Forecasting of AMD generation from sulfide tailing and wastes can be estimated via geological survey, petrography, geological modeling, static test, measuring the rate of chemical changes and mathematical models. Having investigation on many researchers work shows a lot of developed model including one dimensional, numerical model, computational fluid dynamics and finite difference model, to simulate AMD generation and pyrite oxidation in the waste produced by the coal operation. Since, to interconnect the consumption of pyrite and affecting parameters involved in it, many items such as pyrite particle size, reaction depth, temperature, amount of precipitation (abundance of water), role of trace element, pyrite crystallography and other effective parameters must be considered, therefore, challenging with this non-linear mathematical problem is a cost and time consuming problem [9] . During recent decades by developing new mathematical methods and along with it dramatically growth of computer speed, the opportunity to offer and simultaneously solving of very complex non-linear problem by helping of artificial intelligence methods (AI) is arisen. AI algorithms are widely used in many cases of hard problems. The amount of pyrite oxidation process influenced by the discussed factors can be assumed as an NP-hard case. In this research by using adaptive neuro fuzzy inference system (ANFIS) methodology tries to draw a policy for prediction of pyrite oxidation process with in the jig waste dump of Tazareh coal mine.

2. Theoretical Foundations

A schematic model of potential location for AMD generation can be exampled in Figure 1. According to it the aspect of factor affecting on acceleration of oxidation and discompose of balancing drainage status, should be deeply noted. In this regard a brief chemistry summary of sulfide oxidation is shown in equation 1.

\[
4 \text{FeS}_2 + 15 \text{O}_2 + 14 \text{H}_2\text{O} \rightarrow 4 \text{Fe(OH)}_3 + 8 \text{H}_2\text{SO}_4
\] (1)

The most common chemical reaction of pyrite oxidation is weathering with the presence of oxygen (Eq 2).

\[
2 \text{FeS}_2 + 7 \text{O}_2 + 2 \text{H}_2\text{O} \rightarrow 2 \text{Fe}^{2+} + 4 \text{SO}_4^{2-} + 4 \text{H}^+
\] (2)

Next reaction is the oxidation of ferrous irons to form ferric iron (Eq3).

\[
4 \text{Fe}^{2+} + \text{O}_2 + 4 \text{H}^+ \rightarrow 4 \text{Fe}^{3+} + 2 \text{H}_2\text{O}
\] (3)

The rate of above equation can be accelerating with the assistance of certain bacteria. The third one is the hydrolysis of ferric iron (Eq 4)

\[
4 \text{Fe}^{3+} + 12 \text{H}_2\text{O} \rightarrow 4 \text{Fe(OH)}_3 + 12 \text{H}^+
\] (4)

The last but not the least reaction is continues cycle oxidation of additional pyrite by ferric iron (Eq5).

\[
\text{FeS}_2 + 14 \text{Fe}^{3+} + 8 \text{H}_2\text{O} \rightarrow 15 \text{Fe}^{2+} + 2 \text{SO}_4^{2-} + 16 \text{H}^+
\] (5)

Figure 2 is an illustration model of pyrite oxidation process in a tailings impoundment. Here, oxygen infiltrate into pore space and then touch pyrite granules which is finally lead to AMD generation.

![Figure 1. A conceptual schematic model of potential location for AMD](image-url)
3. Study Area
Alborz-Sharghi has four main coal sections including Kalariz, Poshkalat, Razmja and Mamdooyeh which are located in northern part of Mehmandoost region in the Shahrood province. Tazareh and Mehmandoost are nearest villages to the coal zone. Mehmandoost is the industrial base of the Alborz-Sharghi which is the accommodation of workshop; warehouse and main power plant complex. This village is 10 Km far away from Razmja coal region (Figure 3). Alborz-Sharghi coal washing plant is in the way of Mehmandoost to Tazareh. This region has a mountainous climate, maximum height of 2600 meters above sea level, with a cold winter and mild to hot summer. This study is conducted according to the core data collected among the piles of Alborz-Sharghi washing coal plant belongs to Tazareh coal mine.

4. Materials and methods
4.1. Preliminary data
Generally the initial data used in this study, includes data derived from sampling conducted of waste dump of coal washing plant for Tazareh mine during different years, monthly precipitation in different years, the average monthly temperature in different years and oxygen penetration depth in waste dump. According to the jodeiri et al (2014) investigation, during four years of sampling operations, some number of sample were taken from waste dump of coal washing plant. Concentration of pyrite remained in sample cores were determined by atomic absorption analysis. A sample of measured values is available in table 2.

<table>
<thead>
<tr>
<th>Depth(m)</th>
<th>Pyrite content remaining for sampling points (mol/m3)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.00</td>
<td>43.35  27.76  50.02  69.19</td>
</tr>
<tr>
<td>0.90</td>
<td>104.20 111.70 113.37 114.78</td>
</tr>
<tr>
<td>1.40</td>
<td>128.38 128.54 134.54 136.94</td>
</tr>
<tr>
<td>2.00</td>
<td>136.71 173.39 145.05 153.38</td>
</tr>
</tbody>
</table>

Also in the above-men tioned sampling procedure the amount of oxygen concentration in the waste dump at different depth is reported in Table 3. In the following Table 4 and Table 5 hold historical data on average monthly temperature and precipitation during the sampling months.

<table>
<thead>
<tr>
<th>Depth(m)</th>
<th>Oxygen content remaining for sampling points (mol/m3)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.0</td>
<td>8.90  8.90  8.90  8.90</td>
</tr>
<tr>
<td>0.9</td>
<td>2.75  2.08  2.75  2.33</td>
</tr>
<tr>
<td>1.4</td>
<td>0.59  0.76  0.42  0.21</td>
</tr>
<tr>
<td>2.00</td>
<td>0    0     0    0</td>
</tr>
</tbody>
</table>
Table 4 – Historical data on average monthly precipitation during the sampling months

<table>
<thead>
<tr>
<th>Year</th>
<th>Month</th>
<th>Sep</th>
<th>Nov</th>
<th>Dec</th>
</tr>
</thead>
<tbody>
<tr>
<td>2002</td>
<td></td>
<td>4.7</td>
<td>25.9</td>
<td>3.2</td>
</tr>
<tr>
<td>2003</td>
<td></td>
<td>5.6</td>
<td>10.3</td>
<td>17.5</td>
</tr>
<tr>
<td>2004</td>
<td></td>
<td>0.1</td>
<td>24.8</td>
<td>18.3</td>
</tr>
<tr>
<td>2010</td>
<td></td>
<td>0.4</td>
<td>0.0</td>
<td>15.8</td>
</tr>
</tbody>
</table>

Table 5 – Historical data on average monthly temperature during the sampling months

<table>
<thead>
<tr>
<th>Year</th>
<th>Temperature (°C)</th>
<th>Month</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Sep</td>
<td>Nov</td>
</tr>
<tr>
<td>2002</td>
<td>21.9</td>
<td>3.7</td>
</tr>
<tr>
<td>2003</td>
<td>19.2</td>
<td>4.5</td>
</tr>
<tr>
<td>2004</td>
<td>18.5</td>
<td>3.9</td>
</tr>
<tr>
<td>2010</td>
<td>20.9</td>
<td>7.6</td>
</tr>
</tbody>
</table>

4.2. Statistical Study of Sampled data to predict amount of pH and Ec

During the sampling operation, the amount of pH and Ec was measured in two points of P9 and P10 (Table 6 and Table 7). So the other point remained undefined values. This impairment can be solved by establishment of cater outfit to predict undefined points. In this regard a statistical analysis to predict unvalued point is the purpose of this section.

Table 6 – Measured pH and Ec data for point P9

<table>
<thead>
<tr>
<th>Depth(m)</th>
<th>pH</th>
<th>Ec</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.00</td>
<td>4.1</td>
<td>565</td>
</tr>
<tr>
<td>0.10</td>
<td>4.18</td>
<td>439</td>
</tr>
<tr>
<td>0.20</td>
<td>4.26</td>
<td>313</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2.00</td>
<td>6.9</td>
<td>153</td>
</tr>
</tbody>
</table>

Table 7 – Measured pH and Ec data for point P10

<table>
<thead>
<tr>
<th>Depth(m)</th>
<th>pH</th>
<th>Ec</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.00</td>
<td>5.9</td>
<td>123</td>
</tr>
<tr>
<td>0.10</td>
<td>6.02</td>
<td>249.4</td>
</tr>
<tr>
<td>0.20</td>
<td>6.14</td>
<td>375.8</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2.00</td>
<td>6.9</td>
<td>153</td>
</tr>
</tbody>
</table>

More in the following by considering the amount of pyrite in the samples at different depths as the dependent variable, and amount of oxygen, pH, Ec, as independent variables, and by applying three indecent variables regression model, the following equation in conjunction with pyrite concentration can be calculated (Eq 6 and Eq7).

For point P9:

\[ \text{Used Pyrite} = 3.32 \times O_2 + 47.41 \times \text{pH} + 0.021 \times \text{Ec} - 193.023 \]  \hspace{1cm} (6)

For point P10:

\[ \text{Used Pyrite} = -7.51 \times O_2 + 7.99 \times \text{pH} - 0.049 \times \text{Ec} + 103.404 \]  \hspace{1cm} (7)

The correlation results from estimating with any of the above equations and real values is illustrated in figure 4 and figure 5. Now for any of the remained sampling points without defined pH and Ec, by solving the system of equation in two variables, unknown values for quantity of used pyrite could be predicted.

Figure 4. Correlation between predicted values and actual values for sampling Point 9

\[ y = 0.9615x + 3.5398 \]

\[ R^2 = 0.9615 \]
4.3. ANFIS

ANFIS which is the abbreviation of Adaptive Neural Fuzzy Inference, in fact, is an entanglement of neural network and fuzzy interface system. The phraseology of ANFIS methodology is based on a plan and suggestion of Jang (1993). With similarities to artificial neural network, a translation of human brain inducted to computer software (in the form of fuzzy) which is capable of solving complex equations. The ANFIS sequence structure is shown in Fig. 6. After establishing and initializing the model based on fuzzy rules, derived from pairs of data (Input/Output) and by using subtractive clustering or grid partitioning technique, the system based on incorporation of least-squares and gradient-descent method, establishing training procedure at next step. Here, in the hidden layer of system, the optimal rules criterion, decrease the training data and make the system ready for next step. In continue, the constructed model must be examined by checking error rules and cycling procedure until minimum accessible error is achieved. To arrive on final model of structure an update based on back propagation learning method must take place. In this study, a dataset including, average temperature, average precipitation, depth of sampling and quantity of used pyrite in relevant depth (Eq 6 and Eq 7) was collected as input data and the output is anticipating amount of pyrite remaining. In this research 70 percent of data was awarded to this process and the rest of 30 percent was used as test data.

5. Result and discussion

The performance result of developed model for the foresaid data is shown in Figure 7. According to this graph, this network is well-trained as far as it can purse the different variation. To evaluate the model variation, mean square error of approximation can be a good criterion (Figure 8). As seen in Figure 8 and Figure 9, mean square error and histogram of estimation error of approximation during training data, is scattering around zero which reflex the proper training and ability to keep the error in an acceptable range of developed model.
The result of developed ANFIS, stating that it is capable of forecasting the amount of AMD generation within an acceptable range of error.

It can be proven by using 30 percent of input data as a test data and comparing the result (Fig. 10). Quantities estimation of test data error and relevant histogram is the evidence of it (Fig. 11 and Fig. 12).
Among the advantages of this approach is that with typical and in hand data, the amount of AMD is easily in a short time and minimal cost is predictable while in traditional method different case is usually prevails. If the inherent of AMD generation process in waste dump at other mines would be similar to the studied case, the current model can be adapted for the case.

6. Conclusions

The aim of this study is based on developing an ANFIS model to predict the fraction of used pyrite and therefore the pyrite oxidation rate with a view to precipitation, ambient temperature and Oxygen penetration depth in the wastes dump of coal washing plant of Tazareh mine in north part of IRAN. Among the advantages of this model is the availability and measurability of data. The results show that the estimation error range of model lies on ±10 percent which consider being an acceptable one. It can be imagined by using foresaid model; a systematic method to prevent pollution release to the environment can be designed. Since the time can be expected as a parameter in AMD generation therefore it is suggested by anticipating the production time of acid drainage generation, contributing the relevant cost of prevention in optimal cut-off grade and production scheduling is considered in other to achieve more realistic cost analysis.
Determination of The Optimum Design Parameters of Pb-Zn Mine Tailing Dam Using 2D Modelling

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1 Department of Mining Engineering, Istanbul University, Istanbul, TURKEY

Abstract— The most common tailing disposal methods are tailing dams. The physical stability of tailings stored in tailing dams is very important. The accidents in tailing dams are an important part of environmental events that have taken place in mining operations. It has also a cost in the mining activities of tailing dams. It is very important to plan the tailing dams economically and safely.

In this study, appropriate numerical modeling for different scenarios for the tailing dam was carried out considering the capacity, geographical conditions, geomechanical properties and geometry of the present state of a Pb-Zn tailing dam. In the created models, the areas of the overhead sections are considered as the unit cost of the dams and evaluated together with the safety coefficients. It was observed that limit slope angle is 40 degrees for the downstream tailings dam, and it is about 38-39 degrees for the upstream tailings dam. It was seen that the cross section area of downstream tailing dam is ~3.5 times higher than the cross section area of upstream tailing dam for the same safety factor.

Keywords: Tailing Dam, Mine Planning, Numerical Analyses

1. Introduction

The stability of deposited tailings in tailings dam is important issue for environmental considerations. Coarse or fine-grained enrichment wastes can be used to fill gaps in mines. In practice, however, many mine enrichment wastes are stored in waste ponds or heaps. The work on this issue has intensified in the fact that the accidents that occurred in the dams / pools where mineral wastes were stored in Europe in the 2000s caused serious environmental problems. Enrichment wastes stored in large heaps or in large ponds can cause serious impacts on the environment, human health and safety, as a result of slipping or collapsing ponds [1].

The physical stability of tailings stored in heap or in tailing dams is very important. Since the 1960s, accidents in tailing dams have been an important part of environmental events that have taken place in mining operations. Fine-grained mine tailings can lead to mudflows that are devastating when the water is saturated and exposed to water in terms of their physical properties. The increase in the number of these accidents it becomes more important of related work [2].

In the case of landfilled tailing dams, the cost is directly proportional to the filling capacity. Since the initial investment costs are low in these dams, they are widely used in ore preparation plants. The dam is heavily used, and as a filling material, the cover material and rocks in the region where the ore is produced are generally used [3].

In this study, different designs and scenarios were developed to plan a Pb-Zn mine tailing dam. These scenarios are modeled using the finite element method. As a result of the created models, the cross sectional area of dam, safety factor and maximum displacement outputs are obtained. The model was evaluated in terms of safety and economical optimum dam design and slope angle.

2. Material and Methods

In this study, the present capacity of Pb-Zn metal waste dam in Turkey [8], waste material characterization, topography characteristics, dam geometry are taken as basis. The properties of the tailings used are given in Table 1.

Table 1: The data for numerical analysis [4-5]

<table>
<thead>
<tr>
<th>Material Parameter</th>
<th>Unit Embankment</th>
<th>Tailing</th>
<th>Soil</th>
<th>Clay</th>
<th>Gravel</th>
<th>Sand</th>
</tr>
</thead>
<tbody>
<tr>
<td>Unit Weight (γunsat)</td>
<td>kN/m³</td>
<td>18</td>
<td>18</td>
<td>20</td>
<td>17.28</td>
<td>19.165</td>
</tr>
<tr>
<td>Unit Weight (Saturated) (γsat)</td>
<td>kN/m³</td>
<td>20</td>
<td>21</td>
<td>21</td>
<td>18.065</td>
<td>19.636</td>
</tr>
<tr>
<td>Young’s Modulus (E)</td>
<td>kN/m²</td>
<td>5.0E+04</td>
<td>5.0E+04</td>
<td>1.2E+06</td>
<td>4.3E+04</td>
<td>1.4E+05</td>
</tr>
<tr>
<td>Poisson’s Ratio (ν)</td>
<td>-</td>
<td>0.2</td>
<td>0.2</td>
<td>3</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>Cohesion (c)</td>
<td>kN/m²</td>
<td>5</td>
<td>0</td>
<td>20</td>
<td>33.5</td>
<td>5</td>
</tr>
<tr>
<td>Friction Angle (φ)</td>
<td>°</td>
<td>35</td>
<td>32</td>
<td>45</td>
<td>13</td>
<td>55</td>
</tr>
<tr>
<td>Dilatancy Angle (ψ)</td>
<td>°</td>
<td>0</td>
<td>0</td>
<td>15</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>
The design of the upstream and downstream dams has been tested to obtain maximum and minimum values from waste dam types. The design, construction steps and materials of the downstream and upstream dam are shown in Figure 1.

As seen in Figure 1, downstream waste dumps and upstream tailing dumps are being constructed gradually. Considering the topography and the amount of waste to be produced, the elevation of each flood level will be 7 meters.

The design of 2 types of waste dams designated for comparison were modeled and analyzed by different slope angles. An attempt was made to find the optimum slope angle in set-type waste dams rising upstream with the set-up type rising downstream.

Material properties and designs were put forward, then scenarios for the numerical model were established. As a result of each modeling, the risk analysis was continued by increasing the slope angle by 5 degrees. With this method, 25, 30, 35, 40 and 45 degree models were created as the slope angle in the design of the rising waste dam. 25, 30, 35, 40 and 45 degree models were created in the same way as the slope angle of the flow dam [6].

3. Results and Discussion

In the created designs, the scenarios were started at 25 degrees with a slope angle and each successive scenario was modeled after a 5 degree slope angle increase. In these scenarios, increasing the angle of the slope was continued until the unsuccessful modeling based on slope stability. In this way, 25, 30, 35, 40 degrees in the downstream tailing dam and 25, 30, 35 degrees in the upstream tailing dam have been successful designs [6]. The designs collapsed when the designs had 45 ° downstream and 40 ° slope at the upstream. It can be seen the relationships between slope angle and maximum displacement and outputs at Figure 2.

As can be seen in Fig. 2, the maximum displacement value increases with the increase of the slope angle in general. Displacement values of downstream waste barriers were obtained as 1.95E-04, 1.07E-04, 2.51E-04, 3.63E-04 meters respectively and in upstream waste barriers as 7.69E-05, 8.47E-05, 1.19E-04 meters respectively. As a result, the maximum displacement value of the Upstream tailing dam design with a 30-degree slope angle is lower than the general one. It can be seen from the screen output that if the displacement center is more than one, the total tension can be distributed to the focus and decrease the maximum displacement value. For this reason, the maximum displacement values should be evaluated together with the position and the distribution in the design, rather than alone.

It can be seen from the displacement outputs, the importance of impoundment and tailing material interaction emerges. (Fig. 2) It has been seen that the maximum displacement points are directed from the dam onto the impoundment and the displacement values are increased in the impoundment with the increase of the slope angle in the downstream tailing dam. Besides, the maximum displacement values can be seen in the tailing rather than near the impoundment because which is seated on the tailing material in Upstream tailing dam. As the result of the increase of the slope angle in the Upstream tailing dam, there is an increase in the displacement value which occurs in the impoundment as the load on the which is not on the tailing material but on itself.

The relationship between safety coefficient - slope angle and cross section areas - slope angle is given in Figure 3.
As it is seen in Figure 3, the highest safety coefficient is 1.712 for the downstream tailings dam in the slope angle of 25 degrees, the lowest value of the same kind of tailings dam was obtained in the slope angle of 40 degrees. The highest safety coefficient is 1.546 for the upstream tailings dam in the slope angle of 25 degrees, the lowest safety coefficient was determined as 1.133 in the slope angle of 35 degrees. The decrease of the safety coefficient along with the increase of slope angle and the parallel course of the two dam types are quite clearly seen within two designs. The safety limit was reached in the design of the downstream tailing dam with a slope angle of 40 degrees. In contrast to this, the collapse occurred in the upstream tailing dam with the same slope angle because the safety coefficient was not achieved. Based on this trend, it is seen that the limit slope angle is 40 degrees for the downstream tailings dam, and it is about 38-39 degrees for the upstream tailings dam [7].

As a result of the change of the slope angle in the designs, there are formed different cross-sectional areas. Only the cross-sectional areas of successful scenarios are given in Fig. 3. As shown in Fig. 4, the downstream dam rising to 2835 m², which is the highest cross-sectional area in its design with a 25 degree slope angle, reached its lowest value of 1680 m² in the design with a 40 degree slope angle. When looking at the rising upstream dam, the highest value was 735 m² at a 25 degree slope angle and 560 m² at a 35 degree slope angle. The relationship between safety coefficient and cross section areas is given in Figure 4.

As can be seen in Figure 4, the increase of the safety factor and the increase of the cross-sectional area are directly proportional in both designs. 590 m² in the upstream tailing dam and 1940 m² in the downstream tailing dam were calculated for the lowest value of 1.15 for the safety factor. 740 m² in the upstream tailing dam and 2570 m² in the downstream tailing dam were calculated for the highest value of 1.55. It is seen that the cross section area of downstream tailing dam is ~3.5 times higher than the cross section area of upstream tailing dam for the same safety factor.

4. Conclusions

In this study, the two different types of tailing dam were compared using the numerical modeling method with changing slope angles. The outcomes as a result of numerical modeling, the displacement, safety coefficient values were obtained by the model used and impoundment to the cross section areas were calculated. The upstream tailings dam for 25, 30, 35 and the downstream tailings Dam for 25, 30, 35, 40 degrees with an angle of slope designs have remained stable.

Just to be pressed both in the design of tailings dam and safety factor of the cross section areas to be increased have been revealed. This is the cross section areas of impoundment represent costs to be manufactured. The design coefficient of the same security on the basis of two cross section areas were determined to be quite high the difference between. In addition, tailing has a higher coefficient if you want to make a dam Safety; downstream tailing dam cross section area is rising rapidly, the increase in upstream tailing dam is quite reasonable.

References


Geomechanical justification of the working area parameters for the placer deposit quarry under highly watering conditions

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Abstract: Controlling conditions of rock slope massif at the open pit is one of the key elements of surface mining operations. Stability of geotechnical objects at the pit, including slopes, edges, external and internal dumps, depends on the set of natural and technogenic factors which are subject to time and spatial changes. Considering such factors as variability of physical and mechanical properties of rocks and their moisture, level of underground waters, features of the mining technology applied, geometrical parameters of pit edges and dumps, external loadings from the transport equipment, etc. are the main direction of engineering and geological studies and a subject to numerous applied research.

This paper deals with practical considerations concerning placer deposit development and slope stability analysis at the open-pit of Vilnogirsk Mining and Metallurgical Plant (Vilnogirsk, Ukraine). This company is involved in surface mining of the Malyshevskoe zircon- rutile-ilmenite deposit. The overburden rocks of the benches are presented by greenish–gray clay, Quaternary reddish-brown clays and loess-type loam with total thickness of 60…65m. The main problem of the open-pit technology is that the titanium placer deposit located above the level of subsoil water. Also, hydrogeological conditions of mining operations are complicated by specific properties of the ore bed characterized as quicksand. Another weak point is the contact between loess loams and underlying clay seams with considerable plasticity. Loess-like loams have high porosity and high moisture coefficient.

The open-pit technology is carried out in a combined way using draglines and bucket-wheel excavator systems. Mining operations include extracting ore deposit with loading into trucks, transportation to the pump washing point via hydraulic giant and following hydraulic transport to the processing plant. Rotary excavator ER-5000 extracts and transports overburden rock mass in the external dump through the longwall conveyors, stacking conveyors and console overburden spreader located directly on the dump benches. Development of ore containing benches is operated by dragline excavators ESh-10/50 which placed on the bench roof with loading overburden rocks into dump trucks. Alternative scheme includes development of above-ore bench by dragline with loading rock into dump trucks, but ore bench is mined by dredger with hydraulic monitor and transportation of rocks through a slurry pipeline directly to the processing plant.

A distinctive feature of the mining of the ore bed is its location below the groundwater level, which creates certain uncertainties and challenges of the mining technology. Currently, ore reserves above the water table are close to exhaustion. So, development of watered placer deposit looks like important step in further life cycle of this mining company. Mining operations at considered site are greatly complicated by high water inflows in the quarry area that creates unfavorable conditions for the stability of open-pit benches and slopes. This challenge requires more precise investigation of changeable hydrogeological conditions to justify working area parameters and relevant mining operations.

Keywords: open pit, slope stability assessment, finite element analysis, strength reduction factor, safety factor

1. State-of-the-art and research objective
Justification of safe geometric parameters of the slopes for the whole pit edge should be considered at an early stage of mine planning and revised during the entire time span of technological operations [1]. The effectiveness of such measures depends primarily on the reliability and accuracy of slope stability assessments based on integrated engineering-geological and hydro-geological evaluations. Achievement of the assigned safety technological standards can be ensured by setting appropriate bench height and slope angle. Slopes and quarry benches are traditionally considered as geotechnical structures. Therefore, their development must be carried out in compliance with technical, economic, environmental requirements, and also the safety of mining operations [2]. Also pit edges and slopes are subject to natural geological and geomechanical factors that affect diverse deformations and failures of rock massif.

Innovative technical solutions allow precise analyzing the stability of quarry slopes with the use of geoinformation systems that provide visualization of their instability zones [3], [4], [7]. This approach contributes to the selection of acceptable criteria in the design of the resulting pit edge angles and allows using updated comprehensive data for making economically and technologically reasonable decisions regarding effective and safe mining. In addition, numerical methods provide valuable tools for geotechnical engineering that ensure calculations of slope stability parameters with high precision and durability [5], [6]. Application of the above mentioned approaches and research methods to determine stability of rock massif while open pit mining is very common in the design and planning mine operations. The main parameters taken into geotechnical design are technological characteristics of the development of particular deposit, rock physical properties, arrangement and movement of mining equipment, and the ways to transfer overburden to internal or external dumps especially when the deposit field is heavily watered [8], [9].

The objective of the research presented in this paper is justification of the working area parameters for the placer deposit quarry under highly watering conditions taking into account the stability factors of the rock massif and selection of the relevant mining equipment and open pit technology.
The research includes the following tasks: 1. Analysis of mining and geological conditions of the field development area, study of physical and mechanical properties of overburden rocks; 2. Carrying out geomechanical assessment of pit edge and slopes stability at the working area of the quarry taking into account complex structure of the rock massif, rock physical properties, mining equipment loads; 3. Determination of locations and shapes of potential sliding surfaces as well as shear strains and displacements in the pit edge and slopes of the rock massif; 4. Assessment of slope stability in highly watered conditions.

2. Methodology

To study the geomechanical stability of the open pit edge and slopes during surface mining, the data of engineering and geological surveys, technological schemes open mining operations were used. For numerical simulation of pit edge and slope stability the Phase2, Finite Element Analysis software by Rocscience company was used. The Mohr-Coulomb failure criterion was selected for geotechnical calculations and design.

3. Initial data

The following initial data were used for current research: design solutions at the time of work, results of laboratory testing rocks for determination of physical and mechanical properties, data of previously fulfilled theoretical and experimental studies for Vilnohirsk Mining and Metallurgical Plant (Ukraine) involved in the development of Malyshevskoye placer deposit, materials of research reports of the Institute of Nature Management and Ecology (National Academy of Sciences of Ukraine), Krivoy Rog Technical University, “Krivbassproject” Institute, scientific production association “Geotechnology” (Ukraine), report on the study of hydrogeological conditions of Motronovsko-Annovskyi section of the Malyshevskoye placer deposit, design studies of the Institute “Gorkhimprom” (Ukraine) and other materials available at Vilnohirsk Mining and Metallurgical Plant.

3.1. Engineering-geological estimation of the site

Motronovsko-Annovskyi section of Malyshevskoye placer deposit is located 10 km to the north-west of the operating open pit and Vilnohirsk City (Dnipropetrovsk region, Ukraine). The site area is a plain with a surface marks up to 170-178 m, divided by four valleys with up to 60 m in deep. Average absolute marks of the day surface vary within the range from 111 m to 174 m. Such significant differences in relief predetermine heterogeneous geological structure and landscape features (Fig. 1).

Soft overburden rocks of the placer deposit, having a layered structure, are represented by: loess-like and dense red-brown loams; Quaternary red-brown clays; greenish-gray clays of the Sarmatian stage; fine-grained sands of the Sarmatian stage; fine-grained sands of the Poltava series (ore containing layer). The thickness of overburden varies from 16 m to 70 m.

The ore containing layer presented in fine-grained Sarmatian and Poltava sands have a thickness of 10-12 m. The three upper overburden layers have higher physical and mechanical properties in comparison with the super-ore layer of fine-grained sands and the ore layer of fine-grained sands.

The roof of the ore layer with average thickness of 11 m lies at the elevation +100-110 m. The ore is represented mainly by quartz clayey sand, which is characteristic to the shelf deposits. The Sarmatian sands of the Neogene with a thickness of 6 to 27 m, and Upper Sarmatian plastered mottled clays 5-12 m in thickness are laid above the ore layer. The Upper layers of Quaternary red-brown clays up to 25 m in thickness are widespread only on the watersheds but they are washed out in the valleys. All deposits are covered with a layer of loess-like loams 5-10 m in thickness. The ore layer is underlain by the glauconite sands of the Kharkov stage with a thickness of 15-18 m. Below in the depressions of the base, sandy-argillaceous deposits of the Kiev stage are widespread. Even lower the carbonaceous sands of the Buchak stage of the Paleogene. Granites occur at depths from 50 to 120 m.

The physical and mechanical characteristics of the rocks of the Motronovsko-Annovskyi section of the Malyshevskoye placer deposit are presented in the Table 1.
3.2. Hydrogeological characteristics of the deposit

Four kilometers to the north-east from the investigated area the valleys merge into the valley of the Domotkan River which flows into the Dnieper. Ore sand and sedimentary deposits, including fractured granites, contain an aquifer complex. The water level is at the mark of +100-110 m, which practically coincides with the roof of the ore layer. The engineering and geological conditions for conducting mining operations are complicated, primarily by the quick ground properties of the ore layer. Another weak point is the contact of loess-like loams with underlying clays. Loess-like loams are characterized by high porosity and perched groundwater is occurred in their lower part on the contacts with clays. These waterproof clays under loess loams have high moisture content and plastic consistency. The contact of loams with clays is inclined toward the valleys. When cutting a slope during mine operations, landslides are inevitable.

3.3. Technological schemes of open pit mining

Principal technological schemes for mining the Motronovsko-Annovskyi section are presented in the Fig. 2. The scheme shown in Fig 2, describes mining of overburden with rotary excavators of ER-5000 type with transportation of the rock mass to the external dumps by a system of face-to-face, end and dump conveyors and a console spreader located directly on the dumping dumps. By this scheme, the development of above-ore and ore benches can be carried out by drag-shovels of the ESh-10/50 type, which are placed on the roof of the bench with the loading of rocks into dump trucks. Alternative option of this technological scheme can be realized by a combination of equipment where above-ore bench is developed by a dragline with loading overburden into trucks but ore bench is developed by a dredger in conjunction with a hydraulic monitor and the transportation of rocks through the slurry pipeline directly to the processing plant.

The technological scheme shown in the Fig. 2b differs from the scheme 2a by use of overburden hydraulic excavators of the EG-10 type with the loading of rock mass into dump trucks and its transportation into external dumps. The working of the above-ore and ore benches are identical to the scheme shown in Fig. 2a. A distinctive feature of mining the ore layer is its location below the groundwater level mark, which incurs specific features of mining technology [10].

![Figure 2](image-url)

Figure 2. Technological scheme for the development of the Motronovsko-Annovsky field: a - rotary excavators of ER-5000 type with overburden spreaders; b - hydraulic excavators of the EG-10 type and draglines of the ESh-10/50 type

For excavators, presented in technological schemes, the average parameters of specific pressure on the ground are: for hydraulic excavator EG-10 - 127-205 kPa for static and dynamic loads; for walking dragline excavator ESH 10/50 - 82.84 kPa for static operation and 133.2 kPa – during walking; for the rotary excavator ER-5000 - 140 kPa.

In this paper, a finite element analysis software Phase2 is used as a tool for modeling stability of open pit edges and slopes. The Shear Strength Reduction approach realized in Phase2 allows automatically perform finite element analysis and calculate the SRF (Strength Reduction Factor) for the
selected model, which is equivalent in its meaning to the safety factor (SF).

The algorithm for calculating the safety factor for a slope includes iterative calculation of strength characteristics in all elements of the massif by means of a stepwise loading of the model, as a result of which the stresses in the slope reach the ultimate shear strength until the landslide occurs. The process of SF calculations is repeated until the moment of the slope failure that is graphically visualized as the most probable slip line. If SF > 1, then the slope is in a stable state, and at SF ≤ 1, the landslide processes occur.

The maximum shear deformations in Phase 2 are as follows:

\[ \varepsilon_{\text{max}} = \frac{\varepsilon_1 - \varepsilon_3}{2}, \]  

where \( \varepsilon_1 \) and \( \varepsilon_3 \) are the maximum and minimum major deformations, which correspond to the values of the largest and smallest stresses \( \sigma_1 \) and \( \sigma_3 \) in the case of the flat model.

4. Results

For two possible technological schemes for development of the Motronovsko-Annovskyi section of the Malyshevskoe placer deposit (Fig. 2), geomechanical stability was evaluated.

The Fig. 3 presents results of simulation of the open pit edge and slopes stability for the scheme with rotary excavators ER-5000. For the above-described technology of conducting open mining operations and rock physical-mechanical characteristics, SF = 0.50. The instability of the slopes and development of landslide phenomena is primarily due to the high moisture content in the rocks, as well as to deformation processes at the contacts of layers between water-resistant clays and ore sands. Taking into consideration that the water level is at the mark +101.5 m, the significant shear strains and displacements occurred at two overburden benches. The maximum shear deformations in the slope of the bench with the underlying ore-bearing sands reach the values of \( \varepsilon_{\text{max}} = 1.4 \), which is due to the prevailing effect of horizontal stresses of the massif and their weakening in the direction of the worked out space. The maximum displacements of the massif in the slope of the ore bench reach values of \( U_x = 0.72 \) m. Considering the high watering in the ore layers, it is advisable to extract them by suction dredges.

Fig. 4 presents the results of modeling stability of the working area of the open pit for the scheme using hydraulic excavators EG-10 and draglines ESh-10/50.

For the above-described technology of open mining operations the SF = 0.50 as well. The instability of the open pit edge in this case is due to the extreme values of the slope angles of overburden benches (\( \alpha = 80^\circ \)), which is not a critical factor in the conditions of dynamic stripping operations. However, for the two underlying mining benches, the working angles of the slopes are 30-45\(^\circ\), and under condition of highly watering these geological layers, the slopes are subjected to considerable plastic deformations and landslide processes. Under such conditions, management of mining operations represents a challenge in terms of technological reasonability and work safety.

With the deepening of mining operations, the deformations of slopes and working areas on the benches increase, which is actually observed in the selected sections of the open pit. The underlying sands and blue-green clays are the waterproof rocks and initial cause of geomechanical deformations in the rock benches. Nevertheless, dynamic loads from mining equipment do not have a significant impact on the stability of the rock massif.

According to the described technology, the angles of the slopes of the lower mining bench are accepted up to 45\(^\circ\). However, due to natural humidity of kaolinized sands more than 18% and their wetting from the underlying groundwater, the slopes are subjected to failure, but SF = 0.50.

Analyzing the results of the assessment of geomechanical stability of the pit edge, shown in Fig. 3 and 4, it is obvious that the most susceptible to geomechanical deformations the slopes of the benches from the horizon of...
+130 m to the groundwater level +101.5 m. The watercut of the rock massif is due to the effects of wetting of the overburden rocks by the groundwater level and the amount of atmospheric precipitation.

Using the results of the dependences of physical properties of loams on their moisture content [11], it is possible to determine the slope stability factor of slopes taking into consideration the watering of the rock massif. Thus, the dependence of the angle of internal friction on the moisture content of loams is described by the equation:

$$\varphi = -0.06W_0^2 + 0.12W_0 + 39.86,$$

(2)

where $\varphi$ is the angle of internal friction, degrees; $W_0$ is the humidity, %.

Dependence of the cohesion of rocks in the massif from humidity is described by the equation:

$$C = 0.07W_0^2 - 3.87W_0 + 58.78,$$

(3)

where $C$ is the cohesion value, kPa.

According to dependencies 2 and 3, the physical-mechanical properties of loams and glauconite sands of the Sarmatian layer in the conditions of the watered massif are calculated (Fig. 5). The obtained values are accepted as initial data for modeling stability of the pit edge.

Dependence of the open pit edge stability on the watering rock massif, represented in Fig. 6, has approximately linear character. At the same time, the expected failure of the rock massif ($SF = 1.0$) and development of landslide phenomena occurs at $W_0 = 16\%$. 

Figure 5. Calculated values of cohesion and angle of internal friction for loams and glauconite sands

Figure 4. Results of simulation of the stability of the working pit edge for the scheme with hydraulic excavators and draglines
Thus, the watering of the rock massif is a key factor of instability of the slopes and pit edges. Geomechanical analysis showed the same stability factors for open pit objects using two alternative technological schemes.

5. Conclusions

The open development of watered placer deposits has characteristic features, namely: low bearing capacity of rocks forming mining flank of open pit; difficulty in forming overburden dumps in the worked quarry space in the immediate vicinity of the mining operations; landslide phenomena; availability of aquifers with low water yield; relatively large parameters of the working area due to small angles of slopes and pit edges.

Due to natural humidity of kaolinized sands more than 18% and their wetting from the underlying groundwater, the slopes are subjected to failure. For both technological schemes SF = 0.50. The most susceptible to geomechanical deformations the slopes of the benches from the horizon of +130 m to the groundwater level +101.5 m. The expected failure of the rock massif and development of landslide in the slopes occurs at WO = 16%.

Acknowledgement

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References
Application of Cement-based Sealants for Prevention and Remediation of Environmental Impact of Submarine Resource Mining

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Abstract—Recurrent studies on submarine hydrothermal deposits have been directed toward the practical mine development and the corresponding environmental impacts. Even after the extraction of submarine resources, the original environment shall be maintained in order to prevent harmful effects on the marine ecosystem. Nonetheless, relatively little research about mineral exploitation on the seafloor has been focused on how to prevent its environmental disturbances and rehabilitate the mining site. Authors invented an environment-friendly underwater mining method that aims to control the dispersion of seabed surface sediments and fill the extracted areas by sealing a submerged mine site with an anti-washout cement-based material. This work investigates technological properties and environmental impacts of sealants with different cementitious binders and anti-washout agents on a laboratory scale. Using the remotely operated vehicle Hyper-Dolphin, field trials were conducted in the marine submerged Wakamiko crater. The results indicate that a sealant with particular types of cement and anti-washout agent showed superior anti-washout, filling and hardening properties under submarine hydrothermal conditions. Less dispersion of surface sediments was observed after covering the seafloor surface with the sealant during the field-scale experimental exploitation. Sealants had less negative effects on the seafloor ecosystem.

Keywords: Submarine resource mining, cement, sealant, environmental remediation, hydrothermal deposit

1. Introduction

As cement is one of the most energy- and carbon-efficient of all man-made materials on a volume basis, the use of cement-based materials contributes to development of the sustainable society. Cement is manufactured by using wastes and by-products such as fly ash, plastic waste and surplus soil etc. The Japanese cement industry consumed 28 million tons of wastes and by-products in 2015 in order to produce cement [1]. As a sustainable construction material “cement” has been utilized for cavity supports and backfills of underground mining as well as buildings and bridges.

This work aims to examine and advance the application of cement-based materials for mining methods of submarine hydrothermal deposits. Studies on submarine hydrothermal deposits have been repeatedly directed toward the practical development of mines (i.e. the process of constructing a mining facility and its supporting infrastructure) and the corresponding environmental impacts [2]. Even after the extraction of submarine resources, the original environment shall be maintained to safeguard the habitat for a huge variety of species living in the submarine hydrothermal system and support the biological diversity. Depression contours formed by the extraction could create dysoxic conditions that have harmful effects on the marine ecosystem. Surface sediments in and around submerged mine sites, which occasionally contain mercury and arsenic etc. derived from volcanic activities, can be dispersed by the extraction and bioaccumulate in fish and aquatic invertebrates. Nonetheless, relatively little research about the exploitation of submarine resources has focused on how to prevent its environmental disturbances and rehabilitate the mining site.

Authors invented an environment-friendly underwater mining method that aims to control the dispersion of seabed surface sediments and fill the extracted areas by sealing a submerged mine site with an anti-washout cement-based sealants. Figure 1 illustrates the idea of the underwater mining method. When the underwater mining method is put into practical use, submarine hydrothermal deposits such as rare earth elements that are absorbed on clays can be vacuumed up without dispersing the harmful surface sediments.

![Figure 1. Environment-friendly submarine mining](image-url)
This study measured and investigated technological properties and environmental impacts of sealants with different cementitious binders and anti-washout agents on a laboratory scale. Using the remotely operated vehicle (ROV) Hyper-Dolphin, field trials were conducted with the support of Japan Agency for Marine-Earth Science and Technology (JAMSTEC) in the marine submerged Wakamiko crater.

2. Background

2.1. Underwater concreting
Anti-washout concrete was developed to improve the reliability of concrete placed underwater and has been employed in the construction of bridge foundations and underwater walls etc. Compared to conventional concrete (CVC) using traditional methods such as tremie and concrete pump placing, anti-washout concrete is highly resistant to the washing action of water and rarely separates even when it is dropped into water. Thanks to specific viscosity modifying admixtures called anti-washout agents, e.g. hydroxyethyl cellulose and hydroxypropyl methylcellulose (HPMC), the components of anti-washout concrete never segregate and its flowability is excellent due to the concrete’s low yield value and high viscosity [3].

2.2. Geology of Wakamiko crater
The Wakamiko crater, which contains submerged calderas and two active volcanoes, is located in the eastern portion of the innermost coastal section of Kagoshima bay. The maximum water depth over the crater floor is approximately 200 m. A huge hydrothermal deposit of antimonite with a diameter of 1500 m and a thickness of 5 m, which is estimated 1 million tons of antimony, was discovered in 2011 [4]. Figure 2 shows a calcareous chimney and gravel of antimonite photographed at the Wakamiko crater. A maximum of 80 m-thick layer of surface sediments have accumulated on the antimonite deposit, which contains a maximum of 260 ppm of mercury and 500 ppm of arsenic derived from volcanic activities [5]. For this reason, the deposit must be extracted without dispersing the sediments and having any impacts on fish and aquatic invertebrates.

3. Experimental

3.1. Mix designs
For the purpose of sealing a submerged mine site and filling the extracted area, cement-based sealants need to have excellent self-leveling and anti-washout properties as well as moderate hardening properties. Table 1 shows the mix proportions of the experimental sealants consisting of cementitious binders, aggregate and chemical admixtures. Polycarboxylic ether (PCE) is a dispersant that disperses binder particles and improves fluidity of suspension. HPMC and the tube-like micelle forming agent are anti-washout agents that increase the viscosity of sealants and prevent the component materials from separating in water. To prepare the sealants, 10 kg of all components were mixed with 3 kg of water for 2 min using a hand-held mixer.

<table>
<thead>
<tr>
<th>Cementitious binders</th>
<th>Admixtures</th>
</tr>
</thead>
<tbody>
<tr>
<td>A Portland cement</td>
<td>PCE, HPMC</td>
</tr>
<tr>
<td>B Portland cement, gypsum, calcium aluminate cement</td>
<td>PCE, HPMC, retarders</td>
</tr>
<tr>
<td>C Portland cement</td>
<td>PCE, tube-like micelle forming agent</td>
</tr>
</tbody>
</table>

3.2. Laboratory tests of technological properties
Density, flow value, suspended solids content as an index of anti-washout, setting time and compressive strength of the experimental sealants were measured based on JIS A 1116, EN 12706, JSCE D 104, JIS R 5201 and JSCE F 504, respectively. Plastic viscosity of the sealants was measured under the pressurized condition and was determined from the Bingham plot using the measured share stress–share rate curve. The pressure values were predetermined at 0.1 MPa (standard atmosphere) through to 10 MPa (hydraulic pressure at 1000-m depth).

3.3. Field tests of technological properties
Self-leveling, anti-washout and hardening properties of the experimental sealants were investigated at a field scale through the dive expedition number 1691–1694 of the ROV Hyper-Dolphin of JAMSTEC. Sealants that were loaded onto the ROV were pumped by a peristaltic pump at a discharge rate of 10 L/h and poured into a polyvinyl chloride (PVC) frame set on the seafloor at 200 m below the surface. This 5-cm high PVC frame has a base area of 50 cm². Turbidity and seawater temperature were measured during the pouring of sealants into the frame. Leveling and washout (spreading) of sealants were monitored by the camcorder equipped with the ROV. Figure 3 depicts the ROV with testing equipment.

After the sealants were hardened, sediments under the sealants were vacuumed using a suction sampler in order to observe sealing behavior of the sealants. Specimens for the compressive strength test were casted at the same site where the PVC frames were set (Fig. 4).
3.4. Laboratory tests of environmental impacts

A huge variety of species, e.g. bacterium as a primary producer and gastropoda as a primary consumer etc., live in the seafloor hydrothermal system. There is a similar ecosystem in the coastal sea area. Diatoms, which are primary producers, are fed by gastropods such as *Omphalius rusticus*. On a laboratory scale, this study monitored the diatom breeding performance and population on the surface of hardened sealants A and B, and of CVC with/without seashell repellent applied to these. In addition, the gastropod feeding behavior on the diatoms was monitored. Diatoms were bred for a month on hardened samples (13 cm by 10 cm) in seawater and its breeding success was quantified by the value of chlorophyll a [6]. An *Omphalius rusticus* was placed for 12 h on the hardened samples in seawater where the diatoms were bred, and subsequently the hardened samples were air dried for 12 h to detect visually the trail of *Omphalius rusticus*.

Marine biofouling in the coastal area of Seto Inland Sea was investigated by placing cubic hardened sealants in the littoral zone for 6 months (May–November in 2015).

4. Results and Discussion

4.1. Technological properties of experimental sealants

Table 2 shows density, flow value, suspended solid content, setting time and compressive strength of the experimental sealants. The suspended solid content of ordinary anti-washout concrete is defined as less than 50 mg/L [3]. As the suspended solid content of sealants A–C were less than 50 mg/L, sealants A–C have a good anti-washout characteristic at standard atmosphere. Flow-ability of sealants A and B were higher than that of sealant C. Compared to sealants A and C, setting was earlier and strength development was accelerated in sealant B.

Figure 5 shows the plastic viscosity as a function of the applied pressure. Plastic viscosity of sealants B and C was drastically decreased when 0.5 MPa of pressure was applied and further decreased with the increase of applied pressure. Effects of pressure on plastic viscosity were less pronounced in sealant A. Figure 6 represents the relationship between plastic viscosity and suspended solid content under pressurized conditions. The suspended solid content was increased in direct proportion to the decrease of plastic viscosity due to the applied pressure (the coefficient of determination was 0.7).

<table>
<thead>
<tr>
<th>Sealants</th>
<th>Density (g/cm³)</th>
<th>Flow value (mm)</th>
<th>Suspended solid content (mg/L)</th>
<th>Setting time (min)</th>
<th>3 h strength (N/mm²)</th>
<th>1 d strength (N/mm²)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>1.9</td>
<td>183</td>
<td>9.0</td>
<td>630</td>
<td>0.0</td>
<td>8.0</td>
</tr>
<tr>
<td>B</td>
<td>1.9</td>
<td>185</td>
<td>15.9</td>
<td>220</td>
<td>6.2</td>
<td>19.1</td>
</tr>
<tr>
<td>C</td>
<td>1.9</td>
<td>152</td>
<td>4.2</td>
<td>730</td>
<td>0.0</td>
<td>5.6</td>
</tr>
</tbody>
</table>

Figure 5. Plastic viscosity as a function of the applied pressure

Figure 6. Relationship between viscosity and suspended solid under the pressurized condition

The reason for changes in the plastic viscosity of the pressurized sealants could be explained by the different thickening mechanisms of binders and anti-washout agents. HPMC adsorbs on the binder particles and forms net-like
structures [7], which prevent component materials from separating in water. On the other hand, the tube-like micelle forming agent does not adsorb on the binder particle. When pressure is applied to the sealant with tube-like micelle, the tube-like micelle in the water phase might be agglomerated or its net-like structures might be deformed, and subsequently the anti-washout properties of the sealant degrade. The hydration (reaction) of cementitious binders also causes net-like structures among the binder particles to form and increases the viscosity of sealants [8]. Retarder regulates the hydration rate and delays the formation of the structures at the early hydration period. The net-like structures of sealant B including retarder might be broken easily under the pressurized and shared condition.

4.2. Field tests

Leveling and anti-washout behavior of sealants A–C during the pouring of sealants into the PVC frame were shown in Figs. 7–9, respectively. Sealant A showed superior self-leveling and anti-washout behavior, however, sealants B and C were washed out. The spreading of sealant C was more pronounced. Changes in turbidity and seawater temperature (Fig. 10) that were provoked by the pouring of sealant A were small. These results indicate that the sealant with Portland cement and HPMC (sealant A) showed superior anti-washout and filling properties under submarine hydrothermal conditions.

Table 3 shows the compressive strength of sealants A and B casted and cured under submarine hydrothermal conditions. There was little difference in compressive strength of sealant A cured under laboratory and submarine conditions. As sealant B was washed out during the casting, the compressive strength of the sample cured under submarine conditions was about half of the sample cured in lab.

Figure 11 shows that less dispersion of surface sediments was observed after covering the seafloor surface with sealant A during the field-scale experimental suction.

<table>
<thead>
<tr>
<th>Sealants</th>
<th>Curing conditions</th>
<th>3 h strength</th>
<th>24 h strength</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>Laboratory</td>
<td>0.0</td>
<td>6.5</td>
</tr>
<tr>
<td>A</td>
<td>Submarine</td>
<td>0.0</td>
<td>6.2</td>
</tr>
<tr>
<td>B</td>
<td>Laboratory</td>
<td>5.7</td>
<td>19.6</td>
</tr>
<tr>
<td>B</td>
<td>Submarine</td>
<td>2.9</td>
<td>9.1</td>
</tr>
</tbody>
</table>

4.3. Impacts on seafloor ecosystem

Breeding success of diatoms on the hardened sealants A, B and the CVC with/without repellent is quantified in Table 4. Experimental seawater was obtained from Seto Inland Sea, where 28–120 mg/m² of diatoms naturally live [9]. Except for the case of CVC with repellent, diatoms established on the hardened samples in the lab at the same level as in the natural marine environment.

Figure 12 depicts the trails of *Omphalius rusticus* that were detected in sealants A, B and CVC without repellent.

Figure 13 shows the cube-shaped specimens used for marine biofouling in the littoral zone of Seto Inland Sea. The attachment of *botrylloides violaceus* was observed after one month. Two months later, diatoms covered whole specimens of sealants A and B. Balanomorpha, ascidians, porifera as well as diatoms were attached after 6 months. Figure 14
represents the amounts and species of biofouling after 6 months. Ascidians preferably attached to sealant A and porifera preferably attached to sealant B.

These results indicate that the negative effects of the experimental sealants on seafloor ecosystem could be minimized.

Table 4. Breeding of diatoms on sealants A, B and CVC

<table>
<thead>
<tr>
<th>Sealants</th>
<th>Chlorophyll a concentration (mg/m²)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>59.2</td>
</tr>
<tr>
<td>B</td>
<td>59.5</td>
</tr>
<tr>
<td>CVC without repellent</td>
<td>68.8</td>
</tr>
<tr>
<td>CVC with repellent</td>
<td>24.3</td>
</tr>
</tbody>
</table>

Figure 12. Trails of Omphalius rusticus

Figure 13. Specimens for marine biofouling

Figure 14. Amounts and species of biofouling

5. Conclusions and Perspectives

By examining the application of cement-based sealants for the mining of submarine hydrothermal deposits on both a field scale and a laboratory scale, it was evident that the sealant with Portland cement and HPMC showed superior anti-washout and filling properties and moderate hardening property under the submarine hydrothermal conditions at the Wakamiko crater. Few negative effects of the experimental sealants on the seafloor ecosystem were observed.

There is currently great emphasis on the practical mine development stage. Further research is necessary in order to carry out the next stage for prevention and remediation of environmental damage resulting from submarine resource mining, which would involve field trials in a deep-sea environment (over 1000-m depth) and optimization of the sealing method through simulations on a laboratory scale etc.

References

[9] S. Montani et al., (2003), Seasonal and intertidal patterns of intertidal microphytobenthos in combination with laboratory and areal production estimates, Marine Ecology Progress Series 248, pp. 79–91
Effectiveness of mineral waste management

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Abstract—Mineral waste are the side effect of basic production processes used in the mineral industry. In the case of metals, the flotation waste in ore treatment processes and slags in metallurgical processes are especially large in quantity. Implementation of effective methods for the use of waste is one of the priorities, leading to the measurable economic and environmental effects. Taking the above into consideration, four effectiveness models were analysed in terms of waste management in basic processes as well as their utilization for making new products. The new calculation algorithms for all models mentioned above have been developed.

Keywords: waste minimization; landfill mining; economic assessment.

1. Introduction

The main purpose of waste minimization is creation of technical and technological conditions to process them. Considering present state of technology and science it is possible to find a quite well operating technical system with its lowest impact to environment, but that system will be expensive. Economic aspects of the system are very essential and very often they are the decisive element of selection of a given system. The best technical solutions will not operate properly if the actions are not accepted by society. Thus it is necessary to convict the society what will increase financial outlay to be spent to the whole system. Analysis of all those elements leads to many difficulties [8, 9]. Multi-criteria decisive methods are the mathematic tool enabling to evaluate waste management in many fields and aspects. They enable to perform objective evaluation of waste management system, replacing intuition evaluations of those requiring the expert’s opinions. They also enable to evaluate the system, in conformity with requirements of environment management even in case of goal change or change of conditions in a described region. Decision in the area of waste management system is difficult considering various, often contradictory goals and tasks. Thus the most difficult and the most essential element of calculation is correct and full description of a problem, depending on type of decision that could to be taken.

Despite the emerging global attention towards promoting waste management policies that reduce environmental impacts and conserve natural resources, land filling still remains the dominant waste management practice in many parts of the world [5]. Owing to this situation, environmental burdens are bequeathed to and large amounts of potentially valuable materials are lost for future generations. As a means to undo these adverse effects a process known as landfill mining (LFM) could be implemented provided that economic feasibility is ensured. So far, only a few studies have focused on the economic feasibility of LFM from a private point of view and even less studies have attempted to economically justify the need for LMF projects from a social point of view.

Worldwide, the generation of municipal solid waste (MSW) is increasing and landfills continue to be the dominant method for managing solid waste [15]. Because of inadequate diversion of reusable and recoverable materials, MSW landfills continue to receive significant quantities of recyclable materials, especially metals. The economic value of landfilled metals is significant, fostering interest worldwide in recovering the landfilled metals through mining. However, economically viable landfill mining for metals has been elusive due to multiple barriers including technological challenges and high costs of processing waste.

Open waste dumps in Sri Lanka generate adverse environmental and socio-economic impacts due to inadequate maintenance [6]. The concept of open waste dump mining is suggested in order to minimise the environmental and socio-economic impacts, together with resource recovery. A model based on life cycle assessment and life cycle costing has been used to assess the environmental and
economical feasibility of the suggested *open waste dump mining* concept. However, economic profits can be obtained by adjusting waste transport distances and the price of electricity. The environmental analysis further reveals that the higher global warming potential of open waste dumps can be eliminated to a large extent by applying suggested mining and waste valorisation scenarios.

Waste originating from copper mining create a big problem regarding their quantity and toxicity. Flotation tailing of Polish copper sulphide ores represents more than 94% of the mass of run of mine ore (Table 1). The ore is of sedimentary origin and the tailing contains mainly quartz, dolomite, clay minerals, traces of sulphides and some accessory minerals. Nearly 800 million Mg of copper flotation tailings were up to now deposited in the Legnica-Głogów region and currently is stored about 28 mln Mg per year. It is a source of valuable metals, and new hydrometallurgical technologies are becoming more economical in the recovery of these metals [3, 10, 12].

Table 1. Average chemical content in waste flotation disposal of Polish copper industry [12]

<table>
<thead>
<tr>
<th></th>
<th>Lubin</th>
<th>Polkowice</th>
<th>Rudna</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂, %</td>
<td>57,24</td>
<td>19,67</td>
<td>53,27</td>
</tr>
<tr>
<td>CaO, %</td>
<td>11,87</td>
<td>24,85</td>
<td>13,88</td>
</tr>
<tr>
<td>MgO, %</td>
<td>4,23</td>
<td>6,19</td>
<td>5,25</td>
</tr>
<tr>
<td>Al₂O₃, %</td>
<td>4,17</td>
<td>3,25</td>
<td>3,84</td>
</tr>
<tr>
<td>Cu, %</td>
<td>0,15</td>
<td>0,19</td>
<td>0,21</td>
</tr>
<tr>
<td>Pb, %</td>
<td>0,04</td>
<td>0,02</td>
<td>0,02</td>
</tr>
<tr>
<td>Ag, ppm</td>
<td>10</td>
<td>6</td>
<td>8</td>
</tr>
<tr>
<td>As, ppm</td>
<td>50</td>
<td>30</td>
<td>20</td>
</tr>
<tr>
<td>Co, ppm</td>
<td>52</td>
<td>18</td>
<td>12</td>
</tr>
<tr>
<td>Zn, ppm</td>
<td>90</td>
<td>60</td>
<td>60</td>
</tr>
<tr>
<td>Fe, %</td>
<td>0,82</td>
<td>0,54</td>
<td>0,52</td>
</tr>
<tr>
<td>Na, %</td>
<td>0,23</td>
<td>0,28</td>
<td>0,24</td>
</tr>
<tr>
<td>K, %</td>
<td>1,24</td>
<td>1,18</td>
<td>1,14</td>
</tr>
</tbody>
</table>

Mineral activity has, in a direct or indirect manner, a negative impact on the environment. The process of deposit exploitation and further processing of the extracted raw material is accompanied by waste generation and their disposal on the ground surface or in mine workings. The footprints of this activity are visible in the form of heaps, dumps or ponds. Often, they constitute an unnatural landscape element, usually creating upheavals of various shape and height or artificial water reservoirs. They take up agricultural and forestry land and create a hazard due to the risk of landslide formation. At the same time, they have a negative impact on water and soil. Sometimes, they are still visible even after land reclamation. In the copper industry the waste are produced in the mining processes, and in the ore treatment processes encompassing crushing, grinding and flotation as well as thickening, filtration and drying, of major importance are flotation waste dumped into the so-called settling ponds (basins). Analysing the available worldwide publications about possible applications for flotation waste, we can state that the hitherto studies have confirmed the possibility of their application in ceramics using heat methods. In combination with other components brown and black pigments for ceramic glaze and are obtained. It has also been found that an addition of flotation waste to ceramic mixtures causes reduction of shrinkage in manufactured products and reduction of water absorption by them [13, 14]. Another method for reducing the amounts of the above-mentioned wastes is to use them for production of various abrasive or cutting tools and devices or special-purpose tiles [4]. Interesting results were obtained from the studies on utilizing flotation waste as a source of iron compounds for the production of hydraulic binding agent, in the cement production, in the road construction or backfilling the mined out spaces [1, 2, 11]. Considering the large amount of waste produced during different processes, it is necessary to find a solution for the raw materials treatment with zero waste (Figure 1). The innovative technologies will be helpful for this purpose, leading to recovery all products from raw materials.

Fig. 1. Raw materials economy with zero waste

Many types of waste are also generated in the metallurgical processes. They include: shaft furnace slags, electric furnace slags, gas scrubbing sludges, converter sludges and dusts. Shaft furnace slags are used as an aggregate or surface working reclamation material, and granulated slag is used as an abrasive material and as an addition to hydraulic fill. Lead-bearing dusts and sludges are processed into crude lead and partly stored. Desulphurisation waste are used as a flux in metallurgical furnaces, and the remainder is stored after neutralisation in special chambers within the metallurgical plant areas.
2. Methods

In general, waste that can be effectively utilised are divided into mineral waste that contain useful components and other waste having physical-chemical properties that allow their cost-effective utilisation. Waste dumping usually associated with environmental degradation and considerable financial burden has its reflection in many aspects, immeasurable or defined with costs [7, 16, 17]. The subject of this article is to examine the effectiveness of mineral waste management, and especially those generated in the flotation process. Their quantity is calculated from the following formula:

$$\gamma_o = \frac{\beta - \eta}{\beta - \alpha} \times 100$$

where: $\gamma_o$ – waste yield from 1 Mg of feed, %; $\beta$ – content of useful component in concentrate, %; $\eta$ – content of useful component in feed, %; $\alpha$ – content of useful component in waste, %. Assuming the following data:

$\beta = 30\%$, $\eta = 2\%$, $\alpha = 0.2\%$ we obtain $\gamma_o = 94\%$

The following illustrates the scale of flotation waste. For example, with $6 \times 10^6$ Mg of extracted ore we will generate $6 \times 10^5$ Mg of waste.

Most often, mineral waste can be managed through:

- utilisation in the basic processes, e.g. as a hydraulic fill in mine workings or for new products – for manufacture of prefabricated products;
- technological links with useful component recovery,
- application of investment or non-investment variant, following technological or economical requirements.

The limitation of waste management to useful component recovery only does not solve the problem of their disposal. The essential factor influencing the necessity of seeking other solutions is significant damage caused by them to the natural, land and water environment. Taking damage occurrence possibility and place into consideration the following areas are distinguished: agricultural land, woodland, water resources, parks and greenery, population, council housing, industrial plants, road construction, railway construction, territorial development.

In the case of stating damage it must be estimated, which is a complicated and responsible task. The fair and credible opinion will depend on expert’s knowledge and experience. Among the damage estimation methods, restitution, substitution and index methods can be distinguished.

The basis of the restitution method is an assumption that losses caused by environmental contamination, conversion or pollution are equal to expenditures to be incurred on their liquidation and restoration of environmental balance. Loss estimation consists in calculating the necessary investment expenditures and operating costs of equipment (facilities) used to reduce or eliminate the losses, including the restoration of the environment to its previous state. A disadvantage of this method is that estimation is isolated from the real losses. Furthermore, the restitution is not always feasible or considerably limited, which involves the occurrence of unrepairable damage.

The substitution method is used in the cases where there is loss of a specific element or value of the environment. Then, the costs to acquire the lost element in different place or costs of the construction and operation of facilities able to fulfil the same function as the lost element of the environment. The studies shows that the substitution method also gives estimation results differing from the results of direct calculations.

The index method consists in establishing indexes on the basis of experience and empirical studies and their appropriate generalization. Such experience can be the estimation of losses with the direct calculation method in appropriately selected regions (points) or on trial areas. The results of these calculations constitute a basis for calculating losses in any other place.

Most often, two types of generalization are used: calculation of the average increase of examined loss amount in relation to the increase in contamination and determination of general losses (in the country scale) the unit value (per contaminant unit, per surface unit, per inhabitant) of which constitutes the basis for detailed calculations in the selected areas. Companies make appropriate payments established by the respective state administration bodies for environmental damages caused by waste. Such payments are a considerable factor in the cost-effectiveness account of mineral waste management. The article presents a proposal of the cost-effectiveness account for mineral waste management according to the characteristics given above.

3. Results

The effectiveness of waste minimization was examined in three dimensions: advisability of useful components recovery, management methods and advisability of investment expenditure. To evaluate the cost-effectiveness of waste management, used the indicator Net Present Value (NPV). It is the difference between the present value of cash inflows and the present value of cash outflows. NPV is used in capital budgeting to analyze the profitability of a projected investment or project. A positive net present value indicates that the projected earnings generated by a project or investment (in present dollars) exceeds the anticipated costs (also in present dollars).
Considering the above, four effectiveness models were analyzed:
1) cost-effectiveness of waste management in basic processes, technologically linked with mineral components recovery,
2) cost-effectiveness of waste management in basic processes without mineral components recovery,
3) cost-effectiveness of utilization of waste for making new products, technologically linked with mineral components recovery,
4) cost-effectiveness of utilization of waste for making new products, without mineral components recovery.

The part of the calculation of economic efficiency includes all the models mentioned above, giving the calculation algorithm of each case.

1) **Cost-effectiveness of waste management in basic processes, technologically linked with mineral components recovery**

**Non-investment variant**
The cost-effectiveness of waste management is assessed according to the operating profit criterion, which is described by the following formula:

\[ EBIT = 0.01 \cdot \alpha \cdot \epsilon \cdot Q \left( p - c \right) \pm P \cdot \Delta c_p + Q \cdot c_o \]  

and after making the designation:

\[ z = \frac{Q}{P} \] \hspace{1cm} (1.2)

and:

\[ T = \frac{100}{\text{dep}\%} \] \hspace{1cm} (1.3)

and substituting (1.2) in (1.1) we obtain:

\[ EBIT = 0.01 \cdot \alpha \cdot \epsilon \cdot Q \left( p - c \right) \pm \frac{Q}{z} \Delta c_p + Q \cdot c_o \] \hspace{1cm} (1.4)

hence:

\[ EBIT = Q \left[ 0.01 \cdot \alpha \cdot \epsilon \left( p - c \right) \pm \frac{\Delta c_p}{z} + c_o \right] \] \hspace{1cm} (1.5)

where: \( EBIT \) - operating profit with waste utilization in basic production linked with mineral components recovery, USD/year; \( \alpha \) - content of useful mineral components in waste, %; \( \epsilon \) - yield of mineral components in technological processes specified by decimal number; \( p \) - price of mineral components recovered, USD/Mg; \( c \) - cost of acquiring mineral components from waste, USD/Mg; \( z \) - coefficient specifying the ratio of annual amount of utilized waste to annual basic production; \( Q \) - amount of waste managed, Mg/year; \( P \) - annual basic production, Mg/year; \( \Delta c_p \) - decrease (+ sign) or increase (– sign) in costs of basic production caused by waste management, USD/Mg; \( c_o \) - payment for waste disposal, USD/Mg.

**Investment variant**
The cost-effectiveness of mineral waste management in the investment variant is described by the following formula:

\[ NPV = NCF \left( \frac{1 + WACC}{WACC} \right)^T - IC_1 - IC_2 \] \hspace{1cm} (1.6)

where:

\[ NCF = Q \left[ 0.01 \cdot \alpha \cdot \epsilon \left( p - c + \text{dep} \right) + c_o \pm \frac{\Delta c_p - \Delta \text{dep}_p}{z} \right] \] \hspace{1cm} (1.7)

where: \( NPV \) - updated net value of waste management project, USD; \( NCF \) - annual net cash flow, USD/year; \( T \) - calculated period corresponding to average period of fixed asset depreciation, years; \( WACC \) - annual average cost of capital; \( IC_1 \) - initial investment expenditures for technologies associated with mineral waste recovery, USD; \( IC_2 \) - initial investment expenditures for technologies associated with waste management, USD; \( \text{dep} \) - cost of fixed asset depreciation in basic production following waste management, USD/Mg; \( \Delta \text{dep}_p \) - depreciation cost saving in basic production caused by waste management, USD/Mg of basic production, \( \text{dep}\% \) - share of depreciation costs in total costs encompassing mineral waste recovery and waste management, %.

**Calculation example**
Flotation waste from copper ore treatment have been used for backfilling mine workings, as a substitution of filling sand. Mineral components are recovered from the flotation waste in technological processes before using them as a backfill.

**Data:**
- \( \alpha = 0.20\% \), \( \epsilon = 0.7 \), \( p = 6,000 \) USD/MgCu,
- \( c = 4,000 \) USD/MgCu, \( \text{dep} = 2,000 \) USD/MgCu,
- \( \Delta c_p = 100 \) USD/Mg, \( \Delta \text{dep}_p = 50 \) USD/MgCu,
- \( \text{dep}\% = 20\% \), \( c_o = 75 \) USD/Mg, \( Q = 150,000 \) Mg/year,
- \( P = 100,000 \) MgCu/year, \( WACC = 0.1 \),
- \( IC_1 = 20,000 \) USD, \( IC_2 = 15,000 \) USD, \( z = 1.5 \),
- \( T = 5 \) years

**Non-investment variant:**
\[ EBIT = 21,660,000 \] USD/year

**Investment variant:**
\[ NPV = 29,732,000 \] USD/year

2) **Cost-effectiveness of waste management in basic processes without mineral components recovery**
Based on the formulas (1.5; 1.6; 1.7) with omitting the segment specifying the cost-effectiveness of mineral waste recovery the economic result of mineral waste management can be described by the following formulas:
Non investment variant

\[ EBIT = Q \left( c_o + \frac{\Delta c_p}{z} \right) \]  

(2.1)

Investment variant

\[ NPV = NCF \left[ \frac{1 + WACC}{WACC} \right]^{T_1} - IC_2 \]  

(2.2)

where:

\[ NCF = Q \left( c_o + \frac{\Delta c_p - \Delta dep_{np}}{z} \right) \]  

(2.3)

Calculation example

Data: \( Q = 150,000 \text{ Mg/year}, P = 100,000 \text{ MgCu/year}, c_o = 75 \text{USD/Mg}, \Delta c_p = 100 \text{USD/Mg}, \Delta dep_{np} = 50 \text{USD/MgCu}, WACC = 0.1, IC_1 = 15 \times 10^6 \text{USD}, IC_2 = 15 \times 10^6 \text{USD}, dep\% = 20\%, z = 1.5, T = 5 \text{years} \)

Non investment variant: \( EBIT = 21,250,000 \text{USD/year} \)

Investment variant: \( NCF = 16,250,000 \text{USD/year} \)

NPV = 46,425,000 USD

From the comparison of the calculation examples in paragraphs 1) and 2), it follows, that the waste management without mineral components recovery is more advantageous.

3) Cost-effectiveness of utilization of waste for making new products, technologically linked with mineral components recovery

Non investment variant

The cost-effectiveness of such way of waste management is described by the following formula:

\[ EBIT = 0.01 \cdot \alpha \cdot \varepsilon \cdot Q(p - c) + NP(p_{np} - c_{np}) + c_o \cdot Q \]  

(3.1)

where:

\[ NP = Q \cdot \gamma \text{ hence } \gamma' = \frac{NP}{Q} \]  

(3.2)

After the following formula is obtained:

\[ EBIT = Q \left[ 0.01 \cdot \alpha \cdot \varepsilon (p - c) + \gamma(p_{np} - c_{np}) + c_o \right] \]  

(3.3)

where, the symbols: \( \alpha, \varepsilon, Q, p, c, c_o \) correspond to explanations in formulas (1.5; 1.6; 1.7), and: \( NP \) - quantity of new product made from waste, Mg/year, \( \gamma \) - yield of new product from waste, \( p_{np} \) - price of new product, USD/Mg, \( c_{np} \) - cost of making and selling new product from waste, USD/Mg.

Investment variant

The cost-effectiveness of waste management in such variant is described by the following formulas:

\[ NPV = NCF \left[ \frac{1 + WACC}{WACC} \right]^{T_1} - IC_1 - IC_2 \]  

(3.4)

Designations of symbols as in formulas (1.6; 1.7).

Annual cash flow in such case are described by the following formula:

\[ NCF = Q \left[ 0.01 \cdot \alpha \cdot \varepsilon (p - c + dep) + \gamma(p_{np} - c_{np} + dep_{np}) + c_o \right] \]  

(3.5)

where: \( dep_{np} \) - depreciation cost for making new product, USD/Mg

Calculation example

Flotation waste from copper ore treatment have been used for the manufacture of prefabricated products in combination with mineral components recovery from waste.

Data: \( \alpha = 0.20\%, \varepsilon = 0.7, p = 6,000 \text{ USD/MgCu}, c_o = 4,000 \text{USD/MgCu}, \Delta c_p = 12.5 \text{USD/Mg}, Q = 150,000 \text{Mg/year}, \gamma = 2.0, WACC = 0.1, dep = 2,000 \text{USD/MgCu}, dep_{np} = 5 \text{USD/Mg}, c_{np} = 75 \text{USD/Mg}, IC_1 = 20 \times 10^6 \text{USD}, IC_2 = 7.5 \times 10^6 \text{USD}, dep\% = 20\% \)

Non investment variant: \( EBIT = 19,170,000 \text{USD/year} \)

Investment variant: \( NCF = 21,090,000 \text{USD/year} \)

NPV = 52,220,000 USD

4) Cost-effectiveness of utilization of waste for making new products, without mineral components recovery

Based on the formulas (3.3; 3.4; 3.5) with omitting the segment specifying the cost-effectiveness of mineral waste recovery the economic result of mineral waste management can be described by the following formulas:

Non investment variant

\[ EBIT = Q \left[ \gamma'(p_{np} - c_{np}) + c_o \right] \]  

(4.1)

Investment variant

\[ NPV = NCF \left[ \frac{1 + WACC}{WACC} \right]^{T_1} - IC \]  

(4.2)

where:
Designations of symbols as in formulas (4.2; 4.3) as in formulas (3.1 – 3.5).

**Calculation example**

Data: \( Q = 150,000 \text{ Mg/year} \), \( \gamma = 2.0 \), 
\( p_{np} = 37.5 \text{ USD/Mg} \), \( c_{np} = 12.5 \text{ USD/Mg} \), 
\( d_{np} = 5 \text{ USD/Mg} \), \( c_{c} = 75 \text{ USD/Mg} \), 
\( \text{WACC} = 0.1 \), \( T = 5 \text{ years} \), \( IC = 7,500,000 \text{ USD} \)

**Non investment variant: EBIT = 17,750,000 USD/year**

**Investment variant: NCF = 20,250,000 USD/year**

**NPV = 69,045,000 USD**

From the comparison of the calculation examples in paragraphs 3) and 4) it follows, that the waste management without mineral components recovery from waste is more advantageous.

4. **Discussion**

The management of waste, arising during the process production, is currently one of the priority in the pro-environmental activities of different companies. This is due to the great variation in the types of waste, requiring a special technologies for their use. Waste originating from copper mining create a big problem regarding their quantity and toxicity. However, they are a source of valuable metals, and new hydrometallurgical technologies are becoming more economical in the recovery of these metals. The article presents a general method for examining the effectiveness of waste management in basic waste-generating processes and for making new products. The effectiveness of waste management was examined in three dimensions: advisability of useful components recovery, management methods and advisability of investment expenditure. In the case of the cost-effectiveness of waste management in basic processes, it is more profitable without mineral components recovery. This solution increases the NPV about 56%. In the case of the cost-effectiveness of utilization of waste for making new products is more profitable without mineral components recovery. The difference in the NPV is about 32%. As a result, it is concluded that the waste management is more advantageous without mineral components recovery.

5. **Conclusions**

The appropriate legislation provides an adequate waste management. In Europe the most important document on this issue is the Directive 2008/98/EC. It contains the definitions and concepts concerning the waste management and explains when waste becomes a secondary raw material and how to distinguish between waste and by-products. The directive requires that the waste be managed without endangering human life and without threats to the environment.

Mining activity has, in a direct or indirect manner, a negative impact on the environment. The process of deposit exploitation and further processing of the extracted raw materials is accompanied by waste generation and their disposal on the ground surface or in mine workings. The footprints of this activity are visible in the form of heaps, dumps or ponds. Often, they constitute an unnatural landscape element, usually creating upheavals of various shape and height or artificial water reservoirs.

Technological processes in the non-ferrous metal industry are accompanied by mineral waste. Typical mineral waste include in the copper industry are flotation waste and copper slag. The problem for the manufacturer is their large quantity and neutralisation method, for example through appropriate deposition or use for other purposes. Waste disposal is usually associated with the degradation of natural environment in many dimensions, and has its reflection in the cost burden. To compensate for that, companies make certain payments the amount of which is determined by the state administration bodies. In order to minimize the harmful effect of wastes on the environment efforts are made to manage them, e.g. by treating them as recyclable material or feed to other industrial processes and/or reclamation of the areas where they are deposited.

Waste management can be technologically linked with the recovery of mineral components contained in waste. When it is advantageous to recover mineral waste without managing (reducing amount) of wastes, then such waste dumps are anthropogenic in character. In practice, however, waste are usually managed without the recovery of mineral components contained in them.

**References**


MPES2017

Session 11 – Mine Production II

09:00-11:45

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In-stope pillar scaling and fracturing in Southern African deep level gold mines

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Abstract— Pillar support is the most common natural support system which is applied in most of the deep to ultra-deep gold mines, as a way of preventing the fall of ground and maintain the stability of the rock mass. Deep level gold mines are currently operating at a depth of 2.7km to 4km from the ground surface with the use of rectangular yield pillars, 6m wide, 15m long and 5.5m high. In-stope pillar fracturing and scaling is one of the common problems faced at deep level gold mines. This paper investigates the general and technical behavior of the in-stope pillar, taking into consideration; fracture frequencies within the in-stope pillar, scaling of the in-stope pillar, behavior of the in-stope pillar under different micro seismic events and ground closure at the back areas and closer to the stope faces. The investigation involves the analyses of in-stope pillar fracturing, scaling and ground closure through the use of underground monitoring instrumentation which includes; borehole camera, extensometer, Electronic Monitoring Cable Anchor Device (EMCD), Falls of Ground Light (FOG light) and closure meters. Abaqus Explicit software was used to calculate the ground closure and \(\sigma_1\) along the boundaries of the in-stope pillar. The results of the borehole camera indicated that most of the in-stope pillars were adequately fractured to the core, but the scaling of the pillar was found to be extensive in the back area (30m and above, from the face), while occurring gradually towards the face (10m and less to the face). The closure monitoring systems indicated ground closure ranging from 10mm to 500mm. Based on the results calculated by the model, the ground closure were found to range from 300mm to 600mm. \(\sigma_1\) was found to range from 270MPa to 400MPa. The work presented in this paper is part of a Masters of Science in Mining Engineering by research at the University of Witwatersrand Johannesburg, School of Mining Engineering.

Keywords: pillar scaling, pillar fracturing, numerical modeling, and seismic events.

1. Introduction
As mining depth increases, the stress levels ahead of the mining faces increases rapidly, compromising the stability of the tunnels and intersections of the excavations [1, 5, 6, 7, 8, 10 and 13]. These extreme high stresses increase the probability of face bursting, pillar bursting and pillar scaling [12]. Accurate pillar design is the key in maintaining stability of excavations [2], especially in deep level gold mines where pillar scaling becomes more pronounced with depth. The behavior of the pillars under different scaling conditions needs to be understood in a quest to stabilize the pillars. The technical and general behaviors of deep level gold mining in-stope pillars undergoing scaling was evaluated in this study. Aspects evaluated include fracture frequencies within the in-stope pillar, scaling of the in-stope pillar, ground closure and the impact of extracting several long-hole stopes on the in-stope pillar behavior. To aid the evaluation, data related to fracture frequency, rate of scaling, ground closure and observations on the in-stope pillar after experiencing a high seismic event was collected over a period of 13 months, thereby affording a statistically representative database. Numerical modeling of the stress and ground closure was simulated.

2. Study approach
2.1. Analysis of in-stope pillar scaling and fracturing
The analysis of the in-stope pillar scaling and fracturing aims to show the behavior of in-stope pillars in deep to ultra-deep gold mines and indicate factors that influence their behavior. Esterhuizen et al [3] stated that rock fracturing is a useful indicator of pillar instability prior to failure. The evaluation of in-stope pillar scaling and fracturing is based on the extent of fracturing within the in-stope pillar, the scaling of pillar as mining progress, the behavior of the in-stope pillar when huge energy is released from a blast (blasting long hole). The scenarios in Table I show the extent of fracturing on in-stope pillars. A borehole camera (rock vision) was used to analyze the extent of fracturing within the in-stope pillar, a disto-meter was used to measure the variation on the in-stope pillar width, with the purpose of understanding the rate of scaling. Ground closure measurements were achieved using installed ground closure monitoring systems. Seismic monitoring was also used in order to detect seismic events that occurred within the sections and their influence on the behavior of the in-stope pillar.

3. Results and Discussion
3.1. Fracture frequency
Table 1 illustrates fracture frequency analysis, seventy-one holes from different sections were analyzed using borehole camera to understand the frequency of fracturing within the in-stope pillars. The dimension of the in-stope pillar were as follows; 6m wide, 15m long and 5.5m height. The quantity of fracturing was measured and grouped based on a standard designed by the author (see Figure 1). From Table 1, it was noted that most of the in-stope pillars were highly fractured to
the core of the pillars, with 62.3% of the holes consisting of a fracture quantity of more than 20 for the 0m to 3m range. Most of the in-stope pillars were designed along major and minor geological structures which are seismically active. However, most of these in-stope pillars were found to be highly fractured within short duration.

In some scenario 22.6% of the in-stope pillar holes show fracturing between 10 and 20. This was commonly experienced within the holes drilled closer to the face (less than 3m to the face) but the fracturing was found to increase rapidly within the drill hole as the face advances towards 6m from the hole. It was noted that the fracturing decreases when approaching the core of the in-stope pillar. This is clearly shown by Table 1 where 11.1% of fracturing between 5 and 10 per meter was obtained from 71 holes and 4% of less than five fractures per meter were found within 71 holes. Example of one of the analysed borehole is shown in Figure 2.

Table 1. Fracture frequency

<table>
<thead>
<tr>
<th>Fracture Frequency</th>
<th>0-1m</th>
<th>1m-2m</th>
<th>2m-3m</th>
<th>total</th>
<th>Risk Rating</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt;20</td>
<td>65</td>
<td>63</td>
<td>57</td>
<td>185</td>
<td>62.3%</td>
</tr>
<tr>
<td>20 to 10</td>
<td>40</td>
<td>17</td>
<td>10</td>
<td>67</td>
<td>22.6%</td>
</tr>
<tr>
<td>10 to 5</td>
<td>0</td>
<td>11</td>
<td>16</td>
<td>33</td>
<td>11.1%</td>
</tr>
<tr>
<td>&lt;5</td>
<td>0</td>
<td>5</td>
<td>7</td>
<td>12</td>
<td>4.0%</td>
</tr>
<tr>
<td>Total number</td>
<td>297</td>
<td></td>
<td></td>
<td></td>
<td>100%</td>
</tr>
</tbody>
</table>

Figure 1. Fracture frequency rating legend

Figure 2. Fracture frequency template

3.2. In-stope pillar scaling

A study by York and Canbulat [14] shows that the pillar safety factor changes with prolonged pillar edges scaling. The scaling of the in-stope pillar was measured with mining progress. The comparison between the planned widths of the in-stope pillar and actual length was considered with mining progress. It was found that the average in-stope pillar scaling was 3.8m. The examples of in-stope pillar scaling and damage are shown in Figures 3 to 6. It was then noted that most of the in-stope pillars scaled to an extent of 2.2m from their original design. According to Ryder and Jager [9], deep gold mines are found to experience in-stope pillar scaling at high rate due to their k-ratio, which is less or equal to 0.5. This may be attributed to high vertical stress in deep level gold mines which influences the sidewall damage or in stope pillar scaling at high rate.

The structural geology and seismicity of the mine influence the behaviour of the in-stope pillar scaling and fracturing. Deep level gold mines in South Africa, commonly consist of several seismically active geological structures, with large seismic events due to the depth of the mine and poor installation of backfill within mined void which lead to redistribution of stresses along the stopes. These are some of the factors found to influence the scaling and fracturing on the in-stope pillar.
It was also clearly noted that the extensive in-stope pillar scaling occurred after blasting of long-hole stopes which were closer to the development ends and as development stopes approach major geological structure such as major faults, dykes and joints.

### 3.3. Ground closure

The Electronic Monitoring Cable Anchor Device (EMCAD), extensometer, closure meters and Falls of Ground (FOG) light were used to monitor the ground closure at the back area and 30m away from the face. Extensometers were installed on the hanging wall at 90°, 15m and 20m long extensometers were installed along the main access drive. EMCADs were also installed along the access drive, intersections and at the bullnoses of long-hole stopes. Closure meters of about 5.5m height were also installed at the back area and 30m away from the faces. Lastly, FOG light were installed along the main access drive and intersections.

The ground closure monitoring equipment were installed with the aim of identifying the ground closure generated by the extraction of long-hole stoping and its impact on in-stope pillar. The final results were based on the ground closure measurements before extraction of long-hole stopes and after, as well as ground closure during the extraction of mining stopes.

The closure monitoring systems were found to detect the ground closure differently depending on their sensitivity. The extensometer could identify closure ranging from 10mm to 500mm, during the investigation. The closure was found to increase rapidly after extraction of multiple long-hole stopes within the sections. The closure ranged from 200mm to 500mm just after extraction of long-hole stopes. These measurements were only measured using extensometers. At the same period, closure meter measurements were also reviewed and it was found that the closure of between 50mm to 160mm were recorded from different closure meters installed. On the other hand most of the FOG lights showed some indication of ground closure. Lastly EMCAD measurements on strain were also collected, it was found to range from \(-1 \times 10^{-7}\) to \(1.5 \times 10^{-7}\). It was then concluded that rapid blasting and poor installation of backfill
in long-hole stopes had direct impact on the closure of the ground and also influence on in-stope pillar scaling due to redistribution of stress along the in-stope pillars after blasting long-hole stopes.

3.4. Seismic Monitoring and Analysis Strategy at a Deep Level Gold mine
Seismic monitoring forms part of the Deep level Gold mine’s strategy to reduce the risk posed by rock bursts. The following objectives for seismic monitoring were set:

- Quick locations for rescue operations.
- Evaluating seismically hazardous situations related to mining (mining sequences, geometry, pillar design, etc.). This aides in improving safety and in optimising mine designs.
- Back analysis of large events (typically M ≥ 2.5) to assess the cause of the instability.
- Evaluating the seismic hazard associated with geological structures.

3.5. Sensitivity and Location Accuracy of the Seismic Network
The software Egret (TM) was used to determine the system characteristics of seismic networks, through its sensitivity and the location accuracy.

The sensitivity of the network was calculated using the empirical relationship between peak particle velocity (PPV), magnitude (or energy) of the event and hypo-centre distance. In determining the system characteristics, it was assumed that all geophones were functional and working properly. The sensitivity and location errors in XYZ were contoured on a plane dipping at 14° for the mine-wide network. Figure 7 shows that the sensitivity of the seismic network around the current mining areas was better than ML= -1.5 and Figure 8 shows location Accuracy of the mine wide Seismic Network at the Deep level Gold Mine.

Figure 7. Sensitivity of the mine wide Seismic Network at the Deep Level Gold Mine

![Sensitivity of Seismic Network](image1.png)

![Location Accuracy](image2.png)
3.6. Recording and Processing Seismic Data

The seismic system was designed to run continuously and record seismic events whenever they occur. Ground vibrations caused by seismic events were recorded at each underground seismic site and were then sent automatically to the Desktop Run Time System (DRTS). If the DRTS was down when a seismic event occurred, the data was recorded in the seismometer and was, in most cases, retrieved when the DRTS came back on-line. Normally, the DRTS requests seismic triggers, groups them into seismic events and records the information on a hard disc.

Seismic events were then automatically processed, which involved the automatic picking of P- and S-wave arrivals on seismograms and calculation of the location and source parameters. The automatically processed seismograms were then saved to the database. However, the automatic processing was not developed to a level that was reliable and all seismograms of seismic events $ML \geq 0.5$ were manually processed by Seismic Data Processors.

3.7. Comparison of seismic events with the production of the mine

It was noted that large events occurred when the production rate increased. Mine production was found to increase rapidly when long-hole stopes were extracted. Most of the in-stope pillars were found to be affected by either pillar scaling or pillar bursting. A comparison between production and seismic events is shown in Figure 9.

Figure 9. Comparison between production rate and seismic events occurrence

Figure 10 indicates some of the seismic events impact on the in-stope pillar. During the investigation, a variety of seismic events were reported with the largest having magnitude of 3.0, but even a seismic event of magnitude 1.0 was reported and resulted into pillar burst (see Figure 10).
Figure 10. In-stope pillar burst due to seismic event of 1.0 magnitude

The energy released from the events were also taken into consideration as to correlate with the production rate. It was also found that high level of energy was recorded corresponding to high production rate (see Figure 11).

Figure 11. Comparison of energy released from an events to production rate

It was then concluded that in-stope pillar scaling and fracturing rate has correlation with the increase of seismicity and production. Consequently, it was then noted that when production rate increases, there are high chances of experiencing high seismicity and in-stope pillar scaling and fracturing.

The probability of seismic hazards was also generated using historical data through mXrap software. It was found that most of the sections were considered to be prone to seismic events with large magnitudes (see Figure 12). This model located large events within sections where in-stope pillar scaling and fracturing have been experienced at high rate.

Figure 12. Hazards assessments

3.8. Numerical modelling of ground closure

The Abaqus Explicit software was used to simulate ground closure during the investigation. A strain-hardening material model was used through the application of the Hoek-Brown yield criterion and faults were represented using a discontinuum formulation using cohesive finite elements. The material properties used are shown in Table 2.
The results obtained by the model indicated that ground closure range from 300mm to 600mm. It was also found that if the rate of mining remain the same, the mine might expect high ground closure due to insufficient backfill, and rapid blasting of long-hole stopes. However, these could be controlled by backfilling extracted long-hole stopes.

Figure 13 illustrates the ground deformation during de-stress development. The deformation was found to be approximately 300mm. Several factors were also noted to influence the ground closure during de-stress development: the depth at which mining was taking place (+/-3500m), the middling between two de-stress cuts (+/- 15m) and also high vertical stresses.
Figure 13. Deformation during de-stress development

Figure 14 illustrates simulated data for ground deformation using numerical modelling. The deformation condition was simulated when long-hole stoping started at the mine. It was found that the ground closure could be approximately 500mm within de-stress cuts. However, the deformation was estimated to occur during extraction of secondary stoping (long-hole stope) along the strike access drive and main access drive.

Figure 15 illustrates the deformation conditions of the ground during multiple extraction of secondary access within de-stress cuts. The model estimated the closure of approximately 600mm vertical. However, several factors were also stated which might influence the closure. These includes: high release of strain
energy, failure to install backfill, multiple large voids in de-stress cuts which were not backfilled.

Figure 15. Deformation conditions when multiple longhole stope is extracted

It was then concluded that the ground closure in de-stress cuts range from 100mm to 600mm, depending on extraction rate taking place within the cut. The ground closure was also found to influence the scaling and fracturing of the in-stope pillar due to redistribution of stress around the in-stope pillar.

3.9. Numerical Modelling to determine stress magnitudes

In this study, numerical analysis was carried out to simulate the magnitudes of stress ahead of the mining faces using Abaqus Explicit software. Only $\sigma_1$ was generated and it was found that $\sigma_1$ ahead of the face was between 150MPa to 200MPa (Figures 16 and 17).

The stress levels from numerical modelling results show that the stresses ahead of the face are about 200MPa. This is higher than the unconfined compressive strength (UCS) of the quartzite in Witwatersrand basin, which have been found to be 180MPa [8]. This magnitude of stress is likely to generate hazardous rockbursts on daily basis. The rapid extraction of long-hole stopes without backfilling will lead to higher stresses ahead of the faces.

Figure 16. Modelled induced stresses ahead of distress faces

Figure 17. Modelled induced stresses ahead of distress faces and along the yield pillar
4. Conclusions
Fracture frequency assessment conducted along different in-stope pillars indicated high fracture at 10m from the mining faces, while moderate fractures were 3m and less from the mining faces. There were several factors found to influence the fracturing of the in-stope pillars, these include depth of mining, poor installation of backfill within the mined stopes, extraction of multiple long-hole stopes located closer to the mining faces. The ground closure measurements indicated 200mm to 600mm closure, however the ground closure was found to increase gradually outward to the mining faces. On the other hand, mining faces which were located closer to longhole stopes were found to experience high ground closure as compare to destress cuts design with no longhole stopes. Poor installation of backfill support was found to influence the ground closure and the scaling of the in-stope pillars. The seismicity monitoring system indicated gradual occurrence of large events along the geological structures and along the in-stope pillars, gradual increase of the events was found to occur where multiple longhole stopes were extracted. On the other hand, mXrap shows high hazard assessments along the regional pillars, geological structures, and in-stope pillars. The abaqus explicit numerical modelling software simulated the ground closure that ranges between 100mm to 600mm, the was model found to be realistic, it correlated with the ground closure results obtain from other ground closure instrumentation used such as extensometers and FOG lights. High stress of 200MPa was simulated by the model, however, the high stress influences rapid occurrence of in-stope pillar scaling and fracturing. The maximum strength of the rock in the mine was found to be less than the stress generated due to rockburst caused by cracks growing near free surface. Rockbursts and Seismicity in Mines. Proceedings of the 3rd international symposium, Kingston, Ontario, 16-18 August1993. Young, R.P. (ed.). Balkema, Rotterdam. pp. 169-174.

References
Evaluating the effectiveness of the Mine Design Laboratory

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Abstract - To improve student success rates without compromising the quality in the undergraduate program of the School of Mining Engineering at the University of the Witwatersrand (Wits Mining), a new Mine Design Laboratory (MDL) was established in 2010. The MDL became operational in 2011. Although the number of undergraduate students in the Wits Mining program has increased by more than 21% from 2011 to 2015, this increase did not affect the results adversely as the success rate in most subjects taught in the facility remained over 91%. Details of the MDL pertinent to its establishment such as the features, cost and hardware and software compliments of the facility was published in the Journal of the Southern African Institute of Mining and Metallurgy in 2012. After five years of its completion, this paper re-evaluates the MDL’s effectiveness in terms of teaching and learning by Wits Mining students. Furthermore, it shows how important it is to have proper facilities to improve the teaching of large sized classes. Through this initiative, Wits Mining has taken an important step that will contribute addressing the shortage of skills in the mining industry in South Africa.

Keywords: MDL, Wits Mining facilities.

1. Introduction

South Africa is a mineral rich country and hosts the world class ore deposits. The South African mining industry continues to be a major source of employment and contributes largely to the South African economy. Contrary to the unemployment statistics in South Africa, there is a shortage of skills in many of the technical fields in the mining industry [1]. It is important to address this issue as it has adverse effects on the future of the mining industry [2].

Wits Mining is one of the world’s top mining engineering schools. It has an expansive academic programme and also has one of the highest growth rates amongst the engineering schools in the faculty of engineering and built environment at the University [3]. According to the 2016 Quacquarelli Symonds (QS) Ltd World Rankings by Subject (area of study), Wits Mining ranks in the Top 100 of 403 Mining/Mineral Schools worldwide and No. 1 in South Africa for Mining/Mineral [4]. Table 1 shows the breakdown of the number of students from 2011 to 2015 for the different programmes. Therefore, Wits Mining is in a strong position to make a significant contribution to the provision of qualified mining engineers to the mining industry, and consequently, must ensure that its students are taught the appropriate subject matter at a consistently high level.

Table 1 - Wits Mining student numbers (2011–2015)

<table>
<thead>
<tr>
<th>Year</th>
<th>1st</th>
<th>2nd</th>
<th>3rd</th>
<th>4th</th>
<th>Graduates</th>
<th>Total UG</th>
<th>GDE</th>
<th>MEng</th>
<th>MSc</th>
<th>PhD</th>
<th>Total PG &amp; PG</th>
<th>Total UG &amp; PG</th>
<th>MRM &amp; MP Cert</th>
</tr>
</thead>
<tbody>
<tr>
<td>2011</td>
<td>233</td>
<td>96</td>
<td>82</td>
<td>62</td>
<td>62</td>
<td>473</td>
<td>137</td>
<td>32</td>
<td>34</td>
<td>19</td>
<td>222</td>
<td>695</td>
<td>46</td>
</tr>
<tr>
<td>2012</td>
<td>234</td>
<td>143</td>
<td>80</td>
<td>74</td>
<td>70</td>
<td>631</td>
<td>102</td>
<td>15</td>
<td>42</td>
<td>14</td>
<td>173</td>
<td>704</td>
<td>59</td>
</tr>
<tr>
<td>2013</td>
<td>302</td>
<td>124</td>
<td>142</td>
<td>66</td>
<td>53</td>
<td>634</td>
<td>90</td>
<td>15</td>
<td>70</td>
<td>16</td>
<td>191</td>
<td>825</td>
<td>77</td>
</tr>
<tr>
<td>2014</td>
<td>212</td>
<td>139</td>
<td>148</td>
<td>88</td>
<td>72</td>
<td>587</td>
<td>38</td>
<td>3</td>
<td>112</td>
<td>22</td>
<td>175</td>
<td>762</td>
<td>76</td>
</tr>
<tr>
<td>2015</td>
<td>231</td>
<td>144</td>
<td>162</td>
<td>106</td>
<td>83</td>
<td>643</td>
<td>3</td>
<td>2</td>
<td>176</td>
<td>20</td>
<td>201</td>
<td>844</td>
<td>25</td>
</tr>
</tbody>
</table>

The throughput of students and success rates of any academic establishment are constantly under great scrutiny and analysis. The challenge of improving the throughput of students in all years of study...
formed part of the 5-year strategic plan endorsed previously. In trying to improve the throughput of students, the staff to student ratios have been considered as a contributing factor in this regard. The school has the objective of reducing student staff ratios to 1 staff member for 25 students as part of the strategic plan [2]. This will allow for better student-staff interaction and relations. However, as it can be seen in Table 2, the student to staff ratio from 2011 to 2015 averaged at 34 which is an indication that previously set objective was not achieved and it requires further intervention.

Table 2 - Wits Mining student to staff ratio (2011–2015)

<table>
<thead>
<tr>
<th>Year</th>
<th>Full-time Academic staff</th>
<th>Total Students</th>
<th>Student: Staff Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>2011</td>
<td>21</td>
<td>695</td>
<td>33</td>
</tr>
<tr>
<td>2012</td>
<td>20</td>
<td>704</td>
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<tr>
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<td>22</td>
<td>825</td>
<td>37</td>
</tr>
<tr>
<td>2014</td>
<td>24</td>
<td>762</td>
<td>32</td>
</tr>
<tr>
<td>2015</td>
<td>25</td>
<td>844</td>
<td>33</td>
</tr>
</tbody>
</table>

2. MDL

Mine design is an important area of speciality in the mining engineering curriculum. It is a core aspect that encompasses various disciplines such as the technical, financial, environmental, health and safety concerns in a mining venture. In modern mining, it is important that mining engineers are equipped with adequate computer skills that are essential in mine design. This is due to various software being used to convert data obtained in basic exploration into valuable mining assets [2]. It is thus fitting that the Wits Mining provides a platform that will equip their graduates with skills of this nature. The specialized MDL has been established to educate and train mining engineering students in this regard. The two primary aims of establishing this facility are:

- To provide a facility that would not only allow provision of mining-specific computer software, but also make mining engineers proficient in the use of these software applications.
- To enable the ratio of students to computer to 2:1 to ensure increased throughput and success rates of students [2].

The teaching facility has been equipped with the appropriate hardware and software for mine design and relevant mining-related work. Table 3 indicates the courses taught in this venue for each respective year.

Table 3 - Wits Mining computer-related courses for undergraduate programme

<table>
<thead>
<tr>
<th>Course Code</th>
<th>Course Name</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>MINN 1001</td>
<td>Computer Skills</td>
<td>1st year - 1st Block</td>
</tr>
<tr>
<td>MINN 2000</td>
<td>Computer Application in Mining</td>
<td>2nd year - Full Year</td>
</tr>
<tr>
<td>MINN 3003</td>
<td>Technical Valuation</td>
<td>3rd year - 1st Semester</td>
</tr>
<tr>
<td>MINN 3004</td>
<td>Computerised Mine Design</td>
<td>3rd year - 2nd Semester</td>
</tr>
<tr>
<td>MINN 4005</td>
<td>Financial Valuation</td>
<td>4th year - 1st Semester</td>
</tr>
<tr>
<td>MINN 4006</td>
<td>Mine Design</td>
<td>4th year - 2nd Semester</td>
</tr>
<tr>
<td>MINN 4009</td>
<td>Surface Mining</td>
<td>4th year - 2nd Semester</td>
</tr>
</tbody>
</table>

The new MDL was established in 2010 and it became fully operational in 2011. The MDL is equipped with 100 high-performance desktop PCs which covers just over 550 m² area. The MDL has four data projectors and 32 ceiling speakers strategically positioned in the venue. It also has a document camera, which eliminates the need for a white-board. It can accommodate up to 198 students at a time. Since its establishment, the MDL played a significant role when it comes to improving the success rate in all the subjects taught in this facility. Although the number of undergraduate students in the Wits Mining programme has increased by more than 21% from 2011 to 2015, this increase did not affect the results adversely as the success rate in most subjects taught in the facility remained over
90%. Details of the MDL pertinent to its establishment such as the features, cost and hardware and software compliments of the facility was published in the *Journal of the Southern African Institute of Mining and Metallurgy* in 2012. After five years of its completion, this paper re-evaluates the MDL’s effectiveness in terms of teaching and learning by Wits Mining students. Table 4 shows the main software complements in the MDL. Section 3 discusses the outcome of the computer-related subjects taught in the new MDL from 2011 to 2015.

**Table 4 - MDL software list**

<table>
<thead>
<tr>
<th>Software</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>MS Office</td>
<td>Productivity Software [5].</td>
</tr>
<tr>
<td>Microstation</td>
<td>Market-leading software for engineering design [6].</td>
</tr>
<tr>
<td>RocScience</td>
<td>State of the art specialized rock engineering software [7].</td>
</tr>
<tr>
<td>Surfer</td>
<td>Powerful contouring, gridding and 3D Surface Mapping Software [8].</td>
</tr>
<tr>
<td>Surpac</td>
<td>Surpac is a geology and mine planning Software. Efficient, easy and accurate, powerful 3D graphics, work flow automation [9].</td>
</tr>
<tr>
<td>Whittle</td>
<td>Gemcom Whittle is the strategic mine planning software needed to determine and optimize the economics of open pit mining projects [10].</td>
</tr>
<tr>
<td>MineSched</td>
<td>Geovia MineSched is an innovative scheduling software that allows one to maximise productivity and profit [12].</td>
</tr>
</tbody>
</table>

### 3. Discussion and Analysis

This section compiles the outcome of the computer-related subjects taught in the new MDL from 2011 to 2015. The results indicate the number of students, course success rates, and the average mark for each course. Table 5 shows the number of students who attended various computer-related subjects at the MDL from 2011 to 2015.

**Table 5 - Student count for courses taught in the MDL (2011-2015)**

<table>
<thead>
<tr>
<th>Course Code</th>
<th>Course Name</th>
<th>Number of Students</th>
<th>2011</th>
<th>2012</th>
<th>2013</th>
<th>2014</th>
<th>2015</th>
<th>Average</th>
</tr>
</thead>
<tbody>
<tr>
<td>MINN 1001</td>
<td>Computer Skills</td>
<td></td>
<td>196</td>
<td>198</td>
<td>240</td>
<td>150</td>
<td>202</td>
<td>197</td>
</tr>
<tr>
<td>MINN 2000</td>
<td>Computer Application in Mining</td>
<td></td>
<td>81</td>
<td>121</td>
<td>116</td>
<td>129</td>
<td>114</td>
<td>112</td>
</tr>
<tr>
<td>MINN 3003</td>
<td>Technical Valuation</td>
<td></td>
<td>66</td>
<td>73</td>
<td>127</td>
<td>107</td>
<td>125</td>
<td>107</td>
</tr>
<tr>
<td>MINN 3004</td>
<td>Computerised Mine Design</td>
<td></td>
<td>61</td>
<td>69</td>
<td>122</td>
<td>102</td>
<td>122</td>
<td>109</td>
</tr>
<tr>
<td>MINN 4005</td>
<td>Financial Valuation</td>
<td></td>
<td>66</td>
<td>67</td>
<td>68</td>
<td>100</td>
<td>91</td>
<td>78</td>
</tr>
<tr>
<td>MINN 4006</td>
<td>Mine Design</td>
<td></td>
<td>64</td>
<td>65</td>
<td>64</td>
<td>80</td>
<td>94</td>
<td>78</td>
</tr>
<tr>
<td>MINN 4009</td>
<td>Surface Mining</td>
<td></td>
<td>67</td>
<td>70</td>
<td>65</td>
<td>86</td>
<td>93</td>
<td>76</td>
</tr>
</tbody>
</table>

As it can be seen in Table 5, there is a larger number of students in the first year course due to the high initial intake of students for the programme. Furthermore, the computer to student ratio remains either at 2 or below 2 for all the courses. If this is not possible, Wits Mining policy states that the class must be divided into two groups, each group attending in different time slots to maintain the computer to student ratio to 1 to 2. This is ideal in aiding towards higher success rates and consistent with one of the primary aims of the establishment of this facility. It further shows that the facility is accommodating of the number of students using it. Table 6 compares the success rates for courses taught in the MDL from 2011 to 2015.

Table 6 shows an average increase in the success rates of MINN 1001 which averages at 75% and MINN 4009 which averages at 92%. This could be the direct result of new and improved productivity software and mine planning software used in these courses respectively and the provision of updated...
recommended books that allow an entry level computer user to equip themselves with the relevant skills. An average decrease is experienced with the success rates of MINN 2000, MINN 3003, MINN 3004, MINN 4005 and MINN 4006. This can primarily be due to the complexity of the specialized software as it is further developed and updated. Furthermore, the change in teaching styles of different lecturers as they change within the school and the course content applied, could largely contribute to the challenges faced by the students participating in these courses. However, the average decrease in the success rates of these courses ranges between 1% and 6% which may be regarded as insignificant considering the number of students registered for each course. This range also highlights that the cause of this difference may not be facility or course related but may be due to individual student negligence.

The data in Table 6 demonstrates quite clearly the benefits that the new MDL has provided to the students and Wits Mining, as the average success rate remains over 91% for all of the courses except MINN 1001 and MINN 3003 which averaged 75% and 89% respectively over the five years under consideration. However, the MINN 1001 course showed over 17% of improvement when the 2011 and the average figures were compared. Nevertheless, the very heartening result is that the success rates have remained over 91% for the remaining courses. This success is a clear demonstration that the throughput problems associated with teaching and learning in large sized classes can partially be addressed if students and lecturers have access to appropriate facilities. Figure 1 shows the average success rate for the courses taught in the MDL from 2011 to 2015.

Table 6 - Success rates for the courses taught in the MDL (2011-2015)

<table>
<thead>
<tr>
<th>Course Code</th>
<th>Course Name</th>
<th>2011</th>
<th>2012</th>
<th>2013</th>
<th>2014</th>
<th>2015</th>
<th>Average</th>
</tr>
</thead>
<tbody>
<tr>
<td>MINN 1001</td>
<td>Computer Skills</td>
<td>64%</td>
<td>80%</td>
<td>75%</td>
<td>75%</td>
<td>79%</td>
<td>75%</td>
</tr>
<tr>
<td>MINN 2000</td>
<td>Computer Application in Mining</td>
<td>100%</td>
<td>99%</td>
<td>97%</td>
<td>100%</td>
<td>97%</td>
<td>99%</td>
</tr>
<tr>
<td>MINN 3003</td>
<td>Technical Valuation</td>
<td>95%</td>
<td>93%</td>
<td>91%</td>
<td>80%</td>
<td>83%</td>
<td>89%</td>
</tr>
<tr>
<td>MINN 3004</td>
<td>Computerised Mine Design</td>
<td>100%</td>
<td>100%</td>
<td>90%</td>
<td>88%</td>
<td>82%</td>
<td>92%</td>
</tr>
<tr>
<td>MINN 4005</td>
<td>Financial Valuation</td>
<td>95%</td>
<td>88%</td>
<td>99%</td>
<td>90%</td>
<td>87%</td>
<td>92%</td>
</tr>
<tr>
<td>MINN 4006</td>
<td>Mine Design</td>
<td>100%</td>
<td>100%</td>
<td>100%</td>
<td>95%</td>
<td>100%</td>
<td>99%</td>
</tr>
<tr>
<td>MINN 4009</td>
<td>Surface Mining</td>
<td>87%</td>
<td>93%</td>
<td>86%</td>
<td>95%</td>
<td>97%</td>
<td>92%</td>
</tr>
</tbody>
</table>

Figure 1. Average success rate for the courses taught in the MDL (2011-2015)
Table 7 compares the average marks for courses taught in the MDL from 2011 to 2015. The overall average mark for all the courses taught in the MDL remains above a pass mark, which indicates that an adequate level of understanding is achieved in these courses. The total average marks for all the courses taught in the MDL in each year drops every year by an average of 2.3% from 2011 to 2014 and then increases by 1% in the year 2015. This can also be attributed, as previously mentioned, to the further development of the specialized software utilized in these courses. The level of difficulty increases as the software allows for more specific uses and tasks, thus not only posing a challenge for the students to learn but also for the lecturers to teach. This may contribute largely to the decrease in average marks obtained in these courses. Figure 2 shows the average marks obtained for the courses taught in the MDL from 2011 to 2015.

<table>
<thead>
<tr>
<th>Course Code</th>
<th>Course Name</th>
<th>Average Marks</th>
<th>Average</th>
</tr>
</thead>
<tbody>
<tr>
<td>MINN 1001</td>
<td>Computer Skills</td>
<td>52% 57% 57% 54% 56%</td>
<td>55%</td>
</tr>
<tr>
<td>MINN 2000</td>
<td>Computer Application in Mining</td>
<td>67% 60% 57% 62% 67%</td>
<td>63%</td>
</tr>
<tr>
<td>MINN 3003</td>
<td>Technical Valuation</td>
<td>68% 68% 63% 57% 59%</td>
<td>63%</td>
</tr>
<tr>
<td>MINN 3004</td>
<td>Computerised Mine Design</td>
<td>88% 74% 68% 68% 64%</td>
<td>72%</td>
</tr>
<tr>
<td>MINN 4005</td>
<td>Financial Valuation</td>
<td>65% 64% 68% 61% 59%</td>
<td>63%</td>
</tr>
<tr>
<td>MINN 4006</td>
<td>Mine Design</td>
<td>65% 61% 64% 59% 59%</td>
<td>62%</td>
</tr>
<tr>
<td>MINN 4009</td>
<td>Surface Mining</td>
<td>61% 59% 59% 61% 62%</td>
<td>60%</td>
</tr>
<tr>
<td><strong>Total average marks for each year</strong></td>
<td></td>
<td>67% 64% 62% 60% 61%</td>
<td></td>
</tr>
</tbody>
</table>

Figure 2: Average marks obtained for the courses taught in the MDL (2011-2015)

4. User Satisfaction Survey

To obtain first hand comments regarding the effectiveness of the MDL, a user satisfaction survey has been conducted on 65% of the fourth year class of 2016, considering the fact that they have made use of the facility most over the last four years. The survey consists of 18 yes/no and descriptive questions as seen in Appendix A. These questions were designed to determine the impact the MDL has had on the students, how much the facilities of the MDL are used, how efficient and effective the facilities are, and to draw on recommendations to improve the use of the MDL. The survey results will
enable the Wits Mining to determine if the facility is operating at its optimum efficiency. The results of the survey is shown in Table 8.

<table>
<thead>
<tr>
<th>Questions</th>
<th>Yes</th>
<th>No</th>
</tr>
</thead>
<tbody>
<tr>
<td>Is the MDL beneficial to your learning experience?</td>
<td>64</td>
<td>8</td>
</tr>
<tr>
<td>Is the desk arrangement comfortable to use?</td>
<td>64</td>
<td>8</td>
</tr>
<tr>
<td>Are the projectors offering clear visuals?</td>
<td>64</td>
<td>8</td>
</tr>
<tr>
<td>In your opinion are the document camera project images more efficient as compared to white and chalk boards?</td>
<td>61</td>
<td>11</td>
</tr>
<tr>
<td>When the mic is made use of by the lecturer is the voice clear and audible?</td>
<td>67</td>
<td>5</td>
</tr>
<tr>
<td>Have you as a student made use of the mics installed on the desks in MDL?</td>
<td>56</td>
<td>16</td>
</tr>
<tr>
<td>In your opinion, do the mics aid in better interaction between you the student and the lecturer?</td>
<td>65</td>
<td>7</td>
</tr>
<tr>
<td>Has the software and internet connection in the MDL assisted you in computer based and aided learning?</td>
<td>67</td>
<td>5</td>
</tr>
<tr>
<td>In your opinion, is the venue suitable for external use?</td>
<td>64</td>
<td>8</td>
</tr>
<tr>
<td>Have you attended any extra-curricular activities in the MDL?</td>
<td>56</td>
<td>16</td>
</tr>
<tr>
<td>Have any of the events hosted at the MDL positively influenced or impacted your student life?</td>
<td>60</td>
<td>12</td>
</tr>
</tbody>
</table>

From the results in Table 8, 89% of the final year class believe that the MDL is beneficial to their learning experience. Similarly, 89% of them feels that the desk configuration and arrangement is comfortable and the projectors offer clear visuals, while the remaining students raised the concern that when the venue is used for lectures/presentations the multiple projectors become challenging to use together with a laser pointer. This is because the laser pointer only appears on one screen at a time. Due to this reason, the lecturers were advised to use the mice as a pointer to overcome this problem.

85% of the students support the use of the document camera. 78% of the students has made use of the desk mics and 93% of them agrees that the mics aid in better interaction between the students and the lecturers. 78% of them has attended extra-curricular events at the MDL and 89% of them feel that the venue is suitable for use outside the classroom. Furthermore, they believe that this facility has assisted them with their computer based and aided learning and that the events held in this venue have positively influenced their lives. The students have further expressed that they prefer seating in the first four rows to allow for better visuals as the images projected may be of low quality. Furthermore, interaction with the lecturer is considered easier from that point in the class.

**Conclusion**

The new MDL has already demonstrated its ability to improve teaching and learning with the steady success rates in the new facility. Although the number of undergraduate students in the Wits Mining programme has increased by more than 21% from 2011 to 2015, this increase did not affect the results adversely as the success rate in most subjects taught in the facility remained over 91%. After five years of its completion, the MDL continues its contribution towards achieving its goal and the improved throughput of students in computer-related subjects has been apparent.
References


Appendix A: Mine Design Laboratory (MDL) User Satisfaction Survey

1. Is the MDL beneficial to your learning experience?
   [ ] YES  [ ] NO

2. How has learning in the MDL impacted your educational experience?
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________

3. Which row from the podium do you prefer sitting in and why?
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________

4. How many courses do you attend in the MDL? (i.e.: 4)
   __________________________________________________________

5. Which of the courses that you attend in the MDL do you find most suitable to learn in the MDL?
   __________________________________________________________

6. Is the desk arrangement comfortable to use?
   [ ] YES  [ ] NO

7. If the answer to question 6 is no, explain why?
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________
   __________________________________________________________

8. Are the projectors offering clear visuals? If not, why?
   [ ] YES  [ ] NO

9. When the mic is made use of by the lecturer, is the voice clear and audible?
   [ ] YES  [ ] NO

10. In your opinion are the document camera project images more efficient as compared to white and chalk boards? If not why?
    [ ] YES  [ ] NO

11. Have you as a student made use of the mics installed on the desks in MDL?
    [ ] YES  [ ] NO

12. In your opinion, does the use of the above mentioned mics aid in better interaction between you the student and the lecturer?
    [ ] YES  [ ] NO
13. Has the software and internet connection in the MDL assisted you in computer based and aided learning?

[ ] YES  [ ] NO

14. In your opinion, is the venue suitable for external use? If not, why?

[ ] YES  [ ] NO

________________________________
________________________________
________________________________
________________________________
________________________________

15. Have you attended any extracurricular activities in the MDL?

[ ] YES  [ ] NO

16. Have any of the events hosted at the MDL positively influenced or impacted your student life?

[ ] YES  [ ] NO

17. In your opinion, do the lecturers make adequate use of the MDL facilities when teaching? If not why?

18. Any further comments on the MDL?

Software:

________________________________
________________________________
________________________________
________________________________

Hardware:

________________________________
________________________________
________________________________
________________________________

Lecturer and Student use:

________________________________
________________________________
________________________________
________________________________
Long-term production scheduling in open-pit mines with special emphasis on processing plant feed constraint
(Case study: Saheb Saghez iron ore mine of Iran)

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2 and 3Department of Mining Engineering, Qaemshahr Branch, Islamic Azad University, Qaemshahr, Iran.

Abstract—The issue of production planning optimization (conceptual, long-term, mid-term, short-term and daily schedules) in open pit mines is to find optimum sequence of extracting blocks from ore zone during mining cycle. In this regard, maximizing the Net present value (NPV) and minimizing required energy, especially in processing plants, play major role in productivity and effectiveness of mining operation. The role of long-term production planning optimization is very sensitive for the mine and will reduce investment risks, leading to a capital return with a logical rate in a short period. In this paper, the long-term production planning is optimized for Saheb Saghez iron ore mine located in Kurdistan Province of Iran. Considering the block modelling of ore resources based on gathered data, the economic model was first prepared. In this regard, sensitivity analysis was carried out on the ultimate pit slope for achieved economic model and 42 degree slope was selected as the appropriate and optimized slope for mine walls. Then, the final working area was determined according to the data. In the following, the geometric parameters of excavation area were estimated and the final area was designed using Studio 3 software. To optimize the production planning aiming to meet the required energy for the primary crusher, push-backs were designed to obtain proper production planning. Optimized production planning is then presented according to the objective constraints analyzed by NPV Scheduler program. Results showed that there are 864 thousand tons of economic and minable reserves with average grade of 40 percent in the final extraction area. Accordingly, the production planning, achieved using NPVS, the 200 thousand tones annual extraction of minerals to supply for the primary crusher with average grade of 40 percent, less than 1 percent sulfur for a 5-year mining period.

Keywords: Long-term production planning, Optimization, Saheb Saghez iron ore, NPV scheduler, Primary crusher.

1. Introduction
The problem of optimizing production scheduling in open-pit mines includes finding an order to extract blocks from the pit inside during the mining life and in order to achieve a special objective, so that net present value (NPV) of total mining operation is maximized. Optimizing the production scheduling of a mine is a multi-field process that needs data collection from different fields that their quantity and quality depend on the scheduling type (conceptual, long-term, mid-term, short-term and daily scheduling). Given that readily available reserves are exhausting, remaining reserves—will be deeper, mining conditions will be more complicated than before and the investment volume to run an open-pit mine will be manifold of the investment volume for an open-pit mine in the past years. Having a detailed planning that is also consistent with the actual conditions seems necessary. Here, the role of long-term mine production scheduling is highly sensitive, reduces the investment risk and allows return on investment with reasonable rates in a shorter period.

When designing open-pit mines, first the ultimate pit limit and then production scheduling are determined. To do this, according to economic block model, the order to remove waste blocks and ore is according to the slope limit, mining equipment capacity, processing plant feed capacity and other limitations so that the maximum profit should be obtained from the operation. Optimizing production scheduling in open-pit mines includes finding an order to extract blocks from the pit inside during the mining life so that net present value of total mining operation is maximized.

With the development of computer science and understanding parameters affecting open-pit mines’ production scheduling since 1965 with the advancement in math-based methods, engineers can more accurately design the mines. Optimization techniques particularly operations research (OR) methods are considered as the most important factor in the development of
algorithms mines' programming. The most important operation research methods are described below. Mixed integer programming method has been presented by Gershon [10], Dagdelen [5], Ramazan and Dimitrakopoulos [12]. Dynamic programming method has been presented by Dowd and Onur [7], linear programming method has been presented by Gershon [11], integer programming method has been presented by Kim and branch and bound method has been presented by Caccetta and Hill [4]. Different methods of artificial intelligence have also been presented by Tolowinski and Underwood, Elveli [8], Asgari-Nasab et al. [3].

In this paper, optimized long-term production scheduling for an iron ore mine in an open-pit mining method in order to produce input feed the primary crusher has been obtained. So, using craftiness block model and technical and economic parameters, the ultimate limit and mine optimal pit have been determined in software NPVS, then the implemented pit was designed in software Studio 3 and appropriate number of push backs has been obtained. Finally, long-term production scheduling for the ultimate limit and with respect to the obtained push backs has been optimized and presented to produce the primary crusher input feed.

2. Material and Methods

In this study, the method is in a way that first the mine saving block model will be optimized in software optimizign NPV Scheduler according to assumed technical and economic parameters to produce the primary crusher input feed and accordingly the ultimate pit limit is determined, Then, considering appropriate technical and geometry cases, the extracted limit will be designed by software Studio 3. Finally, production scheduling operation is presented by software NPVS based on the implemented extracted limit prepared aimed to produce input feed of the primary crusher. In order to provide the production scheduling first extracted push backs will be designed as the guide for the production scheduler based on the design optimization objective.

3. Open-pit mining method production scheduling models

There are various methods to solve long-term production scheduling. Each of these methods to solve the problem of long-term production scheduling, use one of the following strategies:

1) First, the ultimate pit is determined and then the production scheduling is obtained using mathematical programming and to maximize the net present value. In large pits, after determination of the ultimate pit, some push backs are obtained. Then, mining stages within the push backs is planned using mathematical techniques. Most of algorithms referred in this chapter use this strategy.

2) The ultimate pit and the production scheduling are simultaneously determined. [8,9,13] and the algorithm [6] use this strategy.

Uncertainty of cut-off grade is a very important topic that can be used in the optimization of long-term production scheduling. For considering the considerations reduces the difference between what is mathematically optimal and what is obtained in practice, so the project net present value will be increased. In a model of long-term production scheduling, the block destination and mining order should be determined at the same time.

4. Long-term production scheduling

The mine long term production scheduling is to determine the time and sequence of extracting ore and waste blocks from different points of a mine so that considering a variety of existing operation limitations, the highest economic value is obtained for the mine. Provide the program is very important for mining engineers.

Four main parameters affecting long term mines' scheduling are shown as a cycle in (Fig. 1) that interact with each other. These four parameters are the ultimate pit, annual production scheduling, costs and grade.

![Figure 1 - The variables related to production scheduling of the open-pit mining method](image-url)
(Fig. 1) indicates that in order to determine ore expansion as well as the pit ultimate limit, cut-off grade should be specified. After determining the ultimate pit limit, the mine production scheduling is determined.

As seen in this cycle, each variable values cannot be achieved unless the previous variable values has been obtained. This process is a multivariable optimization process and needs simultaneous solving.

4.1. Modeling and design stages

Stages to get optimal production scheduling is as follows:
1. Log deposit model geological saving model in optimization software.
2. Prepare deposit economic model based on economic parameters.
3. Determine the optimal limit of open-pit mining using software NPVS.
4. Provide the ultimate pit design using technical parameters and utilization of optimal mining limit.
5. Provide and determine the number of push backs.
6. Production scheduling within the implemented pit using software NPVS and dynamic programming method.

In short, the above stages are shown completely in Figure 3.

5. Introducing used software

5.1. Software Studio 3

For the first time software Studio 3 was provided in 1996 by Tolowinski and Underwood and then in 1997 its new version was presented. Software can use the parametric concept of Lerch and Grossman determine also nested pits, the ultimate limit and production scheduling by dynamic programming. It also can set mining scene as a part of a block, a complete block or blocks with the highest value are extracted in the first life years and gradually blocks with less value are considered in future years of mining program.

The most important applications of software are exploration, geology, geochemistry, survey, ore block modeling (block, wired net model and etc.), design open-pit and underground mines and control daily production of mines.

5.2. Introducing used software

Software NPV Scheduler+ for the first time was proposed in 1996 by Tolowinski and Underwood and then in 1997 its new version was presented. Software can use the parametric concept of Lerch and Grossman determine also nested pits, the ultimate limit and production scheduling by dynamic programming. It also can set mining scene as a part of a block, a complete block or blocks with the highest value are extracted in the first life years and gradually blocks with less value are considered in future years of mining program.

Software considerable points are as follows:
1. Software can consider limitations such as money time value, pit slope, min width of the step and send ore to the concentration plant.
2. Software according to the time can determine the pit ultimate limits and produce actual push backs, nested pits and optimal production scheduling aimed to maximize liquidity.
3. Software can also be used in multi-metal reserves.

4. Software allows a scheduler in addition to preparing optimal production scheduling aimed to maximizing NPV, he can also achieve his other objectives.

6. Long term production scheduling for Saheb Saghez iron ore mine

6.1. Introducing studied mine

Saheb Saghez iron deposit is located in Saheb, Sghez, Kordestan Province. The distance between the mine and Saghez is about 15 km through Sanandaj-Saghez road. The distance between the studied region and Saheb is about 16 km in 2nd degree asphaluted road passing Tizabad, Legzi, Ghale Kohne, Chagherloo and Pishkhan and this (Kooh Soltan) is located about 800 m from the asphaluted road. (Fig. 2) shows Saheb Saghez iron deposit location [14,15].

To determine limits with mining priority, examine accurately and achieve parts with higher grade, in addition to drilling old boreholes, 7 boreholes were designed and drilled to be used in modeling and estimating the saving section.

The saving has been calculated by methods of the nearest neighbor and inverse distance for existing and new data for both layers using software Studio 3.

According to the results obtained from the deposit saving block model it was found the highest ore balance is from the horizon of 1640 m and continues to the balance of 1425 m. Ore quantity and quality in different balances have been provided by the nearest neighbor method that total saving is equal to 2669147 tons, sulfur is less than 1%, phosphor is about .08%, iron oxide grade in more than 16.09% and the mine total iron grade is equal to 40.12%.

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Figure 2-Saheb Saghez iron deposit location

6.2. How to extend ore

By drilling new boreholes and drawing geology sections (Fig. 3), the following results were obtained on how to extend ore and its quality [15]:
- Extend ore to 1800 m and not separate zones with ore
- Layers’ extension process as east-west and similar to initial study phase
- Relative reduction in the thickness of ore to the initial phase
- Average increase in total Fe grade analyzed
6.2.1. Saving calculation method

Saving classification in the deposit is based on the nearest distance between the sample and the cell center as follows:
- For the distance to 50 m, category B, measured
- For the distance of 50-100 m, categories C1 and C2, indicated / probable
- For the distance more than 100 m, category C2, inferred / possible

The results of assessing the initial saving as well as its level are given in (Table 1) according to the block model file.

Table 1-Calculate Saheb Saghez deposit saving by the nearest neighbor method (ton)

<table>
<thead>
<tr>
<th>Category</th>
<th>RESERVE</th>
<th>FET</th>
<th>FEO</th>
<th>P</th>
<th>S</th>
</tr>
</thead>
<tbody>
<tr>
<td>NORTH ORE</td>
<td>1536389.445</td>
<td>40.35738</td>
<td>16.2441154</td>
<td>0.076176</td>
<td>0.8240454</td>
</tr>
<tr>
<td>SOUTH ORE</td>
<td>1,132,757.30</td>
<td>39.8752</td>
<td>15.9267662</td>
<td>0.076145</td>
<td>0.89754386</td>
</tr>
<tr>
<td>ALL</td>
<td>2,669,147</td>
<td>40.12</td>
<td>16.09</td>
<td>0.08</td>
<td>0.86</td>
</tr>
</tbody>
</table>

6.3. Optimizing the mining limit

In order to design Saheb Saghez ultimate pit limit, it is necessary to determine saving economic limit. This has been done using software NPV Scheduler+ by the following stages.

6.3.1. The deposit block model

For optimization, geological model prepared by software Studio 3 has been used that the block model has iron grade information as well as the specific weight of the ore and waste. The mentioned model has been processed for more consistency with the software for optimization and entered optimization stage by the block model structure of this software. It should be noted, geology saving level in the entry of software NPV Scheduler, there is 2,668,147 tons of iron ore with an average grade of 40.12 percent and the nearest neighbor method has been used to estimate the saving grade. In (Fig. 4) Saheb Saghez deposit iron 3D block model is shown.

Figure 3-Drilled boreholes to prepare Saheb Saghez deposit model

Figure 4-Sahbe Saghez iron deposit block model
6.3.2. The deposit economic model

Saheb Saghez iron ore deposit economic model according to production costs and sale earnings for each ore and waste blocks has been made by optimization software (see Table 2).

Table 2-Costs of production and sale earning for each of the deposit blocks and waste by optimization software

<table>
<thead>
<tr>
<th>Block Code</th>
<th>Rock (tonnes)</th>
<th>Revenue ($/tonne)</th>
<th>Processing Cost ($)</th>
<th>Mining Cost ($)</th>
<th>NPV ($)</th>
<th>ORE (tonnes)</th>
<th>FETN (%)</th>
<th>FETN R (tonnes)</th>
<th>Strip ratio (%)</th>
<th>mine-life</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pit 80</td>
<td>6,816,753</td>
<td>31,787,501</td>
<td>8,528,760</td>
<td>10,825,120</td>
<td>9,624,926</td>
<td>1,418,752</td>
<td>615,364</td>
<td>514,444</td>
<td>3.8048</td>
<td>43.3736</td>
</tr>
<tr>
<td>Pit 75</td>
<td>6,676,967</td>
<td>31,460,866</td>
<td>8,439,966</td>
<td>10,595,457</td>
<td>9,620,498</td>
<td>1,403,982</td>
<td>609,041</td>
<td>509,158</td>
<td>3.7557</td>
<td>43.3795</td>
</tr>
<tr>
<td>Pit 65</td>
<td>6,162,468</td>
<td>30,115,759</td>
<td>8,093,203</td>
<td>9,753,129</td>
<td>9,543,265</td>
<td>1,346,298</td>
<td>583,001</td>
<td>487,389</td>
<td>3.5773</td>
<td>43.3040</td>
</tr>
<tr>
<td>Pit 60</td>
<td>5,311,167</td>
<td>27,736,161</td>
<td>7,435,070</td>
<td>8,362,910</td>
<td>9,378,389</td>
<td>1,236,818</td>
<td>536,935</td>
<td>448,879</td>
<td>3.2942</td>
<td>43.4126</td>
</tr>
<tr>
<td>Pit 55</td>
<td>4,987,761</td>
<td>26,785,369</td>
<td>7,172,290</td>
<td>8,093,203</td>
<td>9,290,684</td>
<td>1,193,105</td>
<td>518,529</td>
<td>433,490</td>
<td>3.1805</td>
<td>43.4605</td>
</tr>
<tr>
<td>Pit 50</td>
<td>4,306,340</td>
<td>24,667,043</td>
<td>6,603,223</td>
<td>6,729,949</td>
<td>9,043,495</td>
<td>1,098,441</td>
<td>477,521</td>
<td>399,208</td>
<td>2.9204</td>
<td>43.4726</td>
</tr>
<tr>
<td>Pit 45</td>
<td>4,040,569</td>
<td>23,754,645</td>
<td>6,362,463</td>
<td>6,300,370</td>
<td>8,902,695</td>
<td>1,058,390</td>
<td>459,858</td>
<td>384,442</td>
<td>2.8177</td>
<td>43.4489</td>
</tr>
<tr>
<td>Pit 40</td>
<td>3,742,368</td>
<td>22,716,045</td>
<td>6,077,113</td>
<td>5,818,752</td>
<td>8,741,392</td>
<td>1,010,923</td>
<td>439,752</td>
<td>367,633</td>
<td>2.7019</td>
<td>43.5001</td>
</tr>
<tr>
<td>Pit 35</td>
<td>3,380,284</td>
<td>21,315,954</td>
<td>5,703,434</td>
<td>5,235,897</td>
<td>8,470,389</td>
<td>948,431</td>
<td>369,674</td>
<td>309,047</td>
<td>2.5828</td>
<td>43.4934</td>
</tr>
<tr>
<td>Pit 30</td>
<td>3,081,431</td>
<td>20,174,569</td>
<td>5,395,360</td>
<td>4,754,642</td>
<td>8,249,098</td>
<td>897,514</td>
<td>390,553</td>
<td>326,502</td>
<td>2.4333</td>
<td>43.5135</td>
</tr>
<tr>
<td>Pit 25</td>
<td>2,800,572</td>
<td>19,096,032</td>
<td>5,106,315</td>
<td>4,576,488</td>
<td>7,592,739</td>
<td>773,437</td>
<td>337,142</td>
<td>281,851</td>
<td>2.1571</td>
<td>43.5738</td>
</tr>
<tr>
<td>Pit 20</td>
<td>2,442,742</td>
<td>17,415,580</td>
<td>4,651,222</td>
<td>4,651,222</td>
<td>7,397,239</td>
<td>577,327</td>
<td>337,142</td>
<td>281,851</td>
<td>2.1571</td>
<td>43.5738</td>
</tr>
<tr>
<td>Pit 15</td>
<td>1,603,461</td>
<td>13,009,024</td>
<td>3,485,342</td>
<td>2,412,365</td>
<td>6,231,906</td>
<td>251,837</td>
<td>210,536</td>
<td>1.7656</td>
<td>1.7656</td>
<td>43.4364</td>
</tr>
<tr>
<td>Pit 10</td>
<td>1,228,578</td>
<td>10,781,196</td>
<td>2,895,217</td>
<td>1,828,504</td>
<td>5,424,748</td>
<td>481,617</td>
<td>208,710</td>
<td>174,481</td>
<td>1.5509</td>
<td>43.3352</td>
</tr>
<tr>
<td>Pit 5</td>
<td>879,879</td>
<td>6,850,066</td>
<td>1,814,296</td>
<td>952,976</td>
<td>3,800,256</td>
<td>301,807</td>
<td>132,608</td>
<td>110,860</td>
<td>1.1816</td>
<td>43.9381</td>
</tr>
<tr>
<td>Pit 1</td>
<td>315,498</td>
<td>3,650,109</td>
<td>949,610</td>
<td>448,672</td>
<td>2,165,283</td>
<td>157,967</td>
<td>70,073</td>
<td>59,073</td>
<td>0.9972</td>
<td>44.7317</td>
</tr>
</tbody>
</table>

The effect of the variable the mine slope after selecting optimal phase on each parameter of sensitivity analysis, the effect of the mine slope on stripping ratio, sensitivity analysis of the mine slope effect on ore tonnage mined, sensitivity analysis of the mine slope effect on the mine life and sensitivity analysis of the mine slope effect on NPV have been studied that for example the results of the effect of the mine slope on stripping ratio are given in (Graph 1).

6.3.3. Sensitivity analysis of the ultimate slope

One of the important and effective variables on stability and economy parameters in open-pit mines is the mine ultimate slope, its effect on the parameters is not identical and acts distinct and against each other so that by increasing the slope the level of removed waste, then stripping ratio and finally the mining operating costs are reduced and conventionally the pit extension, NPV and finally the mine life are increased. On the other hand, by reducing the slope the mine walls' stability is increased, but the mine NPV and stripping ratio will be reduced, and hence the mine design with a low slope is known as a conservative design.

According to what mentioned, first 5 geologically possible slopes were selected, then to determine the optimal ultimate slope of the mine assuming other variables as constant, the effects of the ultimate slope change were studied among geotechnical possible options (35, 37, 40, 42 and 45 °) on parameters of stripping ratio, mined ore tonnage and finally the life of the mine. For this purpose, the model and the optimal pit of the mine were conducted for four slopes of 35, 37, 40, 42 and 45. That for example, the results for the slope of 42 degree are shown in (Table 3).

Table 3-The results obtained from software NPV Scheduler+ for the slope 42 degree

<table>
<thead>
<tr>
<th>Phase/Code</th>
<th>Rock (tonnes)</th>
<th>Revenue ($/tonne)</th>
<th>Processing Cost ($)</th>
<th>Mining Cost ($)</th>
<th>NPV ($)</th>
<th>ORE (tonnes)</th>
<th>FETN (%)</th>
<th>FETN R (tonnes)</th>
<th>Strip ratio (%)</th>
<th>mine-life</th>
</tr>
</thead>
<tbody>
<tr>
<td>(1) Pit 1</td>
<td>315,498</td>
<td>3,650,109</td>
<td>949,610</td>
<td>448,672</td>
<td>2,165,283</td>
<td>157,967</td>
<td>70,073</td>
<td>59,073</td>
<td>0.9972</td>
<td>44.7317</td>
</tr>
</tbody>
</table>

The deposit economic model according to production costs and sale earnings for each ore and waste blocks has been made by optimization software (see Table 2).
It should be noted, ore level changes affected by the slope to a large extent stems from ore shape. Also, output of software data of ore level in the slope 42 to 45 degrees is constant. According to what mentioned and examine the results of Saheb Saghez iron ore mine as seen in (Graph 2) by increasing the ultimate slope of mine stripping ratio is reduced, the level of mined ore, mine life and NPV will be reduced and since the slope 42 degree has met all the mentioned objectives as well as it has more stability than the slope 45 degree, the slope has been selected as optimal slope and the pit design and ultimate limit have been done according to the slope.

6.3.4. Sensitivity analysis of the product price

The experience of recent years has shown that the most important factor in running mines and annual production is the product’s ultimate price. Although the factors affecting the price of metals or other ore products are the two factors of supply and demand.

The experience has shown that consuming metals periodically is increased or reduced, accordingly it is predicted that under conventional conditions metals’ price is increased per 5-7 y period tangibly and effectively, while the demand of increased the mine or metal product lasts only for a few months. On the other hand, the metal supply to meet increased global consumers' demand needs a special mechanism and investment that due to uncertainty of global economic conditions always 2 parts of supply and demand equation are not equal. Given that the life of most of mines does not exceed a few years in order to predict metals’ price it is suggested to consider the metal average price during the last 20-35 years as a basis of future calculation and under the min profit conditions the min metal price during the last 35 years is used.

According to (Graph 3) which represents iron price changes in the last 5 years, it was necessary to study the sensitivity of Saheb Saghez iron ore project to increased and/or reduced price. So, here sensitivity analysis was performed using daily metal price and 10 and percent more or less than the price specified on the product ultimate price that for example the results of the price of the employer are given in (Table 4) and the price equal to 10% of the price stated are given in (Table 5).
### Table 4 - The results obtained from software NPV Scheduler+ on the employer price

<table>
<thead>
<tr>
<th>Phase/Code</th>
<th>Rock (tonnes)</th>
<th>Revenue ($</th>
<th>Processing Cost ($)</th>
<th>Mining Cost ($)</th>
<th>NPV ($)</th>
<th>ORE (tonnes)</th>
<th>FETN ($</th>
<th>FETN R (tonnes)</th>
<th>Strip ratio (%)</th>
<th>fe (%)</th>
<th>mine-life</th>
</tr>
</thead>
<tbody>
<tr>
<td>(1) Pit 1</td>
<td>658,412</td>
<td>6,850,066</td>
<td>1,814,296</td>
<td>952,976</td>
<td>3,798,245</td>
<td>301,807</td>
<td>132,608</td>
<td>110,860</td>
<td>1,181,166</td>
<td>43.9381</td>
<td>1.509</td>
</tr>
<tr>
<td>(2) Pit 2</td>
<td>1,228,578</td>
<td>10,781,196</td>
<td>2,895,217</td>
<td>1,828,504</td>
<td>5,417,777</td>
<td>481,617</td>
<td>208,710</td>
<td>174,481</td>
<td>1,5509</td>
<td>43.3352</td>
<td>2.4081</td>
</tr>
<tr>
<td>(3) Pit 3</td>
<td>1,603,461</td>
<td>13,009,024</td>
<td>3,485,342</td>
<td>2,412,365</td>
<td>6,224,446</td>
<td>579,784</td>
<td>251,837</td>
<td>210,536</td>
<td>1,7656</td>
<td>43.3464</td>
<td>2.8989</td>
</tr>
<tr>
<td>(4) Pit 4</td>
<td>2,442,742</td>
<td>17,415,580</td>
<td>4,651,222</td>
<td>3,736,488</td>
<td>7,578,370</td>
<td>737,727</td>
<td>337,142</td>
<td>281,851</td>
<td>2,1571</td>
<td>43.5738</td>
<td>3.8686</td>
</tr>
<tr>
<td>(5) Pit 5</td>
<td>2,800,572</td>
<td>19,096,032</td>
<td>5,106,315</td>
<td>4,304,557</td>
<td>8,015,104</td>
<td>849,431</td>
<td>369,674</td>
<td>309,047</td>
<td>2,297</td>
<td>43.5201</td>
<td>4.2472</td>
</tr>
<tr>
<td>(6) Pit 6</td>
<td>3,081,431</td>
<td>20,174,569</td>
<td>5,395,360</td>
<td>4,754,642</td>
<td>8,233,833</td>
<td>897,514</td>
<td>390,553</td>
<td>326,502</td>
<td>2,4333</td>
<td>43.515</td>
<td>4.4786</td>
</tr>
<tr>
<td>(7) Pit 7</td>
<td>3,380,284</td>
<td>21,315,954</td>
<td>5,703,434</td>
<td>5,235,897</td>
<td>8,454,773</td>
<td>948,762</td>
<td>412,649</td>
<td>344,974</td>
<td>2,5628</td>
<td>43.4934</td>
<td>4.7483</td>
</tr>
</tbody>
</table>

### Table 5 - The results obtained from software NPV Scheduler+ on the price equal to 10% of the price stated

<table>
<thead>
<tr>
<th>Phase/Code</th>
<th>Rock (tonnes)</th>
<th>Revenue ($</th>
<th>Processing Cost ($)</th>
<th>Mining Cost ($)</th>
<th>NPV ($)</th>
<th>ORE (tonnes)</th>
<th>FETN ($</th>
<th>FETN R (tonnes)</th>
<th>Strip ratio (%)</th>
<th>fe (%)</th>
<th>mine-life</th>
</tr>
</thead>
<tbody>
<tr>
<td>(1) Pit 1</td>
<td>940,906</td>
<td>10,571,143</td>
<td>2,340,279</td>
<td>1,386,630</td>
<td>6,208,198</td>
<td>392,631</td>
<td>170,410</td>
<td>142,462</td>
<td>1,3964</td>
<td>43.402</td>
<td>1.9632</td>
</tr>
<tr>
<td>(2) Pit 2</td>
<td>1,569,899</td>
<td>15,325,382</td>
<td>3,444,128</td>
<td>2,361,380</td>
<td>8,320,305</td>
<td>572,928</td>
<td>247,049</td>
<td>206,533</td>
<td>1,7401</td>
<td>43.1205</td>
<td>2.8646</td>
</tr>
<tr>
<td>(4) Pit 4</td>
<td>3,161,549</td>
<td>24,395,749</td>
<td>5,467,517</td>
<td>4,887,155</td>
<td>11,433,634</td>
<td>909,517</td>
<td>393,266</td>
<td>328,770</td>
<td>2,4755</td>
<td>43.239</td>
<td>4.5476</td>
</tr>
<tr>
<td>(5) Pit 5</td>
<td>3,546,725</td>
<td>26,182,335</td>
<td>5,864,368</td>
<td>5,508,074</td>
<td>11,911,942</td>
<td>975,533</td>
<td>422,066</td>
<td>352,847</td>
<td>2,6537</td>
<td>43.2652</td>
<td>4.8777</td>
</tr>
<tr>
<td>(6) Pit 6</td>
<td>4,020,261</td>
<td>28,258,255</td>
<td>6,336,752</td>
<td>6,272,257</td>
<td>12,414,982</td>
<td>1,054,114</td>
<td>455,531</td>
<td>380,824</td>
<td>2,8319</td>
<td>43.2146</td>
<td>5.2706</td>
</tr>
<tr>
<td>(7) Pit 7</td>
<td>4,567,203</td>
<td>30,484,530</td>
<td>6,856,763</td>
<td>7,156,481</td>
<td>12,887,382</td>
<td>1,140,617</td>
<td>491,419</td>
<td>410,826</td>
<td>3,0042</td>
<td>43.0836</td>
<td>5.7031</td>
</tr>
</tbody>
</table>

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After selecting the optimal phase in each option available, the effect of the variable of the product ultimate price on each parameter of the sensitivity of stripping ratio to the product ultimate price, minable deposit sensitivity to the product ultimate price and the mine’s life sensitivity to the product ultimate price were studied that for example the results of the sensitivity of stripping ratio to the product ultimate price are given in (Graph 4).

In general, both low and high prices’ prediction states are costly for the mine so that:

1. With regard to a higher price than average product ultimate price during the life of the mine, the mine may be completed before the deadline. Reduced mines’ production value usually affects large mines' operation, but the greatest effect is on small-scale mining contractors.

2. Considering a lower price than the average ultimate price of the product during the life of the mine can cause cost delay, lost opportunities and cost of change to the mine. As if it is predicted that the mine ultimate price of the product in the next year will be considerably less than this year. It is better in this case the mine produces more products instead the next year the mine provides the market with the product at a lower price.

According to what mentioned and examine the results of optimizing Saheb Saghez iron ore mine as seen in (Graph 5), by reducing the price of mined ore the mine life is reduced. On the other hand, according to the results, stripping ratio is not highly sensitive to price changes, but the amount of extractable ore price sensitive is the final product and designing studies to be done on the current the price changes but the level of mined ore is sensitive to the ultimate product price and the current and future price of the mine product should be studied accurately. Also according to the calculations we can conclude that the optimization process is totally affected by the mentioned factors and this activity should be revised again as completely dynamic regarding the changes.

According to the mentioned factors and the above, Saheb Saghez iron deposit economic saving is shown in (Table 6) with the ultimate slope 42 degree.
Table 6-Saheb Saghez iron deposit economic saving with the ultimate slope of 42 degree

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Minable saving as economic</td>
<td>897514 tons</td>
</tr>
<tr>
<td>Ore removal amount</td>
<td>2184000 tons</td>
</tr>
<tr>
<td>Stripping ratio</td>
<td>2.4</td>
</tr>
<tr>
<td>Average mining grade</td>
<td>43.5%</td>
</tr>
<tr>
<td>according to the economic model</td>
<td></td>
</tr>
<tr>
<td>The mine's life per 200000 t annual ore production</td>
<td>5 y</td>
</tr>
</tbody>
</table>

It should be noted, the deposit economic saving as seen in (Fig. 5) is divided in 2 east and west parts.

6.4. The mine ultimate limit design

The mine ultimate pit has been designed according to optimized limits by software NPV Scheduler+ and geometric technical specifications needed by mining conditions, optimal limits specified in software include two east and west parts. Also, according to the limit topographic state, the method of mining Saheb Saghez mine will be mainly quarry.

The mining operation starts in the east part and then continues to the west.

6.4.1. Determine mining limit geometry

The main factor of determining the step geometry is the step height selection, because other the step dimensions follow it directly. Now, the step height in open-pit mines is between 4 and 20 m and this value in large metal mines is between 12 and 15 m [1].

6.4.2. Step slope

Steps' slope is obtained according to parameters such as geomechanic features. In most of open-pit mines steps' slope of ore is between 55 and 80 m [1]. Under conventional conditions and to start the work the slope of 60-70% is recommended. According to the previous experiences and information available on different mines, step face angle has been considered 65 degree.

To determine the ultimate steps' width in this mine, according to the above parameters and using the following Eq.

\[ W_b = \frac{n h}{\tan \alpha} - \frac{1}{n-1} \left[ n W_r + n_2 W_s \right] \]

Where:
- \( W_b \): ultimate steps' width
- \( W_r \): ramps' width
- \( W_s \): safe steps' width
- \( n \): number of steps
- \( h \): step height
- \( \alpha \): step slope

6.4.3. Width of access road

Width of the road is determined by taking the width of the truck, the width of the confidence wall and etc. The road ultimate width is obtained equal to 10 meters due to the factors mentioned by the implemented slope of 8% (in order to use 20 ton tracks) and shown in (Fig. 6) as calculation method schematic of the width.

So considering the above, the design parameters for the main limits in Central and West deposit (Table 7) are as follows:

Table 7- The main design parameters in Central and West deposit

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining bench height</td>
<td>7.5 m</td>
</tr>
<tr>
<td>Ultimate bench width</td>
<td>5 m</td>
</tr>
<tr>
<td>Face slope</td>
<td>60 degree</td>
</tr>
<tr>
<td>Ultimate wall slope</td>
<td>42 degree</td>
</tr>
<tr>
<td>Access road width (the ultimate limit)</td>
<td>10 m</td>
</tr>
<tr>
<td>Access road slope (the ultimate limit)</td>
<td>8%</td>
</tr>
</tbody>
</table>

The mining waste and saving in both designed mining limits according to the type of iron ore and its grade are shown in different mining balances.
As seen in (Table 8), based on the design done, the mining saving in the east part is calculated about 711 thousand tons with an average grade of 41.1% and in the west part is about 153 thousand tons with a grade of 40%.

### 6.4.4. The production scheduler limitations and policies to be used in the primary crusher

Mining scheduling prepared should meet annual mining objectives of 200,000 tons with an average grade of 40 percent. Also, in order to cover the initial investment, the first mining years should have NPV higher than the mine last years. So, considering the predicted objectives first the mining push backs' design has been prepared and then the mining program has been presented. The program (which will be presented in later sections) is the optimized production scheduling to produce input feed of the primary crusher on which the thesis purpose is written.

### 6.4.5. The mining push backs' design

The push back method has the following advantages:

- Reduce the investment in the first years of mine life due to less use of equipment;
- Earlier access to ore and liquidity faster and thus reduce interest payments on the money the mine borrows from the financial sources [2].

According to the above, designed push backs' specifications of Saheb Saghez mine are given in (Table 9).

---

**Western Final Pit**

<table>
<thead>
<tr>
<th>level</th>
<th>Density (gr/cm³)</th>
<th>Volume (m³)</th>
<th>Weight (Ton)</th>
<th>Strip Ratio (W/O)</th>
<th>Medium Grade %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ore</td>
<td>Waste</td>
<td>Ore</td>
<td>Waste</td>
<td>Ore</td>
</tr>
<tr>
<td>1614.5-1612</td>
<td>0.0</td>
<td>2.6</td>
<td>-</td>
<td>99</td>
<td>-</td>
</tr>
<tr>
<td>1612-1604.5</td>
<td>0.0</td>
<td>2.6</td>
<td>-</td>
<td>3,437</td>
<td>-</td>
</tr>
<tr>
<td>1604.5-1597</td>
<td>3.7</td>
<td>2.6</td>
<td>-</td>
<td>8,164</td>
<td>-</td>
</tr>
<tr>
<td>1597-1589.5</td>
<td>3.3</td>
<td>2.6</td>
<td>4,533</td>
<td>32,571</td>
<td>15,059</td>
</tr>
<tr>
<td>1589.5-1582</td>
<td>3.5</td>
<td>2.6</td>
<td>7,241</td>
<td>71,899</td>
<td>25,268</td>
</tr>
<tr>
<td>1582-1574.5</td>
<td>3.7</td>
<td>2.6</td>
<td>14,511</td>
<td>89,638</td>
<td>54,294</td>
</tr>
<tr>
<td>1574.5-1567</td>
<td>3.9</td>
<td>2.6</td>
<td>11,770</td>
<td>49,826</td>
<td>46,141</td>
</tr>
<tr>
<td>1567-1559.5</td>
<td>3.7</td>
<td>2.6</td>
<td>3,272</td>
<td>14,662</td>
<td>12,041</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>3.6</td>
<td>2.6</td>
<td>41,327</td>
<td>270,296</td>
<td>152,803</td>
</tr>
</tbody>
</table>

**Eastern Final Pit**

<table>
<thead>
<tr>
<th>level</th>
<th>Density (gr/cm³)</th>
<th>Volume (m³)</th>
<th>Weight (Ton)</th>
<th>Strip Ratio (W/O)</th>
<th>Medium Grade %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ore</td>
<td>Waste</td>
<td>Ore</td>
<td>Waste</td>
<td>Ore</td>
</tr>
<tr>
<td>1597-1589.5</td>
<td>0.00</td>
<td>2.6</td>
<td>-</td>
<td>4,656</td>
<td>-</td>
</tr>
<tr>
<td>1589.5-1582</td>
<td>0.00</td>
<td>2.6</td>
<td>-</td>
<td>14,256</td>
<td>-</td>
</tr>
<tr>
<td>1582-1574.5</td>
<td>3.69</td>
<td>2.6</td>
<td>-</td>
<td>28,162</td>
<td>-</td>
</tr>
<tr>
<td>1574.5-1567</td>
<td>4.16</td>
<td>2.6</td>
<td>11,720</td>
<td>49,598</td>
<td>48,785.93</td>
</tr>
<tr>
<td>1567-1559.5</td>
<td>3.73</td>
<td>2.6</td>
<td>22,019</td>
<td>79,028</td>
<td>82,143.52</td>
</tr>
<tr>
<td>1559.5-1552</td>
<td>1.99</td>
<td>2.6</td>
<td>20,714</td>
<td>94,678</td>
<td>41,317.78</td>
</tr>
<tr>
<td>1552-1544.5</td>
<td>5.15</td>
<td>2.6</td>
<td>25,750</td>
<td>123,557</td>
<td>132,620.49</td>
</tr>
<tr>
<td>1544.5-1537</td>
<td>3.61</td>
<td>2.6</td>
<td>25,480</td>
<td>115,820</td>
<td>92,100.48</td>
</tr>
<tr>
<td>1537-1529.5</td>
<td>3.26</td>
<td>2.6</td>
<td>22,978</td>
<td>105,438</td>
<td>74,881.14</td>
</tr>
<tr>
<td>1529.5-1522</td>
<td>3.15</td>
<td>2.6</td>
<td>19,600</td>
<td>97,415</td>
<td>61,721.33</td>
</tr>
<tr>
<td>1522-1514.5</td>
<td>3.94</td>
<td>2.6</td>
<td>12,761</td>
<td>82,165</td>
<td>50,311.84</td>
</tr>
<tr>
<td>1514.5-1507</td>
<td>3.93</td>
<td>2.6</td>
<td>12,123</td>
<td>73,003</td>
<td>47,660.10</td>
</tr>
<tr>
<td>1507-1499.5</td>
<td>3.84</td>
<td>2.6</td>
<td>11,457</td>
<td>59,249</td>
<td>43,959.62</td>
</tr>
<tr>
<td>1499.5-1492</td>
<td>3.70</td>
<td>2.6</td>
<td>9,575</td>
<td>36,547</td>
<td>35,427.50</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>3.7</td>
<td>2.6</td>
<td>194,177</td>
<td>963,569</td>
<td>710,930</td>
</tr>
</tbody>
</table>
As seen in (Table 9), three push backs have been considered with capacities of 155, 356 and 353 thousand tons to optimize Saheb Saghez mine production scheduling. It should be noted, the number of push backs required with respect to the trial and error to achieve the objective of optimizing production scheduling has been designed and prepared.

6.4.6. Optimal production scheduling
After designing the mining push backs as a guide for the production scheduling, using software NPVS Saheb Saghez mine production scheduling has been done.

<table>
<thead>
<tr>
<th>No.</th>
<th>The number of push backs</th>
<th>Ore amount / ton</th>
<th>Waste / ton</th>
<th>Stripping ratio / ton</th>
<th>Ore average grade / %</th>
<th>Average sulfur grade / %</th>
<th>The life of each push back / y</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>1</td>
<td>154981</td>
<td>590461</td>
<td>3.8</td>
<td>40.9</td>
<td>1.02</td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>2</td>
<td>356693</td>
<td>999627</td>
<td>2.8</td>
<td>40.3</td>
<td>0.55</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>3</td>
<td>352957</td>
<td>1617975</td>
<td>4.6</td>
<td>40.7</td>
<td>1.38</td>
<td>2</td>
</tr>
<tr>
<td>Total</td>
<td>3</td>
<td>864631</td>
<td>3208063</td>
<td>3.7</td>
<td>40.6</td>
<td>0.97</td>
<td>5</td>
</tr>
</tbody>
</table>

As seen in (Table 9), three push backs have been considered with capacities of 155, 356 and 353 thousand tons to optimize Saheb Saghez mine production scheduling. It should be noted, the number of push backs required with respect to the trial and error to achieve the objective of optimizing production scheduling has been designed and prepared.

Limitations and constraints used in the production scheduling are as follows:
- Annual mining of 200,000 tons of ore sent to the processing unit;
- Mining of ore with an average grade of 40% iron and sulfur below 1%;
- High NPV in the first mining years according to push back and to cover the initial investment.
- After applying the above limitations in software NPVS, the results are given in (Table 10).

<table>
<thead>
<tr>
<th>No.</th>
<th>Year</th>
<th>Ore amount / ton</th>
<th>Waste / ton</th>
<th>Stripping ratio / ton</th>
<th>Ore average grade / %</th>
<th>Average sulfur grade / %</th>
<th>NPV</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>1st</td>
<td>200231</td>
<td>673698</td>
<td>3.4</td>
<td>40.85</td>
<td>1.04</td>
<td>1955481</td>
</tr>
<tr>
<td>2</td>
<td>2nd</td>
<td>200352</td>
<td>676354</td>
<td>3.4</td>
<td>40.60</td>
<td>0.98</td>
<td>1844420</td>
</tr>
<tr>
<td>3</td>
<td>3rd</td>
<td>200924</td>
<td>675062</td>
<td>3.4</td>
<td>40.05</td>
<td>0.88</td>
<td>1336802</td>
</tr>
<tr>
<td>4</td>
<td>4th</td>
<td>200242</td>
<td>761401</td>
<td>3.8</td>
<td>40.56</td>
<td>1.10</td>
<td>1277172</td>
</tr>
<tr>
<td>5</td>
<td>5th</td>
<td>62882</td>
<td>421401</td>
<td>6.7</td>
<td>40.76</td>
<td>0.11</td>
<td>394864</td>
</tr>
<tr>
<td>Total</td>
<td>5 years</td>
<td>864631</td>
<td>3208063</td>
<td>3.7</td>
<td>40.5</td>
<td>0.9</td>
<td>1361748</td>
</tr>
</tbody>
</table>

As seen in Table 10, the needed mining for the plant will be provided during the years of operation (except the last year that is not a full year). Also, the grade of ore, sulfur and NPV value have been achieved based on the objectives of the project.

7. Conclusions
After making designing the economic model using software NPVS by presented assumptions in the previous sections, sensitivity analysis was performed on the ultimate slope and according to changes in NPV nested pits and stripping ratio the appropriate ultimate slope has been selected. It should be noted if there are slope stability studies, the information will also involve in and affect the selection process of the mine ultimate slope. To compare and select the optimal pit, criteria such as cumulative NPV changes of nested pits, stripping ratio of nested pits, the ratio of NPV / Rock and NPV / Ore have been used. Nested pit No. 35 with 948,700 tons deposit and total stripping ratio 2.5 has been selected as optimal mining limit. According to criteria needed in the production scheduling and trial and error to design push backs finally 3 push backs were selected that the highest net present value, as well as functionality were provided. After designing push backs, according to defined limits in software NPVS, the production scheduling has been presented for 5 years of the mine to feed the primary crusher to an annual rate of 200 thousand tons, an average grade of 40% for ore and to a maximum of 1% for sulfur. Due to the reduction in readily available ore in near future and increased ore need with increasing the population, the importance of optimal production scheduling in mining has been increased economically. Many researchers are examining and presenting methods of the production scheduling with less error and high return. As seen in case study sensitivity analysis, the ultimate product price is one of the factors affecting NPV and production strategy selection. Limits defined in software NPVS are based on mining policies needed in order to optimize production scheduling. Also, in the process of preparing an economic model, providing the ultimate mining limit, push backs and etc. has been also considered as one of optimal production scheduling objectives. For all mining set objectives, to set mining stages, various options have been defined for the production scheduling, in the options 2 factors of the amount of mining ore and the number of push backs have been changed and the results obtained have been analyzed in the scheduling. In the options, the number of push backs varied between 2 and 5 (comparison of the net present value shown in push backs). Then, the effect of these changes on the economic factors,
operational limitations, production rate curve and meet the defined objectives have been examined and finally an option with push backs has been considered to optimize production scheduling. The results obtained in the case study have included 864 thousand tons of economic ore with an average grade of 40 percent in the ultimate mining limit. Accordingly, the production scheduling has been presented regarding annual mining of 200 thousand tons of ore to feed the primary crusher with average ore of 40% and sulfur 1% by software NPVS during 5 years of the mine's life.

References
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[16] www.knoema.com
Application of compound Poisson process for modelling of ore flow in a belt conveyor system with cyclic loading

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Abstract—This paper deals with the analysis of the random process of cyclic loading of a mining belt conveyor with portions of ore discharged by loaders or trucks. Such transfer of transported ore from a cyclic to a continuous transport is typical for the specific mining operations implemented in the underground copper ore mines with room and pillar mining. The conveyors in such systems are usually significantly oversized in order to match the peak loads of cumulated discharges of ore hauled from mining fields by loaders. Therefore the actual loadings that occur in the mining transportation systems need to be analysed to provide the data for more accurate design and control of belt conveyors. The large dataset of actual loadings of the belt conveyors has been used for the stochastic modelling of the analysed process. The compound Poisson process has been proposed as mathematical tool to analyse/describe properties of ore flow. The discussion of the chosen distribution functions and results of the fitted model simulations compared with the examined measurement data are presented. In this paper the case of a belt conveyor loaded only by a single feeding point where loaders are randomly discharged has been analysed. More complex cases (several loading points, mixed ore supply from cyclic and continuous ore stream from preceding conveyors or ore bunkers) are under investigation and will be presented in the future.

Keywords: Belt conveyor; ore stream; stochastic modelling; compound Poisson process; estimation

1 Introduction

The transportation systems used in analysed copper ore mine were designed for the specific needs of their room and pillar mining system. Its characteristic feature is the use, at the various stages and on different scales, of both cyclic transport (loaders and trucks) and the high-capacity continuous transport - belt conveyors. The cyclic transport involves the haulage of a blasted ore, from a mining face to a mining unit discharge point, from which the ore is dumped directly onto a grid (a conveyor feeding point). The actual layout of transportation routes of loaders and trucks as well as the location of discharge points on conveyors depend on the mining faces advances in the adjacent mining fields. The priority of the transportation system is maximizing the output of active mining fields. Therefore, the belt conveyors capacity should enable to acquire the ore supplied by loaders or trucks without forced delays of discharging. However, conveyors should not be oversized as it increases total costs of ore transportation. Belt conveyors instantaneous capacity should match the peaks of ore volumes discharged onto the feeding points. Conveyor belt strength and conveyor drives should be rated according to the maximum averaged capacity with regard to the period of time necessary for loading the whole length of a conveyor [1, 2]. Therefore, both the analysis of ore volume from a single loader or truck discharged onto the conveyor feeding point and of the summed loadings that could fill the whole conveyor belt are vital for the optimized design and control of belt conveyors [3, 4, 5].

Above problem has already been raised for the conveyors used in continuous mining operations (lignite surface mines). The statistical investigation of volumes of material supplied by the continuously mining excavators was done with the use of Extreme Value Theory for analysing time series of actual volume capacities [6]. The obtained results have proved that significant savings due to optimized equipment selection of belt conveyors without any loss of their required capacity are possible [7].

Another issue related to the transfer of ore from loaders and trucks onto belt conveyors is tracking the parameters of mined copper ore. These are recognized in-situ but after being discharged onto the conveyor loading points, portions of ore loadings from various mining faces are eventually blended when supplied to the processing plant. The investigation of the ore stream transported by belt conveyors should help to provide with the knowledge of ore compound that is needed by the operators of the processing plant [8] and in general - necessary for the implementation of Quality Assurance / Quality Control procedures.

However, first step to achieve it is to model of volume/mass
of ore transported by conveyor in order to understand the ore flow. In the complex underground-transportation network, a conveyor can be supplied with an ore either cyclically – by loader/trucks (directly from mining fields) or from preceding conveyors/bunker – then ore stream will be very different, much more continuous and stationary. The most complex is mixing of cyclic and continuous ore supply. This paper deals with the cyclic ore supplying. We propose to model the ore flow in belt conveyor by using compound Poisson process (CPP). It is the one of the most common continuous-time processes used in stochastic modelling. Typically, it is used in the insurance risk theory [9]. Indeed, it is useful in modelling the total amounts of claims. Moreover, it can be also used in finance. The optimal dividend strategies with compound Poisson total amount processes have been described in [10]. In [11] the exemplary application to stock price and shock model is illustrated. Furthermore, compound Poisson process can be applied in many different branches of sciences e.g. genetics [12] or reliability modeling [13]. This type of process gives a possibility to apply various types of distributions, commonly the positive distributions like Pareto, exponential or gamma are used [9].

It is worth mentioning that distribution of analysed weights of the ore mass reveal the bimodal properties. Therefore, the slight modification of the classical CPP has to be performed. In particular, we propose to apply mixture of two positive random variables. Furthermore, the statistical tests based on the empirical distribution are performed in order to verify the estimation results.

The paper is structured as follows. In the second section the analysed data set is described. In the next section the compound Poisson process and positive distributions are recalled. Moreover, the approach of bimodal distributions is presented. The estimation of the model parameters is presented in next section. The application of described methods to real data of ore mass transported by belt conveyor is contained in subsequent section. The conclusions are drawn in the last section.

2 Data description

Analyzed data set consists of ore mass measurements which is supplied to the grid cyclically. Moreover, data contain information about the time which the loader spent on an ore transportation from the mining face to the dumping point. One belt conveyor can be supported by several loaders, which transport the ore from different mining faces. In view of that, it is important to model the process of loaders’ arrivals to the dumping point. Thus, the stochastic modelling approach can be applied, because neither waiting time for the loader nor the amount of transported ore are constant. The data acquired from the weight measurement system installed on the belt conveyor located in the underground mine is analysed. The scale situated on the belt conveyor measures the instantaneous load. In the recorded signal the sum of the transported ore mass is acquired. The system is summing the mass and once the 1 tonne is reached the increment is saved. It is worth mentioning that the scale is on the belt conveyor, but below the dumping point. Therefore, after loader’s discharging one can observe a few increments in the signal recorded on the scale. Analysed belt conveyor is not supplied by any other conveyor or any bunker, moreover it is connected with only one grid. It gives a possibility to model the cyclic charging process. The length of the belt conveyor is equal to 830 meters and the nominal speed is 2m/s. The available data includes the measurements of weight from November 2015 to March 2016. To use collected data it is necessary to clean acquired signals from outliers observations and measurement errors. All pre-processing procedures are thoroughly described in section subsequent.

3 Compound Poisson process

To the analysis of the ore flow in belt conveyor system we propose to apply the compound Poisson process. In this section we remind the definition of this process and its main properties. Firstly, let us denote that the stochastic process \( \{N_t\}_{t \geq 0} \) is said to be homogeneous Poisson process (HPP) with time \( t \) and intensity \( \lambda > 0 \) when it satisfies three conditions [14]:

- \( N_0 = 0 \),
- it has independent increments,
- \( N_{t+s} - N_s \sim \text{Poisson}(\lambda s) \).

The HPP is a positive counting process. Furthermore, it can be applied to analyze the moments of arrival, e.g. the ore to the grid. As it was mentioned, in order to model the whole supplying process we apply the CPP \( \{X_t\}_{t \geq 0} \) which is given by the following formula [14]:

\[
X_t = \sum_{i=1}^{N_t} Y_i,
\]

where \( \{N_t\}_{t \geq 0} \) is homogeneous Poisson process with the intensity parameter \( \lambda \) and \( Y_i \) are independent identically distributed (i.i.d.) random variables. The CPP process in general is characterized by the intensity \( \lambda \) and the distribution of \( Y_i \) random variables. It is a continuous-time stochastic process, which arrival time is following the HPP and its jumps are random variables. Therefore, it can be used to model data with exponentially distributed waiting times between the consecutive jumps and with varying jump sizes. From the definition it can be concluded that its expected value is equal to \( \mathbb{E}(X_t) = \lambda t \mathbb{E}(Y) \). The application of compound Poisson process to the ore flow on the grid and the belt conveyor seems to be relevant in this problem. The process \( N_t \) can be considered as the moments of loaders’ arrivals to the dumping point. To estimate parameters of the \( N_t \) process we use times between consecutive discharges, which will be called waiting times. The amount of ore discharged by the loader will be treated as the random variable \( Y \) and is called increments or jumps. In such approach two tasks are essential:

- estimation of intensity \( \lambda \) of the homogeneous Poisson process \( \{N_t\}_{t \geq 0} \) based on real data,
- fitting the distribution of compound Poisson process increments \( Y \) and estimation of its parameters.
One of the most crucial and challenging part is fitting the proper distribution for the jumps. As it was stated the mass of the ore which flow through the grid is treated as an increment of stochastic process. Therefore, only positive distributions can be analyzed.

3.1 Positive distributions

Let us firstly recall some of the well-known distributions that can be useful in modelling of amount of ore discharged by the loader. The formulas for probability density function (pdf) and cumulative distribution function (cdf) are presented. Moreover, some basic properties are described.

Log-normal distribution

Let Z be a Gaussian random variable with mean equals to $\mu$ and variance $\sigma^2$. Then the random variable $X = \exp(Z)$ is log-normal. Its probability density function $f_X(x)$ is given by following equation [15]:

$$f_X(x) = \frac{1}{x\sigma\sqrt{2\pi}} \exp\left(-\frac{(\ln(x) - \mu)^2}{2\sigma^2}\right),$$

where $x > 0$, $\mu \in \mathbb{R}$, $\sigma > 0$ and $erfc(x) = \frac{2}{\sqrt{\pi}} \int_{x}^{\infty} e^{-t^2} dt$ is the complementary error function. When the parameter $\sigma$ is small the log-normal distribution can be similar to normal one. On the other hand, for large $\sigma$ the tails are semi-heavy [16]. Moreover, the k-th moment and variance can be expressed by formula:

$$\mathbb{E}(X^k) = \exp\left(\frac{1}{2}\sigma^2 + k\mu \right),$$

$$\text{Var}(X) = [\exp(\sigma^2) - 1] \exp(2\mu + \sigma^2) - [\exp(\sigma^2) - 1] \exp(2\mu + \sigma^2).$$

This distribution is especially useful in insurance for modeling the claim sizes. Furthermore, it has been applied to geochemistry [17], epidemiology [18] or geology [19]. On the other hand the Laplace transform does not have explicit formula and moment generating function does not exist. This fact are the main drawback of this distribution.

Gamma distribution

The second recalled distribution is the Gamma one. It is described by two parameters shape $k > 0$ and scale $\theta > 0$, then the probability density function is following [20]:

$$f_X(x) = \frac{1}{\Gamma(k)\theta^k} x^{k-1} \exp\left(-\frac{x}{\theta}\right),$$

where $x > 0$, $\Gamma(\cdot)$ is a gamma function and $\gamma(k, \frac{x}{\theta}) = \int_{0}^{\frac{x}{\theta}} t^{k-1} e^{-t} dt$ is a lower incomplete gamma function. It is worth mentioning that the sum of $k$ i.i.d. exponentially distributed random variables $\sum_{i=1}^{k} X_i$ follows Gamma distribution with shape parameter $k$ and scale $\lambda^{-1}$, where $\lambda$ is an intensity of the $X_i$. The r-th moment can be calculated from the formula

$$\mathbb{E}(X^r) = \theta^r \frac{\Gamma(k+r)}{\Gamma(k)},$$

thus the expected value is equal $\mathbb{E}(X) = k\theta$. It is worth mentioning that random variable $X \sim \text{Gamma}(v/2, 2)$ has chi-squared distribution with $v$ degrees of freedom. Therefore, gamma distribution is associated to the sum of the squares of independent normal variables [15]. The gamma distribution is typically applied to insurance data ([21]), however it is used also in other branches of science e.g. hydrography [22] or finance [23].

Weibull distribution

Another positive distribution, which can be applied to the considered problem, is the Weibull one. It also depends on two positive parameters: scale $\alpha$ and shape $k$. For $x \geq 0$ the probability density function is given by the formula [15]:

$$f_X(x) = \frac{k}{\alpha} \left(\frac{x}{\alpha}\right)^{k-1} \exp\left(-\left(\frac{x}{\alpha}\right)^k\right),$$

The expected value is equal to $\mathbb{E}(X) = \alpha\Gamma(1+1/k)$. It is related to the exponential distribution, namely $Y = (X/\alpha)^k$ is exponentially distributed with intensity 1. In case of $k < 1$ Weibull distribution becomes heavy tailed. This distribution is especially useful in lifetime models, [24], [25], [26]. Moreover, it can be also applied to wind speed distribution [27] or finance [28].

Burr distribution

Finally, let us present the three parameters family of positive Burr distribution. Its probability density function is [29]:

$$f_X(x) = \frac{k c}{(\alpha + \frac{c}{x})^{k+1}},$$

where $x > 0$, $\alpha > 0$ is the scale parameter, $c > 0$ and $k > 0$ are shape parameters. This distribution is related to Pareto one, namely if $Y$ has Pareto distribution then $X = Y^{1/c}$ follows Burr distribution. Clearly, the inverse of cdf has simply form, thus it can be used for simulation of Burr random variable. This distribution is very flexible and can express many types of real data. The limit distribution when $k \to \infty$ is Weibull [29]. It can be applied to many different type of data. In [30] it was applied to model the diameter of the trees. Moreover it was applied as a failure model in [31].

3.2 Bimodal distributions

In the real data related to amount of ore discharged by the loader we recognized that presented above non-negative distributions can not be applied in the classical form. Indeed, it is related to the fact that in the data we observe bimodal behavior. Bimodal distributions in the real data analysis occurs frequently [32, 33]. Furthermore, none of the above introduced non-negative distributions are bimodal. In order to overcome this problem we propose to take under consideration the mixture of two distributions. Thus, let us assume that $X$ is a random variable which probability distribution function $f_X(x)$ of the following form:

$$f_X(x) = pg_1(x) + (1-p)g_2(x), \quad (2)$$

where $p \in [0, 1]$ is a mixing parameter, $g_1(x)$ and $g_2(x)$ are the probability distribution functions. Obviously the similar
relation can be observed in the cdf: \( F_X(x) = pG_1(x) + (1 - p)G_2(x) \), where \( G_1(x) \) and \( G_2(x) \) are the cumulative distribution functions related to \( g_1 \) and \( g_2 \) respectively. Using such mixing techniques we are able to create a wide class of the distributions. Each distribution with the explicit form of pdf can be applied. Moreover, as one can expect it is possible to create bimodal density functions. In our analysis we consider the mixtures of the above described non-negative distributions.

4 Estimation of CPP parameters and testing

4.1 Estimation of intensity of the Poisson process

In order to estimate the arrival time process it is sufficient to find the intensity value \( \lambda \). In case of the HPP it is a constant value. Moreover, as it was mentioned the waiting times between consecutive arrivals are following the exponential distribution. The cumulative distribution function in that case is defined as [16]:

\[
F(t) = 1 - e^{-\lambda t}, \quad t > 0.
\]

(3)

Furthermore, the tail of the distribution \( 1 - F(t) \) is the exponential function \( e^{-\lambda t} \). Using this property we can estimate the \( \lambda \) parameter. In our estimation procedure we fit the exponential function \( e^{-\lambda t} \) to the tail of the empirical distribution function \( 1 - F_n(t) \). Let us recall that, the empirical distribution function for the vector of independent observations \( x_1, \ldots, x_n \) is defined as follows [16]:

\[
\hat{F}_n(t) = \frac{1}{n} \sum_{i=1}^{n} 1(x_i \leq t).
\]

(4)

The fitting of proper parameter can be performed with the least square method.

4.2 Jump analysis

As it was mentioned above, the jumps in the considered data exhibit behavior related to bimodal distribution. We propose to take under consideration the mixtures of non-negative distributions described above. The distribution parameters estimation can be performed with Maximum Likelihood Estimation (MLE) method. For the vector of independent observations \( x_1, \ldots, x_n \) and the set of the distribution parameters \( \theta \) the estimation is done by maximizing the log-likelihood function:

\[
\max_{\theta} \left\{ L(\hat{\theta}) \right\} = \max_{\theta} \left\{ \sum_{i=1}^{n} f(x_i; \theta) \right\}.
\]

The procedure can be easily applied for the distribution with known pdf. Thus, the parameters of bimodal distribution can be estimated with MLE. In Fig. 1 there is presented a flowchart of the estimation method. Firstly, we assume that the mixing parameter is equal to \( p = 0.01 \), next in each loop iteration we increase its value until \( 1 \) is reached. Then the mixture of two densities \( f(x) = p g_1(x) + (1 - p) g_2(x) \) is fitted. One can observe that MLE method is used to estimate the set of parameters \( \theta_1 \) for densities \( g_1(x) \) and parameters \( \theta_2 \) for \( g_2(x) \) with a priori given value of the mixing parameter \( p \), where \( \theta = [\theta_1, \theta_2] \).

![Flowchart of the estimation method for the mixture of the densities parameters.](https://example.com/flowchart)

In order to find the best \( p \) the Kolmogorov-Smirnov test is performed. The test statistics is defined as:

\[
K(\hat{\theta}, p) = \max_x \left\{ |F_E(x) - F_T(x)| \right\},
\]

where \( F_E(x) \) is the empirical cumulative distribution function and \( F_T(x) \) is the fitted theoretical cdf. The statistic measures the maximum distance between the theoretical and empirical cdf. Then we increase the value of the mixing parameter and repeat the whole procedure. For each step the value of the Kolmogorov-Smirnov test is computed. Finally, the most suitable value of the mixing parameter is chosen. We are looking for parameter \( p \) which minimizes test statistic \( K(\hat{\theta}, p) \). This procedure allows us to fit the bimodal distribution to the data.

4.3 Statistical tests based on empirical distribution function

In order to confirm that proper distribution of jumps is fitted, the statistical tests can be performed. Obviously, the mentioned Kolmogorov-Smirnov test can be applied. Let us also recall three other tests based on the empirical distribution function: Kuiper, Cramer-von Mises and Anderson-Darling tests [16]. Let us assume that \( F(x) \) is a true cdf of the random variable \( X \) and \( z_i = F(x_i) \), where \( Z \) is uniformly distributed on \( (0, 1) \) and \( z_i \) is a realization of \( Z \). Then the distance between empirical cdf of \( z_i \) and uniform distribution are the same as for empirical cdf of \( x_i \) and its true cdf [16] for any \( i = 1, \ldots, n \). This fact can be used in computation of the test statistics. Let us assume that \( z_{(1)} < \cdots < z_{(n)} \) are the order statistics of \( z \) then the test statistics \( V = \text{Kuiper}, W^2 = \text{Cramer-von Mises}, A^2 = \text{Anderson-Darling} \).
The simulation of increments \( f \) random variable \( Y \) can be performed by composition method. Namely, when the pdf of \( Y \) is \( f_Y(x) = p g_1(x) + (1 - p) g_2(x) \), where \( p \) is a mixing parameter and \( g_1(x) \) and \( g_2(x) \) are the pdfs. Moreover, we know how to simulate the random variable \( X_1 \) which pdf is \( g_1(x) \) and \( X_2 \) which pdf is \( g_2(x) \). The algorithm for simulation of \( Y \) is following:

1. Generate a discrete random variable \( I \in \{1, 2\} \).
2. \( P(I = 1) = p \) and \( P(I = 2) = 1 - p \).
3. Generate \( X_I \) with pdf \( g_I \).
4. Return \( Y = X_I \).

The simulation results will be conducted for one shift time interval, namely 6 hours. It should be mentioned specific behaviour of data during one shift. At the beginning and at the end of the shift in most cases the ore is not discharging on the belt conveyor. This fact can be connected with the change of the crew for the next shift. Due to that, the compound Poisson process was simulated starting from the first jump, during analysed shift, in the ore flow process. Clearly, each process can have different length.

5 Real data analysis

In this section the performance of the described compound Poisson process in modelling of the ore mass flow through the grid is tested. In the first step the data pre-processing has to be applied, such data are going to be used for stochastic model fitting. In the next stages the intensity of the waiting times and the distribution of the jump size have to be found. The estimation methods previously illustrated are applied. In particular, the intensity of the process is estimated from the tails of the waiting times empirical distribution. Furthermore, using the MLE the proper distribution is fitted to the jump sizes. Then, the statistic tests are applied in order to ensure that proper distribution was chosen. In the estimation process the merged data from whole month is used. Thus, the parameters are estimated for the data from one month. Finally, the trajectories of the fitted compound process are simulated and compared with the real ore mass flow process. It is worth mentioning that the signal from one shift is taken as a one real trajectory. At the estimation stage we assume that signals from all shifts in one month follow the CPP with the same parameters.

5.1 Real data pre-processing

The proper data pre-processing is an issue that is of great importance in model fitting procedure. In order to illustrate applied procedures the exemplary data from March 2016 is presented. The pre-processing methods for data from other months are exactly the same. The raw data is depicted in Fig. 2. The measurement system acquires cumulative weight of ore mass transported through the conveyor scale. One can observe the outliers values which are present in the plot of weight. They need to be eliminated from further analysis. For this purpose the raw data was differentiated (Fig. 2a). Moreover, values bigger than 1 are also set to zero. The data after pre-processing is
presented in Fig. 3. According to cleaned data the ore is not constantly supplied to the belt conveyor. One can notice that mine is not working during Sundays (6th, 13th, 20th, 27th of March), moreover the regular stoppages of belt conveyor are related to blasting (about 6 a.m. and 6 p.m.). In the zoomed plot of the differentiated data (Fig. 2b) one can observe the „1 tonne” value related to cyclic discharging ore from the loader. Clearly, the weight should not be negative. Therefore, in order to clean data from the measurement errors, all negative values of differentiated weight data were set to zero.

Next step of the pre-processing is an extraction of the times between consecutive arrivals of the loaders to the dumping point. It is strictly associated with the ore appearance on the belt conveyor. This conveyor is supplied by only one dumping point, thus during some shifts no ore is transported (Fig. 3a)) which is not an abnormal situation. Let us consider exemplary data from one shift. In Fig. 3b data acquired during second shift at 10th of March 2016 are illustrated (red stars). The difference between consecutive discharges was at least 50 seconds. That approach allows to determine the moments of discharging starts which are marked in Fig. 3b. The sum of differentiated weight between two consecutive discharges are treated as weight of ore mass supplied on the belt conveyor by one loader.

The obtained data of waiting times and weights are presented in Fig. 4. For further analysis there were taken only data with waiting times less than 1500 seconds (25 min.) and weights higher than 1 tonne and less than 40 tonnes. According to the mine condition, the other observations were determined as incorrect by the algorithm. In particular, the waiting times longer then 25 minutes correspond to the blasting or the shift change, hence the ore can not be transported. Moreover, longer breaks are deterministic or can be caused by some machine break downs. Furthermore, typically one loader transports at least 2 tonnes of the ore and significantly less than 40 tonnes, thus such bounds seem to be relevant. The data obtained from such pre-processing procedure is a starting point to solve the problem of modeling of the ore flow process.

5.2 CPP intensity estimation

Let us recall that in estimation procedures we are going to analyse months separately. Thus, for each month we analyse all observation from pre-processed data. Firstly the kernel density estimator, with normal kernel, is fitted to the waiting times data. As one can observe in Fig. 5a the shape of the kernel density estimator is similar for different months. It may suggest that variables describing behaviour of waiting times have the same distribution. The differences between particular months are reflected only in the parameters of this distribution. Furthermore, obtained kernel density estimators resemble the density of the exponential distribution. In order to estimate the intensity parameter let us analyze the tails of the empirical distribution (Fig. 5b). The non-linear least squares method [35] was used to fit exponential function \( e^{-bx} \) to the tail of empirical distribution function. The results of the analysis are presented in Fig. 6. The estimated \( b \) parameters of exponential function for different months are varying from 0.002 up to 0.0028. Therefore the
Figure 4: a) Waiting times for $i$-th discharges, b) weight of ore mass supplied on the conveyor in $i$-th discharge.

Figure 5: For waiting times of each month between November 2015 and March 2016 there are presented: a) fitted kernel density estimator and b) tails of empirical distribution function.

Figure 6: Fitted exponential function $f(x) = e^{-bx}$ (red line) to the tails of empirical distribution function (black line) related to data from each month between November 2015 and March 2016. Estimators of $a$ and $b$ parameters are showed on the plots.

The intensity estimator $\hat{\lambda}$ of the homogeneous Poisson process is equal to the estimated parameter $b$ for each month.
5.3 Estimation of the compound Poisson process increments for real data

In the CPP model the jumps $Y_i$ constitute sample of i.i.d. random variables. Clearly, the proper distribution has to be fitted to the data. Analysed data sample consists of ore mass, which flow through the grid. First of all, let us analyse the kernel density estimator of the weights of ore mass. One can observe that the densities significantly differ for each month. Furthermore, they are bimodal. Obviously, the mass cannot be smaller than zero, hence only non-negative distribution can be fitted. In modelling process we tried to fit positive distributions, which have been previously recalled.

In Fig. 7 there are presented the kernel density estimators for the weights distribution with normal kernel. One can observe that all of the distributions are bimodal. Thus, the mixture of two distributions can be applied. Let us estimate the density for each month, four different mixing distributions were tested. In Fig. 8 the fitted densities are presented and compared with the normalized histogram and the kernel density estimator. Some of the distributions explain the data properly. Indeed, especially mixture of log-normal and Weibull distributions seem to be adequate. They are able to detect both modes in the data. Interestingly, in March 2016 the result for Weibull, Burr cases are almost similar. One can observe that only mixture of Gamma distributions significantly differs from the real data. The comparison of the histogram and fitted pdf give satisfactory results. However, we concentrate also on testing the appropriate distribution by using the statistic tests described in this arti. The output of the test for each month is presented in Tab.1. According to the results of statistic tests, for January and March the best results are obtained for mixture of Weibull distributions. Moreover for November 2015 the mixture of Burr and for December 2015 the mixture of log-normal fits the data significantly better. It is worth mentioning that for all months the $p$-values are high and there is no evidence to reject the hypothesis, with confidence level 5%, that our data sample is following the mixture of Weibull distributions or log-normal or Burr distributions. The only exception is the Kupier test for which in most cases $p$-values are around 0. Tests results ensure us that the fit of mixture of two positive distributions can be beneficial in case of modelling of the ore weight. The estimated parameters for the distribution which fits data the best are presented in

![Figure 7: The fitted kernel density estimator for the weights in tonnes distribution for each month between November 2015 and March 2016. The weights smaller than 2 tonnes and bigger than 40 tonnes are ignored.](image)

![Figure 8: Comparison of histograms for real data of the ore mass supplied to the grid, kernel density estimators and several mixtures of fitted distributions.](image)
Table 1: The test statistics and the p-value for different mixing distributions and estimation performed by algorithm presented in Fig. 1.

<table>
<thead>
<tr>
<th>Mix distribution</th>
<th>K</th>
<th>p-val</th>
<th>V</th>
<th>p-val</th>
<th>W²</th>
<th>p-val</th>
<th>A²</th>
<th>p-val</th>
</tr>
</thead>
<tbody>
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<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Gamma</td>
<td>2.13</td>
<td>0.00</td>
<td>4.05</td>
<td>0.00</td>
<td>1.09</td>
<td>0.00</td>
<td>7.19</td>
<td>0.00</td>
</tr>
<tr>
<td>log-normal</td>
<td>1.48</td>
<td>0.01</td>
<td>2.46</td>
<td>0.00</td>
<td>0.35</td>
<td>0.13</td>
<td>2.28</td>
<td>0.03</td>
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<tr>
<td>Weibull</td>
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<td>0.03</td>
<td>2.82</td>
<td>0.00</td>
<td>0.38</td>
<td>0.06</td>
<td>2.99</td>
<td>0.02</td>
</tr>
<tr>
<td>Burr</td>
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<td>0.04</td>
<td>2.58</td>
<td>0.00</td>
<td>0.20</td>
<td>0.21</td>
<td>2.05</td>
<td>0.05</td>
</tr>
<tr>
<td>December 2015</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Gamma</td>
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<td>0.00</td>
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<td>0.95</td>
<td>0.00</td>
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<td>2.02</td>
<td>0.00</td>
<td>0.15</td>
<td>0.40</td>
<td>1.38</td>
<td>0.26</td>
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<tr>
<td>Weibull</td>
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<td>2.35</td>
<td>0.00</td>
<td>0.28</td>
<td>0.06</td>
<td>2.25</td>
<td>0.01</td>
</tr>
<tr>
<td>Burr</td>
<td>1.13</td>
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<td>2.11</td>
<td>0.00</td>
<td>0.18</td>
<td>0.23</td>
<td>1.59</td>
<td>0.08</td>
</tr>
<tr>
<td>January 2016</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
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<td>1.99</td>
<td>0.03</td>
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<tr>
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<td>0.37</td>
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<tr>
<td>Burr</td>
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<td>0.40</td>
<td>2.01</td>
<td>0.00</td>
<td>0.17</td>
<td>0.54</td>
<td>1.26</td>
<td>0.44</td>
</tr>
<tr>
<td>March 2016</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
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<td>3.11</td>
<td>0.00</td>
<td>0.42</td>
<td>0.34</td>
<td>2.43</td>
<td>0.40</td>
</tr>
<tr>
<td>Burr</td>
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<td>0.00</td>
<td>3.11</td>
<td>0.00</td>
<td>0.44</td>
<td>0.03</td>
<td>2.57</td>
<td>0.02</td>
</tr>
</tbody>
</table>

Table 2: Estimated parameters of bimodal probability density function. For each month the best distribution was chosen according to statistical tests.

<table>
<thead>
<tr>
<th>Month</th>
<th>Mixing distribution</th>
<th>Parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td>November 2015</td>
<td>Burr</td>
<td>p=0.55\ a_1=2.81\ c_1=4.52\ c_2=0.36\ a_2=147\ c_3=4.05\ a_3=5287</td>
</tr>
<tr>
<td>December 2015</td>
<td>log-normal</td>
<td>p=0.29\ a_1=1.34\ a_2=0.51\ a_3=2.69\ a_4=0.34</td>
</tr>
<tr>
<td>January 2016</td>
<td>Weibull</td>
<td>p=0.47\ a_1=13.19\ a_2=1.42\ a_3=19.92\ a_4=6.50</td>
</tr>
<tr>
<td>March 2016</td>
<td>Weibull</td>
<td>p=0.48\ a_1=14.76\ a_2=1.47\ a_3=15.90\ a_4=8.50</td>
</tr>
</tbody>
</table>

parameters were estimated using MLE. The summary of the analysis is presented in Tab. 2. One can observe that in December 2015 the log-normal distribution was selected, in November 2015 Burr distribution is chosen, in other months the best fit is obtained for Weibull distribution. Furthermore, the mixing distribution parameters are different for each month. Basing on the estimated parameters of mixture of distributions, the 1000 trajectories were simulated. The results are presented in Fig. 9 together with the quantile lines of order 0.01 and 0.99. Moreover, the trajectories from real weight data related to work of the belt conveyor during one shift (respectively for each month) are presented. The model for each month was fitted separately, thus we assume that ore flow process during one month is the same. Therefore, all shifts should follow the CPP with the same parameters and should fall into simulated quantile lines. Indeed, one can observe that most of the real trajectories are between the quantile lines (Fig. 9). Hence, the proposed stochastic model properly describes the behaviour of the real process. Furthermore, the quantile lines differs for each month, thus analyzed ore in the belt conveyor is changing with respect to time. Moreover, the proposed CPP is able to model the process describing the ore supply process.

5.4 Model validation

As it was mentioned the aim of the analysis is to model the process of ore flow on the grid using compound Poisson process. In previous section the process \( \{N_t\}_{t \geq 0} \) was identified as homogeneous Poisson process with the intensity \( \lambda \). Moreover, the assumption about bimodal probability distribution function of variables \( Y_i \) was confirmed. In the simulation the best bimodal distributions are chosen. The choice is based on on results of statistical tests presented in Tab. 1. For each month the trajectories from real weight data related to work of the mine. Therefore, the proposed can be used as simulation tool for the needs of improvement of both belt conveyors control and QA/QC procedures in the mine. In this paper we have considered the cases, when conveyor is supplied with ore by LHD machines, thus material stream is provided in cyclic way. Modelling of such process should provide an answer for 2 questions: what is a time between consecutive ore supplies and what is volume/mass of ore provided during a single supply. The compound Poisson process was proposed as a mathematical tool for describing the phenomena. The waiting times for the arrival and the jump sizes are modeled. We have shown that for the analysed real data the trajectories fall in the quantile lines from the simulated model which indicates the fitted model is proper for examined vectors of observations. The ability to estimate the flow process gives a possibility to predict the mean value of the transport ore by the belt conveyor, which is supplied by one grid. It is extremely desirable feature in the mine. Therefore, the proposed can be used as simulation tool to play and optimize/tune complex transportation system consisted of LHDs and conveyors throughout the development of the mine.

Figure 9: Quantile lines of simulated trajectories for mixture of distributions determined in Tab. 2 of order 0.01 and 0.99 (blue lines) and real data trajectories (gray lines).
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References


Determining parameters of inclined railway access roads in conditions of iron ore open-pit mines being constructed and implemented

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Abstract—Analysis of opening trenches parameters of deep iron ore mineral occurrences with combined types of transport is conducted. Method of determining parameters of railway crossovers during transportation of rock mass by electric trains is developed. Amount of opening horizons is established, as well as angles of hade of exit roads depending on width of an open pit field.

Keywords: deep open pit mine, mine construction group trench, gradient, railway transport.

1. Introduction
Main features of open pit mines operating deep inclined and steeply dipping mineral occurrences are significant finite depth, gradual increase of depth and amount of opening pit banks, change of transportation volume, transportation of rock mass to borders an open pit mine, significant amount of rock and semi-firm rock, ensuring high stability of open pit edges. Such mineral occurrences are usually opened by a system of joint or group of permanent trenches of internal or mixed beddings. Therefore, justifying parameters of inclined trenches construction is a topical task.

2. Materials and methods
2.1 Implementation of railway transport in mining
Considering experience of enterprises in mining deep iron ore mineral occurrences, method of determining parameters of conducting inclined workings in conditions of mines being constructed and implemented using railway transport as connecting link is offered by National Mining University of Ukraine and Kazakh National Research Technical University.

Certain limits are established while justifying stripping method; these limits ensure minimum rock mass transportation costs, minimum volume of capital mining operations and efficient division of overburden extraction for the entire period of mining an open pit field [1,5]. It was offered to classify extraction workings formation methods by minimum transportation distance of rock mass to destination $l_{mp}$ (km), cost price of its transportation $C_{mp}$ (USD/tons) and storing till dumps $C_{con}$ (USD/tons) according to angles of dip of mineral occurrences, spatial position of those mineral occurrences relatively to the surface and finite depth of open pit field and mechanization of mining-transport operations. Thus, following dependences were derived:

$$H_e \leq H_c; \quad l_{mp} \rightarrow \min; \quad C_{mp} \rightarrow \min; \quad C_{con} \rightarrow \min \quad (1)$$

Route of permanent trenches could be stationary or non-stationary (sliding) depending on the angle of dip of mineral occurrences. Route of permanent trench is usually located on the bottom wall of a mineral occurrence on non-operating open pit edge in its final location and stationary while mining inclined mineral occurrences with a value of an angle of dip close to slope angle of a non-operating open pit edge. Route of permanent trenches is non-stationary at the beginning of mining steep and steeply dipping mineral occurrences, because it is located on one or two operating open pit edges. Routes of permanent trenches become stationary within pit banks, when those pit banks reach their final positions on surface. Usage of non-stationary routes decreases volume of mining-construction operations; however, additional difficulties are caused due to the use of non-stationary routes [2-4]. Such difficulties are described by decrease of upward gradient by 35% compared to a leading one. If an excavator, blasted rock and auxiliary communications are located on a non-stationary crossover, then its width is set.

Experience of operating deep open pit mines shows that in modern economic era, when much attention is paid to environmental protection from negative effects of mining industry, and primarily, to
rational use of land resources, stripping horizons of lower technological zone should be done considering all industrial factors. Main factors are justification of limit contours of open pit mining and productive capacity of a mine by rock mass, method and parameters of overburden excavation, rock mass transportation means by established types, also possibilities of storing waste on the mined area. Conduction and implementation of stripping workings should ensure not only minimum costs for rock mass transportation, but also maximum profit from operating a mine in total.

Depth of external trenches with low thickness of covering soft rocks usually reaches the top of bedrock, and has a value of 30-60 meters. If the thickness of covering rocks is 180-200 meters then the depth of external trenches does not exceed 120-150 meters. Further, railways on upper horizons are looped in most cases with deepening of mining operations and overburden excavation intensity decrease on upper part of a working zone of an open pit mine. Consequently, favorable conditions for continuous supply of trains for loading and their movement in one direction are created. One or two excavators are used on a producing level. Maximum capacity of excavators and trains is reached in such case. Length of platforms \( L_{pa} \) (meters) used for equipping dead end and transitional distribution centres with one and two sided abutment is determined by Eq. (2).

\[
L_{pa} = L_c + (129 - 195) \tag{2}
\]

Length of platform \( L_{pa} \) (m) for dead end centres of double-track crossover with one or two-sided abutment would be calculated using Eq. (3).

\[
L_{pa} = 2L_c + (166 - 293) \tag{3}
\]

where, \( L_c \) – length of train, meters.

For low movement intensity and short routes it would be calculated using Eq. (4).

\[
L_{pa} = L_a + 30 \tag{4}
\]

Maximum depth of an open pit mine \( H_a \) (meters), which is opened by straight crossover from the surface, is calculated using Eq. (5).

\[
H_{a.m} = H_{a.m.c} + (l_{n,c} - l_{mc})p \tag{5}
\]

where, \( H_{a.m} \) – depth of external trench, meters; \( l_{n,c} \) – length of horizon area being prepared for mining, meters; \( l_{mc} \) – length of dead end station, meters; \( K_{m,p} \) – route development coefficient.

Route development coefficient with abutment of haulage track on horizontal platforms is derived from Eq. (6).

\[
K_{m,p} = \frac{n_i}{H_{an}} \sum (h_y + l_{an}l_p) \tag{6}
\]

On soft gradient \( i_s \) (%o) it calculated by using Eq. (7).

\[
K_{m,p} = \frac{n_i}{H_{an}} \sum (h_y + l_{an}l_p) \tag{7}
\]

Values of \( K_{m,p} \) increase if adhesive weight of locomotives rises with number of axis \( n_o \) and leading gradient \( i_p \). At the same time \( K_{m,p} \) does not depend on the value of adhesive weight of a locomotive \( n_o \), and with increase of \( i_p \) changes within small range. Abutment of a track on horizontal platforms is characterized by increase of \( K_{m,p} \) against \( K_{m,p} \) 1.5-3.16 times. Value of \( K_{m,p} \) could be decreased 1.3-1.5 times while increasing height of pit banks \( h_y \) from 15 to 30 meters.

Analysis of known methods of stripping and mining deep mineral occurrences using railway transport shows that route of haulage tracks is characterized by equipping distribution and dead end stations on each horizon. Implementing such method significantly increases rock mass transportation distance; consequently, it limits the scope of railway transport [3-7]. Development of an open pit electric trains allows decreasing length of inclined part of an exit. Wherein, development of mining operations with formation of permanent pit edges on crosscut ends and adjacent areas of frontal pit edges allows placing group stripping workings on those pit edges and constructing a direct crossover for electric trains on each of operating horizon (Fig. 1).

3. Results and Discussion

Three group of horizons could be distinguished within operated open pit mine, each of those horizons is stripped by a corresponding internal separate group trench. Average thickness of first group of horizons consisting of 3-4 units with slope angle of 40-60% is 60-150 meters depending on thickness of covering rock mass. Following 6-7 horizons of second group have average thickness of 200 – 250 meters. Average thickness of third group of horizons, depending on mineral occurrence depth, is 100 – 150 meters. Distance between groups of horizons is limited by the width of open pit field and equals to 750-2500 meters. Further increase of number of horizons being stripped in this direction could be limited only by a width of an open pit field, and inclined exits are formed within the crosscut end of the field. Thus, deep direct crossovers should be constructed by placing the initial part of group trenches on areas of side open pit edges adjacent to the crosscut end. Deep horizons of crosscut end are stripped with distancing of trench outfall to maximum possible interval from that crosscut end of an open pit mine. However, it calls for crossing of routes of adjacent group trenches. More simply, they could be implemented at different levels by constructing special overpasses (Fig. 1b).
First group trench 6 is formed on the crosscut end of an open pit mine and opens horizons from the top wall of a mineral occurrence at a depth $H_1$ (meters), which is calculated using Eq. (8).

$$H_1 = \frac{(B_x - a - R)\omega}{\left(\cos \omega + \tan \beta_p\right)} \quad (8)$$

Length of a first trench $L_{mp,1}$ (meters) up to the intersection with second group trench 7 is calculated using Eq. (9).

$$L_{mp,1} = \frac{(B_x - a - R)}{\left(\cos \omega + \tan \beta_p\right) + 1.57R} \quad (9)$$

where, $a$ – distance from group trench axis to the lower edge of stripped group of pit banks, meters; $R$ – radius of curvature of railways, meters; $\omega = \arcsin \omega = \tan \beta_p$ – angle of slope of first group trench to the crosscut end of an open pit mine or frontal open pit edges (second and third group trenches), degrees.

Second group trench is formed in opposite direction from the first one from the top wall side 3 at a depth $H_2$ (meters), which is calculated using Eq. 10.

$$H_2 = \frac{(B_x - 2H_1 \tan \beta - a - R)\omega}{\left(\cos \omega + \tan \beta_p\right)} \quad (10)$$

Its distance $L_{mp,2}$ (meters) from the initial part to the intersection point with third group trench 8 is calculated from Eq. (11).

$$L_{mp,2} = \frac{(B_x - 2H_1 \tan \beta - a - R)}{\left(\cos \omega + \tan \beta_p\right) + 1.57R} \quad (11)$$

Third group trench if formed from the top wall 2 side at a depth $H_3$ (meters), which is calculated using Eq. (12).

$$H_3 = \frac{(B_x - 2(H_x + H_1)\tan \beta - a - R)\omega}{\left(\cos \omega + \tan \beta_p\right)} \quad (12)$$

Its distance $L_{mp,3}$ (meters) from the initial part to the exit to lower horizon is calculated using Eq. (13).

$$L_{mp,3} = \frac{(B_x - 2(H_x + H_1)\tan \beta - a - R)\omega}{\left(\cos \omega + \tan \beta_p\right) + 1.57R} \quad (13)$$

Overpasses from enforced concrete are constructed at intersection point of group trenches with haulage tracks on horizon 9. Further, light rock mass are spread on overpasses, and railways are placed on overpasses (Fig. 2). Trucks are used for repairs and auxiliary operations, and special track is constructed for such transport. Lower horizons are stripped by sequential advancing of each of the group trench with placement of railways on corresponding mined areas of side pit edges and opposite crosscut end of an open pit mine. Lower 1-3 horizons are mined using trucks.

Implementing railway crossovers with increased gradient on an open pit mine while using electric trains significantly differs from organization of mining-construction operations with current design of transport communications. Thus, construction of exit railways with a gradient up to 160% is associated with necessity of sinking of group trenches by mined pit banks of non-operating open pit edge and intersection with transport communications, used by separate operating horizons.

Moreover, increased gradients of exit tracks create special requirements to railroad tracks, their stability and durability. It is determined, that group trenches should be equipped with concrete monolith path using bitumen-cement layer as ballast. Railroad tracks could be constructed used modernized monolith plates PZDK-72, estimated for 400 tons load. Parameters of stripping trenches using railway transport are shown in table 1.
Table 1 – Parameters of stripping trenches for implementation of open pit electric trains

<table>
<thead>
<tr>
<th>Parameters</th>
<th>Mine field width, m</th>
<th>$i_p = 80 %$</th>
<th>$i_p = 100 %$</th>
<th>$i_p = 160 %$</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1000</td>
<td>1500</td>
<td>2000</td>
<td>3000</td>
</tr>
<tr>
<td>$H_1$</td>
<td>63</td>
<td>100</td>
<td>137</td>
<td>210</td>
</tr>
<tr>
<td>$L_{mp1}$</td>
<td>980</td>
<td>1442</td>
<td>1905</td>
<td>2317</td>
</tr>
<tr>
<td>$H_2$</td>
<td>53</td>
<td>82</td>
<td>113</td>
<td>173</td>
</tr>
<tr>
<td>$L_{mp2}$</td>
<td>855</td>
<td>1217</td>
<td>1605</td>
<td>2355</td>
</tr>
<tr>
<td>$H_3$</td>
<td>44</td>
<td>68</td>
<td>93</td>
<td>143</td>
</tr>
<tr>
<td>$L_{mp3}$</td>
<td>742</td>
<td>1042</td>
<td>1355</td>
<td>1980</td>
</tr>
<tr>
<td>$H_3 =$</td>
<td>160</td>
<td>250</td>
<td>343</td>
<td>526</td>
</tr>
</tbody>
</table>

$H_1 + H_2 + H_3$

$H_1 + H_2 + H_3$

$H_1 + H_2 + H_3$

$H_1 + H_2 + H_3$

$H_1 + H_2 + H_3$

Researches have shown that depth of a direct crossover increases 1.4-2.8 times with gradient increase from 40 to 60-160%. Wherein, transportation distance of rock mass by open pit edge decreases 3-4 times. It should be noted that decrease of open pit dimensions in a design during reduction of mining operations causes decrease of $H_3$ to 10-30% compared with $H_1$ and 20-30% compared with $H_2$. Rise of gradient increases values of $H_1$ 1.45-3.6 times; values of $H_2$ 1.4-2.7 times; values of $H_3$ 1.2-2 times. Traction units PE-2m or OPE-1A operating on direct or alternating current of with corresponding number of dump trucks are used for transportation needs.

4. Conclusions

Method of determining parameters of inclined workings allowed to establish that three horizons could be distinguished within open pit mine being operated, each of those horizons are stripped by a corresponding internal separate group trench. Upper three-four horizons could be maintained by separate exits with gradient of 40-60%, next six-seven with gradient of 160%. Further increase of number of horizons being stripped in this direction is limited only by width of an open pit field.

It is recommended to place an inclined trench on a temporary non-operating open pit edge with an open pit depth up to 100 meters; on an area of limiting contour of an open pit field with a preliminary conduction of advancing workings at a depth from 100 up to 200 meters; on non-operating open pit edge in permanent stationary position at open pit mine depth from 200 up to 300 meters.

Amount of mining-construction operations could be decreased to minimum if steep trenches are constructed on a slope of non-operating open pit edge at an angle of 36-42°. However, high steepness of a route imposes restrictions on the choice of mechanization means used for removal of crushed rock mass. Wherein, excavators could not be used in such case, only bulldozers could be used.

References

Closure Risk Assessment in a Travertine Quarry Mine in Iran

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Abstract—40% of the premature closed mines in Iran are Quarry mines. To identify and evaluate the risks of premature mine closure, a fuzzy fault tree analysis technique is used. To do so, basic events of the fault tree are identified through a Delphi method. Also, probability of each identified basic event is estimated through a fuzzy procedure. Due to lack of precise data, application of fuzzy variables improves the procedure significantly. Finally, fault tree of a quarry mine closure is constructed and probability of top events and the most critical minimal cut set are calculated. To implement the addressed methodology, a case of a travertine mine is studied. In this case, 28 basic events are identified which can cause premature mine closure, and the probability of premature mine closure is 16.8%. Failure to use modern technology, environmental pollutions and marketing reliability are determined as the most critical basic events that elimination of them decreases the closure probability to less than 10%.

Keywords: mine closure, risk assessment, Fault Tree Analysis, quarry mine.

1. Introduction

Human needs to the mineral and energy resources have made mine exploration and extraction inevitable. Mining causes some impacts on environment, economy, and human society. The main environmental problems of the mining operations include soil, vegetation, air, noise and water pollution and ground vibration [1-6]. Closure may happen before the deposit is completely extracted because of some unexpected events. Premature mine closure increases the problems arising from the obsolete mines and lack of proper reclamation planning. Premature closure of quarry mines approximately account for 40% of premature mine closure in Iran [7]. Also, features of some shipments like flammability, toxicity, corrosiveness, and pollution damage the environment and human health during touching, storing and moving [8].

Uncertainty is an intrinsic feature of mining projects, which causes risks and opportunities. Project Management Association defines risk as an uncertain event or an opportunity, if happens, positively or negatively impacts the objectives of the project [9]. Risk management process helps decisions making such as the future events probability and their effects on the project objectives. Risk assessment is a part of the risk management process that provides an organized process to recognize effect of risks. It aimed at providing and analyzing information based on the evidences to make reasonable decisions about how to encounter the specific risks and how to choose among options [10].

Many different methods are proposed for risk assessment. Fault Tree Analysis (FTA) is one of the most suitable risk assessment techniques which can be applied either qualitatively or quantitatively. Fault tree is a graphical model in which different combinations of faults lead to an undesirable event [11]. In FTA, an event is characterized as the top event and its causes are determined with a top down method. Causes of the top event are usually called basic events. The probability of the top event is calculated by using probabilities of all the basic events. The result is uncertain due to imprecise information about occurrence of the basic events. To overcome this level of uncertainty, fuzzy sets theory is useful. Fuzzy sets theory uses judgmental data instead of historical data. Also, it can convert linguistic judgments of experts to quantitative data.

The concept of FTA has been firstly introduced in 1961 by Watson with cooperation of The American Air force and it was developed by Boeing Company during the next year [12-13]. Then, this method has been widely used to analyze the reliability and safety of the systems. Amrozowicz et al. (1997) analyzed the sitting risk of oil tankers [14].

FTA requires data from a broad range of basic events. In practice, it is not possible to access precise data about the basic events because of the inaccurate and uncertain nature of the data. Clemen and Winkler (1999) proposed fuzzy FTA approach in the case of data existence or non-existence about the basic events [15]. Some other previous studies used fuzzy approach in the FTA. Guimarães and Ebecken (1999) proposed a new calculation system in which they used fuzzy FTA [16]. Huang et al. (2000) also used fuzzy FTA approach in the railway system to evaluate the probability of accidents [17]. Yuhua and Dato (2005) used fuzzy logic in the FTA to
assess the leakage probability of oil and gas pipeline [18]. They overcame the lack of precise data in estimating the probability of basic events through fuzzy sets theory. As observed, to assess the uncertain situations with ambiguous data, fuzzy logic uses human judgment about the ambiguous concepts and it can be one of the basic elements in risk assessment [19-20]. There are many applications of FTA in the literature [21-26].

In this research, risk analysis of quarry mines premature closure is performed with a fuzzy FTA technique. In the previous studies, the FTA has not been applied widely in mining industry particularly in analyzing the reasons of premature mine closures, which causes irreparable consequences in the adjacent mineral and urban regions.

2. Methodology
Figure 1 shows the steps of the proposed methodology where risks factors are identified. To identify maximum possible risks including the local problems, a Delphi method has been used, where a team of experts is used, which consists of economics, social, and environmental professionals. Then, the fault tree is constructed to show the relationship between basic events and the top event. In the final step, the effect of elimination of some events is investigated on the premature mine closure risk. Also, some solution procedures are proposed to eliminate these risks.

![Figure 1. The steps of the proposed methodology](image)

2.1. Specification of risk factors
A team of experts has been selected, and two questionnaires are designed. The first one identifies basic events which lead to premature closure and the second one aims at identification of top event. After the first round, the opinions has been aggregated and sent back to the experts.

2.2. Construction of the fault tree
FTA is a top down, deductive failure analysis in which an undesired state of a system is analyzed using Boolean logic to combine a series of lower-level events. After identification of the top and basic events in the previous step, the logical relationships between the basic and the top events are determined in the fault tree. Fault tree is based on AND and OR gates which define the major characteristics of the fault tree. An OR gate shows that the output occurs if any input occurs. An AND gate shows that the output occurs only if all inputs occur (inputs are independent). A sample fault tree is shown in Figure 2. In this diagram, circles in the bottom of the figure show the basic events. The rectangular at the top of the figure shows the top event. The signs which relate the basic events to the top events are logical gates in which sign shows an OR gate and sign shows an AND gate [25].

2.3. Probability estimation
There are 4 steps to estimate the probability of basic events (Figure 3). These steps start by selecting a team including experts and end with probability.

![Figure 2. A sample fault tree](image)

![Figure 3. Estimation process of probability of basic events](image)
mine to judge the probability of the basic events. Accuracy of experts’ opinion depends on their level of expertise. We assign a weight to each selected expert to show their level of expertise. Criteria such as title, experience, education and age are used to determine the weight of the experts [27]. Table 1 shows the scores regarding each criterion.

Table 1- The criterion of scoring the experts

<table>
<thead>
<tr>
<th>Classification</th>
<th>Score</th>
</tr>
</thead>
<tbody>
<tr>
<td>Professor</td>
<td>5</td>
</tr>
<tr>
<td>Expert</td>
<td>4</td>
</tr>
<tr>
<td>City Council Member</td>
<td>3</td>
</tr>
<tr>
<td>Student</td>
<td>2</td>
</tr>
<tr>
<td>Orchard man - Worker</td>
<td>1</td>
</tr>
<tr>
<td>≥ 50</td>
<td>5</td>
</tr>
<tr>
<td>40-49</td>
<td>4</td>
</tr>
<tr>
<td>30-39</td>
<td>3</td>
</tr>
<tr>
<td>20-29</td>
<td>2</td>
</tr>
<tr>
<td>&lt;20</td>
<td>1</td>
</tr>
<tr>
<td>Ph.D.</td>
<td>5</td>
</tr>
<tr>
<td>Master</td>
<td>4</td>
</tr>
<tr>
<td>Bachelor</td>
<td>3</td>
</tr>
<tr>
<td>Technical secondary</td>
<td>2</td>
</tr>
<tr>
<td>school level</td>
<td>1</td>
</tr>
<tr>
<td>≥30</td>
<td>5</td>
</tr>
<tr>
<td>20-29</td>
<td>4</td>
</tr>
<tr>
<td>10-19</td>
<td>3</td>
</tr>
<tr>
<td>5-9</td>
<td>2</td>
</tr>
<tr>
<td>&lt;5</td>
<td>1</td>
</tr>
</tbody>
</table>

2.3.2. Converting linguistic terms to fuzzy numbers

Linguistic terms are used to quantify the ideas of the experts about probability of the basic events. Five linguistic terms are used. Figure 4 and Table 4 shows the fuzzy range of linguistic terms in this study.

![Figure 4. The linguistic terms used by the elite [28]](image)

Table 2- The weights of linguistic terms

<table>
<thead>
<tr>
<th>Linguistic terms</th>
<th>Weight</th>
</tr>
</thead>
<tbody>
<tr>
<td>very low</td>
<td>0</td>
</tr>
<tr>
<td>low</td>
<td>0.1</td>
</tr>
<tr>
<td>medium</td>
<td>0.3</td>
</tr>
<tr>
<td>high</td>
<td>0.6</td>
</tr>
<tr>
<td>very high</td>
<td>0.8</td>
</tr>
</tbody>
</table>

2.3.3. Defuzzification process

The weight of each expert is multiplied by his linguistic term scores (Eq. 1).

\[
\sum W
\]

where \( A_{ij} \) shows linguistic term score of the expert \( j \) for the basic event \( i \). \( w_j \) is the weight factor of expert \( j \), \( m \) is the number of basic events, and \( n \) is the number of fuzzy scores for basic event \( i \). Then, the fuzzy possibility score of the fuzzy numbers is calculated using center of mass formula (Eq. 2) [29].

\[
\frac{-\left(\frac{(a_{ij2})}{a_{ij1}} - \frac{(a_{ij3})}{a_{ij2}} + \frac{(a_{ij4})}{a_{ij3}} - \frac{(a_{ij5})}{a_{ij4}}\right)}{2}
\]

Where \( FPS \) is fuzzy possibility score and \( a_{ij1}, a_{ij2}, a_{ij3}, a_{ij4} \) are prominent points of the fuzzy numbers calculated with the combined fuzzy score of the basic events.

2.3.4. Transforming fuzzy score into probability

Since probability is acceptable in the fault tree, fuzzy scores should be converted from possibility to probability (Eq. 3 and 4) [30-31].

\[
f(x) = \frac{10}{10 - \text{FP}}
\]

Where \( FP \) is failure probability of basic event. To calculate the probability of a top event, the probability of basic events which cause the corresponding top event should be considered. The gate which relates the basic events to a top event also affects the value of the top events’ probability. An OR gate is the logical gate to create the risk if any basic event occurs. The probability of a top event which is connected to corresponding basic events with an OR gate can be calculated with Eq. 5[25]:

\[
\prod \left( \frac{F_{i}}{FP} \right)
\]

where \( F_{i} \) is the failure probability of a basic event, which is connected to the basic events through an OR gate. \( FP_{i} \) is the failure probability of basic event \( i \). Also, if the top event connects to the corresponding basic events through AND gates, the failure probability of the top event is calculated as follows [25]:

\[
\prod \left( \frac{1}{FP} \right)
\]

The route through a tree between a basic event and the top event is called a cut set. The shortest credible way through the tree from the basic to the top event is called a minimal cut set. Probability of the shortest minimal cuts should be used to calculate the probability of the top event. In the case investigated in this paper, all the gates are OR gates. Therefore, probability of basic events and their minimal cut sets are equal. In order to calculate the probability of the top event, Eqs. 5 and 6 are used in OR and AND gates.
respectively. The most critical minimal cut sets are determined via Eq. 7 [24].

\[ \text{significance index of cut set } i \]

Where \( s_i \) is the significance index of cut set \( i \).

3. Case study
Josheghan is located in Kashan, in province of Isfahan in Iran. It is located in the geographical length of 51°13’ E and 2° N and 1356 meter above sea level. Its weather is temperate semi-desert with the average temperature between 25 – 30°. Travertine stone mine is located in a distance of 500 m away from residential areas. Generally, economy of the town is on agriculture, industry, and service. Quarry mines which are near the city damage the environment and cause some struggles between the people and mine owners. Although there are some positive effects of the mine including youth employment and economic boom for the industry owners, the probability of premature closure intensifies their problem.

3.1. Construction of the fault tree
Use of Delphi technique led to recognize 6 main reasons and 28 basic reasons. Table 3 shows the list of basic events leading to premature closure. Figure 5 shows the Bowtie diagram. Figure 6 shows the fault tree of premature closure risk of quarry mine.

3.2. Calculation
15 experts participated in the process of estimation of probability of basic events (Table 4). The score of each expert and the weighting factor of them are calculated and presented in last two columns of Table 4.

The opinion of each expert about each basic event, and an average of opinions is taken into account as fuzzy numbers. Table 5 shows four prominent points of the trapezoidal fuzzy number corresponding to each basic event. These numbers are weighted average of fuzzy numbers which is calculated according to Eq. 1.

Then, the fuzzy numbers are defuzzified through Eq. 2 and the fuzzy possibility score is converted to probability of each basic event through Eqs. 3 and 4. To determine the most critical minimal cut sets, Eq. 7 is used. This score indicates which of the minimal cut sets plays a more critical role in occurrence of the top event. Table 6 demonstrates probability of each basic events, significance score of minimal cut sets and ranking of them among 28 basic events or minimal cut sets. Finally, the probability of occurrence of the top event is 0.168. In other words, Josheghan quarry mine will experience premature closure with probability of 16.8%.

For example, the fuzzy possibility score of the fuzzy numbers of the first event is calculated as Eq. 8. Then, we have 

\[ \text{fuzzy possibility score} = 2.47 \]

and 

\[ \text{failure probability of the top event} = 0.003. \]

Also, failure probability of the top event is calculated using Eq. 9. Finally, significance score of minimal cut sets is calculated. For example, basic event 22 has a significance score of 0.217, which is calculated from

\[ \text{significance score} = 0.217 \]

Figure 5. The Bowtie diagram of premature closure risk of quarry mine
In this paper, the probability of basic and top event is calculated through fuzzy FTA method. After computing the top event probability according to the fault tree described in Figure 6, one can present some solution to some basic events which lead to the top event. It is reasonable to do some actions to eliminate the basic events which have the greatest effect on the top event. In that regard, we have eliminated some most effective minimal cut sets, which are ranked 1 to 5 in Table 3. Table 7 shows the result of estimating top event in the basic situation.

The results show that the reliability of the top event increases by 3% from 0.83 to 0.86. In addition, as 5 critical basic events, BE22, BE6, BE27, BE5, BE4 are removed simultaneously, reliability rate of the top event is increased by 9.6% from 86% to more than 90%. To remove each basic event, its corresponding control measures should be applied. For instance, if the modern technologies such as the new wire cutting and chain cutting machine are used to improve production and quality, the most critical basic event, BE22, will be removed. The second critical basic event is air pollution due to the emission of particles and gases. Air pollution will significantly decrease if all of the dust facilities would be equipped with high efficient collectors with the ability to attract the coarse particles and gases. Air pollution will significantly decrease if all of the dust facilities would be equipped with high efficient collectors with the ability to attract the coarse particles and gases.

4. Discussion
In this paper, the probability of basic and top event is calculated through fuzzy FTA method. After computing the top event probability according to the fault tree described in Figure 6, one can present some solution to some basic events which lead to the top event. It is reasonable to do some actions to eliminate the basic events which have the greatest effect on the top event. In that regard, we have eliminated some most effective minimal cut sets, which are ranked 1 to 5 in Table 3. Table 7 shows the result of estimating top event in the basic situation.

The results show that the reliability of the top event increases by 3% from 0.83 to 0.86. In addition, as 5 critical basic events, BE22, BE6, BE27, BE5, BE4 are removed simultaneously, reliability rate of the top event is increased by 9.6% from 86% to more than 90%. To remove each basic event, its corresponding control measures should be applied. For instance, if the modern technologies such as the new wire cutting and chain cutting machine are used to improve production and quality, the most critical basic event, BE22, will be removed. The second critical basic event is air pollution due to the emission of particles and gases. Air pollution will significantly decrease if all of the dust facilities would be equipped with high efficient collectors with the ability to attract the coarse particles and gases. Air pollution will significantly decrease if all of the dust facilities would be equipped with high efficient collectors with the ability to attract the coarse particles and gases.

Table 3- The basic events leading to the premature closure of quarry mine

<table>
<thead>
<tr>
<th>Basic Event Code</th>
<th>Risk Factor</th>
<th>Basic Event Code</th>
<th>Risk Factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>BE1</td>
<td>The Loss of Landscape Efficiency</td>
<td>BE15</td>
<td>Wind blow in the area</td>
</tr>
<tr>
<td>BE2</td>
<td>Noise pollution</td>
<td>BE16</td>
<td>Rainfall in the area</td>
</tr>
<tr>
<td>BE3</td>
<td>Ground vibration</td>
<td>BE17</td>
<td>Orientation of the area due to wind direction</td>
</tr>
<tr>
<td>BE4</td>
<td>Soil pollution</td>
<td>BE18</td>
<td>Distance between the mine and the area</td>
</tr>
<tr>
<td>BE5</td>
<td>Water pollution</td>
<td>BE19</td>
<td>Dependency of the region on some special industries except mining (e.g. agriculture)</td>
</tr>
<tr>
<td>BE6</td>
<td>Air pollution</td>
<td>BE20</td>
<td>Lack of sufficient knowledge about advantages of mines</td>
</tr>
<tr>
<td>BE7</td>
<td>Disease due to air pollution</td>
<td>BE21</td>
<td>Using traditional methods in native industries (livestock and agriculture)</td>
</tr>
<tr>
<td>BE8</td>
<td>Disease due to vegetation pollution</td>
<td>BE22</td>
<td>Not using new technologies in exploitation of Quarry mines</td>
</tr>
<tr>
<td>BE9</td>
<td>Disease due to livestock</td>
<td>BE23</td>
<td>Low efficiency of mine workers</td>
</tr>
<tr>
<td>BE10</td>
<td>Not using native forces</td>
<td>BE24</td>
<td>Loading and Hauling cost</td>
</tr>
<tr>
<td>BE11</td>
<td>Not awarding loans by government</td>
<td>BE25</td>
<td>Cost of accessibility to the roads and using them</td>
</tr>
<tr>
<td>BE12</td>
<td>Taxation by government</td>
<td>BE26</td>
<td>Cost of repair and maintenance in mines</td>
</tr>
<tr>
<td>BE13</td>
<td>Import and export costs</td>
<td>BE27</td>
<td>Lack of market demand (Market share)</td>
</tr>
<tr>
<td>BE14</td>
<td>Exploration and exploitation costs</td>
<td>BE28</td>
<td>Decrease in quality and final cost of the product</td>
</tr>
</tbody>
</table>

Table 4- The weight factor of each selected experts

<table>
<thead>
<tr>
<th>No.</th>
<th>Title</th>
<th>Age</th>
<th>Educational level</th>
<th>Service time</th>
<th>Weighting score</th>
<th>Weighting factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Professor</td>
<td>&gt; 50</td>
<td>Ph.D.</td>
<td>10-19</td>
<td>18</td>
<td>0.094</td>
</tr>
<tr>
<td>2</td>
<td>City Council Member</td>
<td>40-49</td>
<td>Bachelor</td>
<td>&lt; 5</td>
<td>11</td>
<td>0.058</td>
</tr>
<tr>
<td>3</td>
<td>City Council Member</td>
<td>30-39</td>
<td>Master</td>
<td>&lt; 5</td>
<td>11</td>
<td>0.058</td>
</tr>
<tr>
<td>4</td>
<td>Student</td>
<td>20-29</td>
<td>Master</td>
<td>5-9</td>
<td>10</td>
<td>0.052</td>
</tr>
<tr>
<td>5</td>
<td>Expert</td>
<td>&gt; 50</td>
<td>Bachelor</td>
<td>20-29</td>
<td>16</td>
<td>0.084</td>
</tr>
<tr>
<td>6</td>
<td>Student</td>
<td>30-39</td>
<td>Ph.D.</td>
<td>5-9</td>
<td>12</td>
<td>0.063</td>
</tr>
<tr>
<td>7</td>
<td>Worker</td>
<td>&gt; 50</td>
<td>school level</td>
<td>&gt; 30</td>
<td>12</td>
<td>0.063</td>
</tr>
<tr>
<td>8</td>
<td>Professor</td>
<td>30-39</td>
<td>Ph.D.</td>
<td>5-9</td>
<td>15</td>
<td>0.079</td>
</tr>
<tr>
<td>9</td>
<td>Expert</td>
<td>40-49</td>
<td>Bachelor</td>
<td>10-19</td>
<td>14</td>
<td>0.073</td>
</tr>
<tr>
<td>10</td>
<td>Worker</td>
<td>&gt; 50</td>
<td>Technical secondary</td>
<td>&gt; 30</td>
<td>13</td>
<td>0.068</td>
</tr>
<tr>
<td>11</td>
<td>Worker</td>
<td>&gt; 50</td>
<td>school level</td>
<td>20-29</td>
<td>11</td>
<td>0.058</td>
</tr>
<tr>
<td>12</td>
<td>Professor</td>
<td>20-29</td>
<td>Master</td>
<td>&lt; 5</td>
<td>12</td>
<td>0.063</td>
</tr>
<tr>
<td>13</td>
<td>Expert</td>
<td>40-49</td>
<td>Bachelor</td>
<td>10-19</td>
<td>14</td>
<td>0.073</td>
</tr>
<tr>
<td>14</td>
<td>Professor</td>
<td>30-39</td>
<td>Master</td>
<td>&lt; 5</td>
<td>13</td>
<td>0.068</td>
</tr>
<tr>
<td>15</td>
<td>Worker</td>
<td>40-49</td>
<td>school level</td>
<td>10-19</td>
<td>9</td>
<td>0.047</td>
</tr>
</tbody>
</table>

For expert i, weighting score = title score + service time score + educational level score + age score.
Figure 6. Fault tree of premature closure risk
### Table 5 - The fuzzy numbers of possibility of each basic event

<table>
<thead>
<tr>
<th>Basic events</th>
<th>Prominent points of the fuzzy number</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1</td>
</tr>
<tr>
<td>BE1</td>
<td>0.295</td>
</tr>
<tr>
<td>BE2</td>
<td>0.268</td>
</tr>
<tr>
<td>BE3</td>
<td>0.250</td>
</tr>
<tr>
<td>BE4</td>
<td>0.431</td>
</tr>
<tr>
<td>BE5</td>
<td>0.502</td>
</tr>
<tr>
<td>BE6</td>
<td>0.613</td>
</tr>
<tr>
<td>BE7</td>
<td>0.384</td>
</tr>
<tr>
<td>BE8</td>
<td>0.284</td>
</tr>
<tr>
<td>BE9</td>
<td>0.231</td>
</tr>
<tr>
<td>BE10</td>
<td>0.245</td>
</tr>
<tr>
<td>BE11</td>
<td>0.167</td>
</tr>
<tr>
<td>BE12</td>
<td>0.169</td>
</tr>
<tr>
<td>BE13</td>
<td>0.258</td>
</tr>
<tr>
<td>BE14</td>
<td>0.246</td>
</tr>
<tr>
<td>BE15</td>
<td>0.301</td>
</tr>
<tr>
<td>BE16</td>
<td>0.220</td>
</tr>
<tr>
<td>BE17</td>
<td>0.326</td>
</tr>
<tr>
<td>BE18</td>
<td>0.395</td>
</tr>
<tr>
<td>BE19</td>
<td>0.291</td>
</tr>
<tr>
<td>BE20</td>
<td>0.193</td>
</tr>
<tr>
<td>BE21</td>
<td>0.170</td>
</tr>
<tr>
<td>BE22</td>
<td>0.663</td>
</tr>
<tr>
<td>BE23</td>
<td>0.320</td>
</tr>
<tr>
<td>BE24</td>
<td>0.346</td>
</tr>
<tr>
<td>BE25</td>
<td>0.298</td>
</tr>
<tr>
<td>BE26</td>
<td>0.204</td>
</tr>
<tr>
<td>BE27</td>
<td>0.560</td>
</tr>
<tr>
<td>BE28</td>
<td>0.362</td>
</tr>
</tbody>
</table>

### Table 6 - The probability rate and classification of basic events

<table>
<thead>
<tr>
<th>Basic Events</th>
<th>Probability of basic events</th>
<th>Significance score of minimal cut sets</th>
<th>Critical minimal cut sets</th>
</tr>
</thead>
<tbody>
<tr>
<td>BE1</td>
<td>0.003329591</td>
<td>0.217611465</td>
<td>13</td>
</tr>
<tr>
<td>BE2</td>
<td>0.002919676</td>
<td>0.162635726</td>
<td>17</td>
</tr>
<tr>
<td>BE3</td>
<td>0.002518084</td>
<td>0.122283577</td>
<td>18</td>
</tr>
<tr>
<td>BE4</td>
<td>0.008780809</td>
<td>0.089543723</td>
<td>5</td>
</tr>
<tr>
<td>BE5</td>
<td>0.020349558</td>
<td>0.052245071</td>
<td>4</td>
</tr>
<tr>
<td>BE6</td>
<td>0.027334126</td>
<td>0.044087242</td>
<td>2</td>
</tr>
<tr>
<td>BE7</td>
<td>0.007409726</td>
<td>0.041090457</td>
<td>6</td>
</tr>
<tr>
<td>BE8</td>
<td>0.003106159</td>
<td>0.035162652</td>
<td>16</td>
</tr>
<tr>
<td>BE9</td>
<td>0.002231079</td>
<td>0.032323139</td>
<td>21</td>
</tr>
<tr>
<td>BE10</td>
<td>0.002516523</td>
<td>0.029477853</td>
<td>19</td>
</tr>
<tr>
<td>BE11</td>
<td>0.000852828</td>
<td>0.025254731</td>
<td>28</td>
</tr>
<tr>
<td>BE12</td>
<td>0.000883027</td>
<td>0.020993499</td>
<td>27</td>
</tr>
<tr>
<td>BE13</td>
<td>0.002398766</td>
<td>0.019810784</td>
<td>20</td>
</tr>
<tr>
<td>BE14</td>
<td>0.0021871</td>
<td>0.019559227</td>
<td>22</td>
</tr>
<tr>
<td>BE15</td>
<td>0.00352837</td>
<td>0.018916343</td>
<td>12</td>
</tr>
<tr>
<td>BE16</td>
<td>0.001843276</td>
<td>0.018481382</td>
<td>23</td>
</tr>
<tr>
<td>BE17</td>
<td>0.004244553</td>
<td>0.017371824</td>
<td>11</td>
</tr>
<tr>
<td>BE18</td>
<td>0.006906058</td>
<td>0.014982388</td>
<td>7</td>
</tr>
<tr>
<td>BE19</td>
<td>0.003287312</td>
<td>0.014973097</td>
<td>14</td>
</tr>
<tr>
<td>BE20</td>
<td>0.001676933</td>
<td>0.014272451</td>
<td>24</td>
</tr>
<tr>
<td>BE21</td>
<td>0.001233049</td>
<td>0.013274731</td>
<td>26</td>
</tr>
<tr>
<td>BE22</td>
<td>0.036573878</td>
<td>0.013013057</td>
<td>1</td>
</tr>
<tr>
<td>BE23</td>
<td>0.004954332</td>
<td>0.010967336</td>
<td>10</td>
</tr>
<tr>
<td>BE24</td>
<td>0.005432532</td>
<td>0.00997761</td>
<td>9</td>
</tr>
<tr>
<td>BE25</td>
<td>0.003179263</td>
<td>0.009388587</td>
<td>15</td>
</tr>
<tr>
<td>BE26</td>
<td>0.001577936</td>
<td>0.007336538</td>
<td>25</td>
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<tr>
<td>BE27</td>
<td>0.020552155</td>
<td>0.005253935</td>
<td>3</td>
</tr>
<tr>
<td>BE28</td>
<td>0.005907974</td>
<td>0.005074254</td>
<td>8</td>
</tr>
</tbody>
</table>
This method could be used in initial steps of risk assessment in any mining complex. To do so, a team of experts should be formed to provide required data to assess the problem. Then, all the factors which have effects on the problem are identified as basic events. These factors depend on local conditions. In the next step, the way these factors are related to each other and to the top event are recognized and the fault tree is constructed according to this information. Then, probability of premature closure is calculated according to the probability of basic events.

<table>
<thead>
<tr>
<th>Controlling action</th>
<th>Possibility of occurrence of TE</th>
<th>The reliability of TE</th>
</tr>
</thead>
<tbody>
<tr>
<td>Before the action</td>
<td>0.1680</td>
<td>0.8319</td>
</tr>
<tr>
<td>Elimination of BE22</td>
<td>0.1364</td>
<td>0.8635</td>
</tr>
<tr>
<td>Elimination of BE6</td>
<td>0.1446</td>
<td>0.8553</td>
</tr>
<tr>
<td>Elimination of BE27</td>
<td>0.1506</td>
<td>0.8493</td>
</tr>
<tr>
<td>Elimination of BE5</td>
<td>0.1553</td>
<td>0.8446</td>
</tr>
<tr>
<td>Elimination of BE4</td>
<td>0.1606</td>
<td>0.8393</td>
</tr>
<tr>
<td>Elimination of BE22, BE6, BE27, BE5, BE4</td>
<td>0.0715</td>
<td>0.9284</td>
</tr>
</tbody>
</table>

5. Conclusions

In this paper, premature closure of quarry mine was investigated. A new fuzzy FTA technique was used to identify and evaluate events which cause premature closure of a quarry mine. Basic events of the fault tree were identified through a Delphi method. Also, probability of each of the identified basic events was estimated through a procedure which used fuzzy sets theory. A case of Travertine mine in Iran was studied to show the applicability of the proposed method. In this case, 28 basic events were identified which can cause premature closure of the mine. Moreover, closure probability is 16.8%. Failure to use the modern technology, environmental pollutions, and marketing reliability were determined as the most critical basic events that elimination of them through control measures increases reliability of mine to over 90%.

Although this research significantly improves the previous studies, we can present some directions for the future studies. Use of system dynamics approach to model complex relationships between mine closure and social, economics, and environmental factors can be a strong research opportunity. Also, considering dependency of basic events in the fault tree is another research direction.

References


An innovative initiative to design a feasible mining method in the deepest coal mine of India

Satyendra K Singh, Dilip Kumbhakarb, Harshit Agrawala and Awanindra P Singha

Abstract— In the deepest coal mine of India, the geotechnical problems encountered in the past, sometimes insurmountable, had upset the applecart such that the workings were disused and waterlogged to the brim of the existing shafts. These problems were related to bumps, roof-sloughages even in existing developed galleries with associated problems of degree-III gassiness. Wongawilli method, as practiced in Australia, has been modified to suit the prevailing geo-mining conditions of India using Continuous Miner (CM) deployment with stowing as goaf treatment. This method has been tried in the same form or as a variant of it with caving only and that too outside India.

Empirical and numerical modelling approaches have been briefed in this paper to design this method of mining with sand-stowing. At the same time geotechnical problems have also been considered in the design process. The method consists of designing and developing a large dimension of solid coal block with a pair of chain pillars on both sides of the blocks. Coal can be safely extracted by slicing of the stooks on retreat. These stooks are of dimension 10m x 70m in the case of Chinakuri mine no. 1 (the deepest mine in India), because less than 10m width has been found inadequate from the stability point of view. The cut-out distance is recommended not to exceed 14m during splitting as well as slicing. The method has a potential to scale-up for higher coal production. To start with, this has been fixed at 700 tonnes per day (TPD), utilising the existing old coal evacuation and stowing arrangements, without any extra outlay on resources or infrastructure. This modified and innovative method, first of its kind, would expedite liquidation of about ~72 Mt high-grade coal reserves.

Keywords: Underground coal mining, Wongawilli method, Numerical Modelling, High-stress Regime, Stability Analysis.

1. Introduction
As far as coal production is concerned, the future lies in large scale extraction of underground deposits. As opencastable reserves (i.e. 0-300m range) are being fast depleted in India, the ever-increasing demand for coal may necessarily be met by augmenting deep-seated (>300m depth) underground coal production. The preponderance of such deposits cannot be over-emphasized in national perspectives, as detailed in Figure 1 [1]. It may also be noted that more concerted efforts of exploration by government agencies (like CMPDIL, GSI, and MECL) are now planned and implemented on faster track to increase the coal resources in each category in the depth range of 300-1200m.

Figure 1. The depth-wise break-up of Indian coal resources (2014 data) [1]

This paper presents a detailed analysis of the design of an innovative method for extraction of coal from India’s deepest underground coal mine. In deep mine, one has to deal with the high-stress regime that has a possible fall-out as proneness to bump. In Dishergarh (R-IV) seam at Chinakuri mine no. 1,
the liquidation method has, therefore, to be in conjunction with sand stowing as a goaf treatment. This is done primarily to deal with bump hazards, quite common in the mine, the depth of cover being 650m or above. This paper describes an innovative approach modifying the existing Wongawilli method with caving to stowing, along with CM deployment. The paper includes the design of stooks, sequence of slicing, stowing scheme and a suggested plan for up scaling in addition to providing remedial measures of likely geotechnical problems.

2. Geomining Details
Raniganj Coalfield (RCF) has ten workable coal seams namely R-X, R-IX, R-VIII, R-VII, R-VI, R-VA, R-V, R-IV, R-III and R-II of varying thickness. Chinakuri mine No. 1 lies in the southwestern part of RCF, in the West Bengal state of India, under Sodepur Area, Eastern Coalfields Limited (ECL), a subsidiary of Coal India Limited (CIL). Based on the surface and sub-surface data obtained by exploratory drilling, the Chinakuri block has been deciphered to have a monotonous cover of alluvium soil, interspersed with a number of patches of laterite and three workable seams namely R-X (now exhausted) above the study area, R-VIII and R-IV seams. These seams have been worked from different collieries having different leasehold boundaries within Sodepur area. The mine block is having a few dykes and minor faults, viz. Deoli fault (150m throw) and Patmohna fault (30m throw) having a north-south trend in the southern portion of the mine. There are faults sympathetic to these boundary faults.

With an extractable reserve of ~65 Mt of semi-coking-I grade coal, the R-IV seam is the most potential workable seam of the mine. The R-IV seam had been extracted in the past by longwall method of extraction with hydraulic sand stowing and using individual friction props of 40 tonnes capacity. At the moment, the mine is water-logged (to the brim of existing shafts) and the workings are disused in want of suitable mining method.

Based on the seam folio plans, it is inferred that the seam varies in thickness between 2.64-5.54m. The seam has been developed around 3.3m height (by Dosco road header) in the southern portion and about 4.5m in the northern part. The seam lies virgin in the southern part of the block having a depth more than 600m. Grey to dark grey shale, sandy shale, arenaceous shale, shaly sandstone and fine-grained sandstone makes immediate roof and floor of the R-IV seam. However, during a comprehensive research study [2] by CSIR-CIMFR, it was observed that Rock Mass Rating (RMR) at places were difficult to determine and therefore, support design was done using Tergazhi’s rock load classification [3]. The immediate roof, i.e., predominantly shale (about 1m thick) overlain with sandstone, though observed to be blocky and seamy, was found to have relatively low resistance against slippage at bedding planes-boundaries. The seam is degree-III gassy with the rate of gas emission ranging from 19-25 m³/ton of coal as observed during longwall extraction in earlier panels.

2.1 Ground control problems
The overlay sandstone is very competent and consisting of massive layers of fine/medium-grained sandstone, having a thickness varying from 9m to 23m. The depth of cover of the working ranges from 500-700m. The strata control is a major concern due to the presence of these overlying massive sandstone strata. Due to expected difficult caving situation and also due to associated bump proneness, earlier panels were extracted in conjunction with hydraulic sand stowing. Over 95% bump occurred during depillaring, including 58% on working faces and 27% in split galleries.

Coal bumps are characterised by a sudden release of elastic strain energy accumulated in the rock mass or pillars surrounding the mine workings in a short time period. A bump or burst is essentially a manifestation of excessive stress, accompanied by seismic energy release which results in pillar or abutment collapse, the rotational collapse of cantilevered beds, etc. These events have the potential to inflict threat to the safety of mining personnel and equipment. Production may invariably be disrupted for several days. Nevertheless, the assessment of bump proneness has to be periodically done such that if the values of drill yield test or the determined value of energy index or both exceeds the safe limits, the coal extraction would be suspended, persons withdrawn and after reassessment only, the operation may be resumed [2, 4, 5]. This has been a usual practice world over [6-8]. The mine personnel are trained to use these tests gainfully.

3. Method of Extraction
Being a bump prone seam, the R-IV seam is notorious. Worldover, there are three approaches to deal with bump proneness and at the same time strive for smooth and safe coal extraction, these are:
(a) Longwall (L/W) mining with stowing,
(b) Bord and pillar (B&P) or a variant of it like room and pillar mining, rib system of mining, etc. with stowing,
(c) Stress management dealing with the accumulation of high strain energy and subsequent coal bumps, which are met by-
Fracturing the competent overlying strata with the use of volley firing (blasting) by drilling boreholes from the surface or from neighbouring panels.

Gainful use of yield pillar technique and/or using partial extraction methods.

In case of Chinakur i mine workings, it was found unfortunately that item (a) was not feasible, primarily because-

A. ECL management decided not to go for high capital investment necessary for mechanized longwalling.

B. After the fatal accident and fiasco related to past implementations of the methods as mentioned in item (ii) of (c), the Indian inspectorates are not permitting, trust-deficit on part of implementation by the mine operators in India being the main cause.

Eliminating (c) and (a) as mentioned above, the option left is to design an innovative method of mining with the deployment of CM in conjunction with stowing. Nowhere in the world, the CM deployment, has been done with stowing in the past. However, the method has a potential in its debut application in R-IV seam at Chinakuri mine no. 1, feasible due to following reasons:

i. Adoption of stowing instead of caving would help in mitigating bump proneness and in managing high-stress regime at high depths

ii. Cutting of coal in place of drilling and blasting is a better method as far as mitigation of bump is concerned

iii. The Wongawilli method inherently has the following merits:
   a. Minimum no of entries for the sectionalized proposed panel, therefore less development of the coal in comparison and therefore less gestation period per panel.
   b. Steady and regular movement of straight face with faster retreat
   c. The sand stowing scheme proposed would ensure effective and proper packing of sand, thus, ensuring better mitigation of bumps and management of strata as detailed in section 6.0.

It was, therefore, decided to tweak Wongawilli method, i.e., in a sense mini-longwall method, with stowing scheme to be put in place to extract R-IV seam with due regard to safety and conservation.

4. Design Scheme

In the Wongawilli method of CM application, coal is extracted from stress relieved area. A continuous large block of coal pillar is left between goaf area and development workings as shown in Figure 2 [9, 10]. Coal blocks are mined by driving long splits (here, 70m of length) into the solid coal block, leaving long, narrow stook (of width 7m -12.5m) between split and goaf/barrier. The large coal block pillar (here, 70m x 600m) is extracted on retreat and progressively by the split and lift method. Before and during lifting, i.e., series of slicing, the ribs (3m recommended here) serves as a safety barrier against goaf and it would not be judiciously reduced while retreating and before stowing the slices. In the modified Wongawilli method, lifting is done on both side of the split (of 6m width), hence, there is a concentration of production, and productivity is enormously increased. The other way of modification of this method is to utilise the same entries for two neighbouring Wongawilli panels.

![Figure 2. Wongawilli system modified with dip-to-rise stowing, slicing sequence in the stook under consideration is also shown along with supports only in the split (not to scale)](image-url)
comparatively slow rate of retreat and associated economic concerns.

- Eight inch dia. (or more) pipes may be laid progressively from dip to rise side as the extraction retreats in the split gallery. This will serve as ventilation pathway even when the area is stowed. However the pipes would be lost forever. Ventilation needs to be put on sincere vigilance, monitoring and course correction, in its first application as suggested by the researchers of CSIR-CIMFR [11].

As understandable, the Wongawilli panel consists of coal block pillar formation of dimensions in the range of 70m width, 10m thickness, because less than 10m width has been found inadequate from a stability point of view. It is better to call them stooks, as they are formed after splitting against a barrier (panel-boundary) or against a stowed goaf, as the case may be. The option of higher width i.e., more than 70m is rejected as R-IV seam is degree-III gassiness and has a history of gas outbursts. In future panels, with Coal Bed Methane (CBM) and Coal Mine Methane (CMM) options, if put in place, the higher face width would be designed and implemented.

The panel development shall be made in apparent dip direction (1 in 8) instead of the true dip of the seam (1 in 5) for splits and for interconnections of chain pillars This will help in easy manoeuvrability of CM as well as shuttle cars and other accessories. The level galleries would be at 1 in 200 gradient, for drainage of stowing water from the panel (Figure 3). Mostly, 3-way junctions are formed after splitting which would be suitably supported as per Figure 2. The scheme of operation can be enumerated as follows:

i. The extraction in a sectionalized Wongawilli block can be undertaken by driving splits of 6m width across the panel width (70m) using the place-changing method, as commonly used in case of CM-deployment. The splitting would be in pairs, to optimise CM workings. Splits would be supported by a scheme of full-column resin roof-bolting and also supported at designated places with bolted breaker-lines (as shown in Figure 2) before slicing starts.

ii. The stooks so formed can be depillared sequentially from dip side to rise side with simultaneous stowing of slice as per the scheme (Figure 3). Sump would be made in return having a gradient of 1 in 100 (not shown in Figure 3) to allow easy passage of drainage water.

The CM would cut an estimated length of 3.3m (cutting-width) x 14m (cut-out) x 3.5m (Avg. Stowing Pipe Layout

Figure 3. Part plan showing proposed CM Wongawilli Panel in R-IV seam at Chinakuri mine no. 1 along with scheme of stowing
thickness) in about one hour. Taking slices of 6.6m dimension (i.e., slice no. 1 & 2 together and so on as per Figure 2), would take roughly two to two and half hours. The estimated coal production taking its density as 1.4t/m³, would be around 450 tonnes. Extracting two such slices would be sufficient to meet the daily production requirement of 700 tonnes per day. At a high depth of 600m, the average rate of stowing would be around 100m³/hr. For a slice of 6.6m dimension, around 330m³ void would be created which may be completely stowed in about three and half hours. Thus, extracting two such slices in conjunction with stowing would take, at most, around 7 hours. Adding five hours of slicing and seven hours of stowing, it would take approximately twelve hours while the production demand for 24 hours would be met. The slack time available would expectedly be off-set while dealing likely stowing-pipe bursting or strata control problems e.g., detection of likely bumps by drill-yield tests. On the other hand, the execution-time analysis will provide an insight into the positive potential of upscaling.

5. Scope of Scale-Up
Once this method is successfully demonstrated in an R&D trial panel, the methodology can be scaled-up:

a) By modification to the existing Wongawilli system of mining, wherein coal panel block development (like 70m x 600m dimension), can be extracted by same entry-chain pillars, thereby reducing blockage of coal in the development and also reducing the gestation period for a proposed panel extraction. The modification of method is also possible by lifting both sides of the same split (as mentioned in section 4.0 above). Exigencies related to ventilation will need to be suitably addressed. The length of the panel in the proposed R&D trial study is restricted to 600m due to the presence of a fault on one side and entries already developed on the other side. The length of a Wongawilli block, in future, can be 1 km or more as the incubation period of R-IV seam is 18 months.

b) The rate of stowing can be increased to a much faster rate by putting the dedicated pipe network and replacing the old infrastructure. The design presented in this study report has been based on no additional resource investment as far as stowing and coal evacuation systems are concerned.

c) Fast rate of coal evacuation and transportation system like Continuous Transport System (CTS) will be adopted later on.

6. Stowing Scheme
The Wongawilli method deploying CM in conjunction with sand stowing has been proposed for the given conditions at Chinakuri mine no. 1 as mentioned in section 4.0. Given a high depth of workings and favourable hydraulic gradient (> 1 in 3) at the proposed locations, sand stowing can easily be implemented with favourable results (Figure 3).

Moreover, the dimensions of the stooks in the proposed panel have been designed to have a factor of safety (FOS) more than 1.0, required for long term stability with stowing. However, the effectiveness of the operation will largely depend on the manner and extent of stowing. The followings may be highlighted:

1. At the junctions between the stooks and the slices, the area of exposure is more, hence the goaf would be immediately stowed after the opening of the slices.
2. The goaf edges formed after extraction of the slices would be supported skin to skin by steel/wooden chock supports before commencing the stowing operation in the next slice. The area would be marked with DANGER CORD and persons would not be allowed to enter beyond this cord.
3. The sequence of stowing would be such that after the opening of the slice, the dip side and the voids formed would be completely stowed. Rib would NOT be judiciously extracted.
4. The slices would be stowed following the above three steps. When the final slice in the rise most side is extracted, it would be stowed forthwith and then the left out portions of the split would be stowed.

High-density polyethylene (HDPE) pipe would be used for stowing purposes in the goaf which would be of ~ 20 ft. (6m) length. T-fork type arrangement would be made indigenously which can be mounted over remote-controlled CMs for stowing inside the slices after the extraction is completed. This will result in the proper packing of the goaf especially at such a higher depth of cover and favourable H/L ratio. The T-fork arrangement will also facilitate in the regular withdrawal of pipes from the slices without any person required to go in the goaf area. Stowing arrangements would match with coal evacuation system.

7. Numerical Modelling
The numerical modelling is the only available tool to the researcher for assessing the geo-mining situations for which any empirical norm is not available. Here, the actual and likely geo-mining situations including mine geometry and extraction sequences may be simulated and the results may be obtained making permutations and combinations of the dimensions, influencing parameters and other expected variations. Inter alia, parametric analysis, applying engineering judgements and calibration
with observed facts are some of the requisites for getting rational use of this tool in application worldover. In this paper, 2-dimensional (2-D) numerical modelling has been done using finite-difference software FLAC3D developed by Itasca Consulting Group, Minnesota, USA [12].

Many excavation problems, though three-dimensional, can be idealised as two-dimensional plane strain problems by analysing a vertical as well as horizontal cross-section as shown in Figure 4. Since any two cross-sections are identical along the length and the excavation along the normal to the cross-section is theoretically infinite in length, a cross-section can deform in its own plane but cannot move normally towards another cross-section [13]. Thus, if a cross-section exists in the x-y plane, we have only three stresses \( \sigma_{x}, \sigma_{y}, \) and \( \tau_{xy} \) in this plane, which can be resolved into two principal stresses \( \sigma_{x} \) and \( \sigma_{y} \). The restraint on the movement of the cross-section along z-axis induces a third stress, as-

\[
\sigma_{z} = \nu (\sigma_{x} + \sigma_{y})
\]

Where, \( \sigma_{x}, \sigma_{y}, \sigma_{z} \) are stresses in x, y, z direction respectively (MPa), \( \nu \) is the Poisson’s ratio. Since \( \sigma_{z} \) is always orthogonal to \( \sigma_{x} \) and \( \sigma_{y} \), it becomes the third principal stress. A plane strain situation, thus, always have three principal stresses, though the situation is two-dimensional. The eqn. (1) is suitably changed during modelling incorporating pre-exavation in-situ stresses, measured at Chinakuri mine no. 1 [14, 15].

Figure 4. 2D planar view showing formation of different ribs formed during depillaring and stooks formed at midway of retreat where likelihood of maximum abutment stress would occur (Plan View of plane strain idealisation)

Figure 5. Strata section of immediate roof of R-IV coal seam taken for 2D numerical modelling

While applying a failure criterion, the maximum and minimum principal stresses should be taken considering all three stresses \( \sigma_{x}, \sigma_{y}, \sigma_{z} \). In
case of Chinakuri, it was observed that the 2D modelling could provide a reasonable and meaningful analysis of results. The strain-hardening/softening model allows representation of nonlinear material softening and hardening behaviour based on prescribed variations of the Mohr-Coulomb model properties (cohesion, friction, dilation, tensile strength) as functions of the deviatoric plastic strain [12].

The following geo-mining parameters were chosen taking the worst geo-mining situation at the mine site into consideration for preparing the numerical model:

- Thickness of R-IV seam: 4 m
- Average Wongawilli Pillar size: 70m x 10m (corner to corner)
- Depth of cover: 600 m

The modelling exercises were conducted in the following stages:

Stage 1: Virgin model, after grid formation as per Figure 5, was loaded with the rock properties (listed in Table 1), in-situ stresses, and boundary conditions, as input parameters. The model after loading with in situ stress values is shown in Figure 6.

Stage 2: Development of R-IV seam forming chain pillars and Wongawilli pillars respectively was done as a next step to simulate the workings. The developed seam with vertical stress redistribution (Figure 7).
Figure 9. Vertical stress contour (in Pa) showing redistribution of stress on ribs subjected to depillaring in conjunction with stowing belowground in 2D numerical modelling simulation (as per Figure 4 in XX direction).

Stage 3: Final extraction of R-IV seam in conjunction with proper stowing is shown in Figure 8 (section YY of Figure 4) and Figure 9 (section XX of Figure 4).

Table 1 --- Properties used in the modelling

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Modulus of elasticity, E, GPa</th>
<th>Poisson’s ratio</th>
<th>Rock density Kg/m³</th>
<th>Intact compressive strength, MPa</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone Floor</td>
<td>7</td>
<td>0.25</td>
<td>2535</td>
<td>69.15</td>
</tr>
<tr>
<td>R-IV coal Seam</td>
<td>4</td>
<td>0.25</td>
<td>1415</td>
<td>20.39</td>
</tr>
<tr>
<td>Shale</td>
<td>7</td>
<td>0.25</td>
<td>2444</td>
<td>52.5</td>
</tr>
<tr>
<td>Coarse grained sandstone</td>
<td>7</td>
<td>0.25</td>
<td>2453</td>
<td>67.5</td>
</tr>
<tr>
<td>Shale</td>
<td>7</td>
<td>0.25</td>
<td>2444</td>
<td>52.5</td>
</tr>
<tr>
<td>Fine grained sandstone</td>
<td>10</td>
<td>0.25</td>
<td>2604</td>
<td>131.7</td>
</tr>
<tr>
<td>Sand Stowed goaf</td>
<td>0.8</td>
<td>0.1</td>
<td>1700</td>
<td>--</td>
</tr>
</tbody>
</table>

8. Stability Assessment: Stooks/Fenders
2D-numerical modelling exercises were done for estimating the stress abutments coming on the stooks formed during depillaring in the proposed panel as shown in Figure 8. Expecting the maximum stress abutment to occur when the workings have retreated around 200m, which have been modelled as per line diagram shown in Figure 4. Two different stooks dimensions were considered i.e. 7m and 10m (Figure 8), the latter was recommended as the first option was found to have FOS < 1.0. The angle of CM operation would be 45° from the axis of the split to maintain cut-out distance (≈14m) during slicing.

8.1 Stooks dimension: 10m
The strength of stooks [4]:

\[ S = 0.27 \sigma_c h^{0.36} + (H/250 + 1) (W/h -1) \text{ MPa} \]
\[ = 0.27 \times 29 \times 4^{0.36} + (600/250 + 1) (17.5/4 -1) \]
\[ = 16.22 \text{ MPa} \]

Avg. stress calculated from numerical modelling= 15.1 MPa.

\[ \text{FOS} = \frac{\text{strength}}{\text{stress}} \]
\[ = \frac{16.22}{15.1} = 1.07 \]

It is recommended to support the split as per Figure 2 before any slicing is started in the split stooks (here a dimension of 10m x 70m x 3.5m or 4m, the last being the seam thickness). Breaker line supports with two additional rows of densely spaced (0.8m grid spacing) bolts would be made before slicing operation. The CM operator would always position himself with a remote control device, outbye of the bolted breaker line. In special situations, suitable supports/reinforcements would also be installed in the bad patches/geologically disturbed places and as and when required. In all the support design, the British Standard BS 7861:2007 would be followed in a manner that the assembled nut with its thread would not fail at less than 295 kN load.

9. Conclusions

The Wongawilli method of mining with caving, established in Australia, its variants in South Africa and the USA, has been modified with a matching stowing scheme to make it a feasible method of coal extraction with CM deployment in R-IV seam at Chinakuri mine no. 1. The method described in this paper borrows ideas from the applications of the Wongawilli method abroad, but beyond that the tweaking of the method with stowing made the extraction approach and stowing scheme completely different in pith and substance. Though high production methods, like Wongawilli, has not been used with stowing, it is a maiden attempt, primarily due to the feasible stowing gradient available at the mine site and due to the bump proneness of the seam. Based on design approach detailed in the paper, 10m stooks are to be formed by splitting the Wongawilli block into pairs using the place-changing method of CM deployment. The 7m stooks are not recommended as they are found to be not stable from established FOS criteria in India. The stowing scheme is custom-designed to suit the geo-mining scenario present in Chinakuri mine no. 1 - the deepest mine in India. The mine at the moment has received due permission from the inspectorate based on the research presented in this paper to go ahead with implementing this innovative method for extraction of R-IV seam with a suite of geotechnical instruments (stress cells, rotary tell-tales, remote convergence indicators, roof bolt load cells and indicator props). The site implementation with these instruments, are to be gainfully applied in the “design by
measurement” approach with an aim to establish this innovative method at such high depth of cover.

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References
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New Powerful Copper Ore Companies in Kazakhstan

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Abstract. Kazakhstan ranks first in the world in terms of the reserves of zinc, tungsten and barite, second place in silver, lead and chromite reserves, third place in reserves of copper and fluorite, fourth - in terms of molybdenum reserves, and sixth place - in gold reserves. This allowed Kazakhstan to become the world's leading mining power. The successes of the mining and metallurgical complex (MMC) are well known.

Development of new powerful copper ore deposits is a major contribution to the further development of MMC of Kazakhstan. In 2015-2016 the group of KAZ Minerals has commissioned the world's two largest ore-dressing complexes for the extraction and processing of copper ores - Bozshakol and Aktogay.

The Bozshakol copper deposit is located in the Pavlodar region. The upper part of the deposit is represented by oxidized and kaolinized sulfide ores. This type of ore is processed in a factory with unique technology with the special hammer and roll crushers. The plant's productivity is 5 million tons of kaolinized ore per year. The sulphide factory processes up to 25 million tons of ore per year. The annual copper production in the cathode equivalent is 75,000 tons for 46 years.

The Aktogay copper deposit of stockwork type is located in the East Kazakhstan region. The upper part of the mineralization is represented by oxidized ores, which at a depth of 30-40 m are replaced by sulphide ores. The reserves of sulphide ores are approximately 15 times more the oxidized ore reserves and are estimated at 1.5 billion tons. The output of Aktogaymining and processing plant (MPP) for ore is 28 mln. tons with an annual copper production in the cathode equivalent of 75,000 tons for 54 years.

These quarries reach the highest technical and economic performance indicators. They are provided at the expense of large ore reserves, the integrated use of ore raw materials, the small value of the stripping ratio and the involvement in exploitation of off-balance ores.

Keywords: sulphide ore, oxidized ore, productivity by ore, new technologies, stripping ratio.

1. Introduction

The presence of own reserves of minerals is the guarantor of the stable development of the country's economy, the entire industry, including smart technologies and cyberphysical systems. Regarding the size of these resources in Kazakhstan, it should be noted that in the past century, the geologists of the Republic, under the guidance of Kanysh Imtayevich Satpayev, an outstanding scientist and academician of Academy of Sciences of the USSR, discovered and put on record all the currently exploited mineral deposits. As a result, Kazakhstan ranks first in the world in reserves of zinc, tungsten and barite, second place in silver, lead and chromite reserves, third place in reserves of copper and fluorite, fourth - in terms of molybdenum reserves, and sixth place in gold reserves. Our country is the largest producer of rhenium (second - third place), beryllium (first - fourth place), titanium sponge (second place), tantalum, niobium, gallium, technical thallium, arsenic (third), uranium (first place), vanadium (fifth place), bismuth (sixth place) [1].

The presence of significant natural resources allowed our republic to become one of the ten largest mining countries in the world. Mining and metallurgical complex (MMC) has a great influence on the formation of macroeconomic indicators of the country. The industry accounts for 13% of GDP, 23% - in general industrial production, 48% - in manufacturing products, 20% - in the country's exports [2].

Development of new powerful copper ore deposits is a major contribution to the further development of MMC of Kazakhstan. In 2015-2016 the group of KAZ Minerals has commissioned the world's two largest ore-dressing complexes for the extraction and processing of copper ores - Bozshakol and Aktogay. These copper deposits of stockwork type were explored back in the Soviet period in the 50-60s years of the past century [3].

Bozshakol copper deposit is located at 80 km from Ekibastuz city of Pavlodar region of the Republic of Kazakhstan. In addition to copper, such associated
components as molybdenum, gold and silver are of industrial importance. The deposit is a stockwork stretched for 7 km from the southwest to the northeast, with dividing into two sections - Central and East. Between them there is a zone about 0.5 km wide, free of ore. The upper part of the deposit is represented by oxidized and kaolinized sulphide ores, the processing of which in Soviet times was extremely difficult due to the lack of appropriate technologies.

Currently, this type of ore is processed at a unique in its technology factory for the processing of kaolinized ores with special crushers of hammer and roller type. In powerful hydrocyclones, ore washing and separation into slurry and sand fractions is carried out. The plant's productivity is 5 million tons of kaolinized ore per year. The sulphide factory processes up to 25 million tons of ore per year. The total productivity of the Bozshakol MPP for ore processing is 30 million tons per year with an annual copper production in concentrate is 100 thousand tons.

The Aktogay stock work copper deposit is located in 25 km east of the railroad station Aktogay of the Ayagoz district of the East Kazakhstan region. Here, as at the Bozshakol deposit, the upper part of the mineralization is represented by oxidized ores, which are replaced by sulphide ores at a depth of 30-40 m. Reserves of the deposit were approved in 1980, and amount of sulfide ores is approximately 15 times more than oxidized ores, and their reserves are estimated at 1.5 billion tons [5]. The average annual ore output is 28 million tons, the service life of the mine is 54 years. The annual volume of copper production in the concentrate is 100 thousand tons.

Bozshakol and Aktogay ore-dressing complexes are the largest mining enterprises in the CIS both in terms of extraction of copper ore and in terms of the scale of its processing. The development of these deposits is unique for the CIS countries. Photo of the production infrastructure of the Bozshakol MPP is shown at the fig. 1.

Figure 1. Complex of the production infrastructure of the Bozshakol MPP

2. Main mining and technical characteristics of Bozshakol and Aktogay quarries

As noted above, the Bozshakol copper deposit consists of two sections (see fig. 2). The Central section is characterized by more favorable conditions for the ore body occurrence and the quality of the ore. The oxidized and kaolinized sulphide ores lie close enough to the day surface, and in some cases have outcrops on the surface. In addition, the reserves in the upper part of the Central Section are characterized by a higher content of useful components. In contrast to the Central section, in the Eastern section the ore body practically has no outcrops on the day surface. As a consequence, the amount of stripping work at this quarry in the initial period is significantly greater than at the Central quarry. The ore reserves of the Eastern section are characterized by a lower content of useful components.

Figure 2. 3D image of the quarries Bozshakol: on the left - the Central quarry, on the right - the East quarry

With these conditions in mind, the concept of mining was accepted for implementation at first at the Central quarry, and after its full development -at the Eastern quarry. This concept was modeled in the computer program NPV Scheduler, which allowed to determine both the final contours of the quarries, and the order of development of mining operations (see fig. 2). The main parameters of these quarries and the Aktogay quarry are given in the table 1.

Since at the Central quarry the ores have outcrops on the day surface and do not require stripping, the project has adopted a circular central development system. In this case, each underlying horizon after the passage of the temporary cross-over is prepared by a working trench, oriented along the strike of the outer contour of the ore deposit. After the formation of the pioneer working trench, its bilateral expansion begins: the internal expansion - for the production of mining operations inside the created annular contour and external - for the horizon expansion in order to prepare the underlying horizon for stripping. In accordance with the indicated order of the working area development, the stripping of each new horizon is carried out at the ore zone by creating a temporary sliding cross-over in a place convenient for unhindered development of its reserves and preparation of a site for a new underlying horizon stripping. The slope of temporary crossovers reaches 80‰.

The upper stripping horizons are worked out by longitudinal stopes, located mainly parallel to the contours of the created ring. In the inner space of the ring, mining operations are carried out by the longitudinal both ring and straight stopes.
Thus, the general direction of mining development is envisaged from the central part of the ore body to the quarry boundaries. This makes it possible to create the favorable conditions in the initial period of a quarry operation to accelerate the formation of a stationary part of the exit trenches. As the working area develops, a larger part of the flanks becomes in the limit position, here an opportunity is created for the formation of a stationary part of the track. Further, the gradual setting of the ledges to the limit position makes it possible to form, by the end of the quarrying, a common stationary route with its exit to the surface to the central part of the field. An auxiliary autonomous exit is provided for the personnel evacuation from the quarry in emergency situations.

Situations.

The deposit development is conducted by a deepening system development. To maintain a relatively high content of copper in sulphide ore at the first five years the deepening velocity is 20 to 60 m per year. Then the deepening velocity decreases to 4-6 m per year due to extension of the quarry’s flanks and the stripping of the western and northwest sections of the quarry. The adopted scheme for the mining operations development makes it possible to carry out advanced processing of not only oxidized, but also sulphide ores.

Reserves remaining out of the quarry contour in the amount of 475 million tons are to be worked by underground methods. The deposit opening is realized by processing complexes. The deposit opening is realized by mining operations in the quarry based on the selected economic criteria. The contour of the final quarry is built in most cases on the basis of the economic criterion - the maximum discounted cash flow - NPV. The scheme of the Aktogay quarry at the end of the mining is shown at the fig. 3 [5].

Figure 3. Position of the mine workings at the end of the Aktogay quarry development

Taking into account that the deposit has a significant horizontal square, in the first years of the quarry the amount of mined ore is limited by the increasing productivity of the processing complexes. The deposit opening is realized by temporary spiral and loop automotive cross-overs at the central part of the field. An auxiliary autonomous exit is provided for the personnel evacuation from the quarry in emergency situations.

The deposit development is conducted by a deepening system development. To maintain a relatively high content of copper in sulphide ore at the first five years the deepening velocity is 20 to 60 m per year. Then the deepening velocity decreases to 4-6 m per year due to extension of the quarry’s flanks and the stripping of the western and northwest sections of the quarry. The adopted scheme for the mining operations development makes it possible to carry out advanced processing of not only oxidized, but also sulphide ores.

Reserves remaining out of the quarry contour in the amount of 475 million tons are to be worked by underground methods. The total life of the mine with open and underground works is estimated at 80 years.

Defined by the computer program NPV Scheduler volumes of the annual production and stripping at the Aktogay quarry for the entire service life are given at the table 3. For the selected periods (2017-2022, 2023-2035, ..., 2057-2068) annual volumes of ore and stripping fluctuate within small limits.
Table 2 – Annual volumes of production and stripping rocks at the Bozshakol quarries for the whole lifetime

<table>
<thead>
<tr>
<th>Indices</th>
<th>for 2016-2021</th>
<th>for 2022-2029</th>
<th>for 2030-2035</th>
<th>for 2036-2054</th>
<th>for 2055-2060</th>
<th>for 2061</th>
<th>TOTAL</th>
</tr>
</thead>
<tbody>
<tr>
<td>Balance ore, thousand tons</td>
<td>31 527-30 788</td>
<td>30 880-35 297</td>
<td>28 117</td>
<td>25 340</td>
<td>8 848</td>
<td>6 412</td>
<td>1 104 671</td>
</tr>
<tr>
<td>including oxidized ore, thousand tons</td>
<td>16 663-10 860</td>
<td>1 185-801</td>
<td>299-103</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>49 353</td>
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<tr>
<td>including sulphide ore, thousand tons</td>
<td>28 738-19 360</td>
<td>33 015</td>
<td>27 450</td>
<td>25 050</td>
<td>8 848</td>
<td>6 412</td>
<td>1 055 318</td>
</tr>
<tr>
<td>Cu, %</td>
<td>0.65-0.52</td>
<td>0.39-0.29</td>
<td>0.27-0.29</td>
<td>0.32</td>
<td>0.27</td>
<td>0.27</td>
<td>0.35</td>
</tr>
<tr>
<td>Au, g/t</td>
<td>0.27-0.22</td>
<td>0.16-0.10</td>
<td>0.11</td>
<td>0.07-0.09</td>
<td>0.14</td>
<td>0.06</td>
<td>0.13</td>
</tr>
<tr>
<td>Ag, g/t</td>
<td>1.54-1.31</td>
<td>1.31-3.57</td>
<td>3.04</td>
<td>3.04</td>
<td>3.04</td>
<td>3.04</td>
<td>1.81</td>
</tr>
<tr>
<td>Mo, %</td>
<td>0.0052-0.0043</td>
<td>0.0045-0.0070</td>
<td>0.0070</td>
<td>0.0070</td>
<td>0.0070</td>
<td>0.0070</td>
<td>0.0067</td>
</tr>
<tr>
<td>Off-balance ore, thousand tons</td>
<td>4 750-5 188</td>
<td>5 188</td>
<td>3 162</td>
<td>2 806-16 826</td>
<td>17 650</td>
<td>16 450</td>
<td>344 159</td>
</tr>
<tr>
<td>Ores, total thousand tons</td>
<td>36277-35976</td>
<td>36068-40485</td>
<td>31279</td>
<td>28146-42166</td>
<td>26498</td>
<td>22862</td>
<td>1448830</td>
</tr>
<tr>
<td>Stripping rocks, thousand tons</td>
<td>7865</td>
<td>18 332-15 760</td>
<td>30 800</td>
<td>18 200</td>
<td>16 258-10 258</td>
<td>14 441</td>
<td>856 583</td>
</tr>
<tr>
<td>Rock mass, thousand tons</td>
<td>44142-43841</td>
<td>54 400</td>
<td>54 330</td>
<td>54 400</td>
<td>36 756</td>
<td>24 303</td>
<td>2 305 413</td>
</tr>
<tr>
<td>K_strip, t/t</td>
<td>0.22</td>
<td>0.51-0.39</td>
<td>0.98</td>
<td>0.52</td>
<td>0.51</td>
<td>0.63</td>
<td>0.59</td>
</tr>
</tbody>
</table>

Table 3 – Annual volumes of production and stripping at the Aktogay quarry for the entire service life

<table>
<thead>
<tr>
<th>Indices</th>
<th>for 2016</th>
<th>for 2017-2022</th>
<th>for 2023-2035</th>
<th>for 2036-2044</th>
<th>for 2045-2056</th>
<th>for 2057-2068</th>
<th>for 2069</th>
<th>TOTA L</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sulphide ore, mln tons</td>
<td>1.9</td>
<td>25</td>
<td>36.4-32.8</td>
<td>21</td>
<td>25</td>
<td>25</td>
<td>11.6</td>
<td>1397</td>
</tr>
<tr>
<td>Copper content, %</td>
<td>0.54</td>
<td>0.51-0.41</td>
<td>0.38</td>
<td>0.26</td>
<td>0.29</td>
<td>0.33</td>
<td>0.32</td>
<td>0.348</td>
</tr>
<tr>
<td>Molybdenum content,%</td>
<td>0.010</td>
<td>0.015-0.010</td>
<td>0.010</td>
<td>0.006</td>
<td>0.006</td>
<td>0.007</td>
<td>0.009</td>
<td>0.008</td>
</tr>
<tr>
<td>Gold content, g/t</td>
<td>0.031</td>
<td>0.028</td>
<td>0.028</td>
<td>0.028</td>
<td>0.028</td>
<td>0.028</td>
<td>0.028</td>
<td>0.028</td>
</tr>
<tr>
<td>Silver content, g/t</td>
<td>1.02</td>
<td>0.98</td>
<td>0.98</td>
<td>0.98</td>
<td>0.98</td>
<td>0.98</td>
<td>0.98</td>
<td>0.98</td>
</tr>
<tr>
<td>Oxidized ore, mln tons</td>
<td>12.4</td>
<td>12.4-15.3</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>101.6</td>
</tr>
<tr>
<td>Copper content, %</td>
<td>0.45</td>
<td>0.36-0.33</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>0.366</td>
</tr>
<tr>
<td>Ores, total,mln tons</td>
<td>14.3</td>
<td>37.4-40.3</td>
<td>36.4-32.8</td>
<td>21</td>
<td>25</td>
<td>25</td>
<td>11.6</td>
<td>1498.6</td>
</tr>
<tr>
<td>Stripping, mln tons</td>
<td>1.7</td>
<td>4.5-14.0</td>
<td>6.4</td>
<td>14.0</td>
<td>10.0</td>
<td>1.5</td>
<td>1.0</td>
<td>374.1</td>
</tr>
<tr>
<td>Rock mass, mln tons</td>
<td>16.0</td>
<td>37.4-53.0</td>
<td>39.0</td>
<td>35.0</td>
<td>35.0</td>
<td>26.5</td>
<td>12.6</td>
<td>1872.7</td>
</tr>
<tr>
<td>Stripping ratio, t/t</td>
<td>0.12</td>
<td>0.12-0.35</td>
<td>0.18</td>
<td>0.67</td>
<td>0.4</td>
<td>0.06</td>
<td>0.08</td>
<td>0.25</td>
</tr>
</tbody>
</table>
3. **Technological complexes of mining operations at the Bozshakol and Aktogay quarries**

In connection with the identical conditions of minerals occurrence and mining and geological characteristics of the rocks being developed, the slope of the cross-overs of the stationary track at the both quarries is assumed equal to 80‰. The width of two-lane transport berm is assumed equal to 30 m, taking into account the location of the drainage ditch and the safety shaft. In the bottom part of the quarries one-way cross-overs with a width of 19.5 m are envisaged.

The height of the working ledge is assumed equal to 10 m, of the limit ledge - from 10 m to 30 m. The angle of the ledge slope in the working position is 60-70º, in the limiting position - 50-68º.

The length of the mining work front must be sufficient to ensure the installed capacity of the quarries for minerals and empty rocks. Proceeding from this, the length of the excavator block is assumed to be 700 m.

Parameters of the working platforms are adopted in accordance with the "Methodological recommendations for the technological design of the mining enterprises by the opencast method" [7]. The calculated value of the minimum allowable working platform width in both soft and rock formations is determined taking into account the location of the excavator stope, the collapse of the blasted mass, the safety strips and the safety shaft. The minimum width of the working platform is 34 m. The width of the working platform by laying trenches is 30 m.

In accordance with the capacity of the enterprises for ore and rock mass and adopted technology of mining, the drilling rigs of rotary drilling Sandvik D55SP are used as the main drilling equipment at the both quarries. In addition, the machine Sandvik DI550 is used. Drilling and blasting operations are used not only on rock formations, but on rocks of the weathering zone. However, some volume of rocks of the weathering zone is worked out without the use of drilling and blasting operations.

For the production of blasting operations, the project envisages the use of an explosive of the Igdanite type (ANFO). In addition to the selected explosive, it is possible to use other explosives listed in the List of Industrial Explosives approved for use in the Republic of Kazakhstan.

The processes of charging and boring wells are fully mechanized. Explosion of borehole charges is carried out by non-electric initiation system Rionel. As an intermediate detonator, TNF-hexogen Riobooster or PDP weighing 400 g are used. Initiation of the explosive network is envisaged by non-electrically by means of a starting device for initiating the waveguide or electrically from the explosive typewriter KPM-3U. In all cases, the construction of charges is solid, the fighters are put into charges after charging 0.2-0.5 times the charge length.

Electrohydraulic excavators with a bucket capacity of 22 cubic meters were used as excavating and loading machines at the considered quarries. They carry out loading of rock mass into dump trucks with carrying capacity of 140 tons (see the fig. 4). For implementation of the auxiliary production operations the bulldozers, graders on caterpillar tracks, as well as wheeled bulldozers and front loaders are used.

![Figure 4. Ore loading with a help of an electrohydraulic excavator at the Aktogay quarry](image)

Thus, cyclic mining technology is used at Bozshakol and Aktogay quarries. To implement it, the most widespread in the world excavator-automotive technological complex of open-cast mining is used. The general view of mining operations at the Central quarry of the Bozshakol MPP is shown at the fig. 5 [7]. The transition to the cyclic-flow technology is the subject of the further modernization of technology and technological complexes of mining operations at these enterprises.

![Figure 5. General view of mining operations at the Central quarry of the Bozshakol MPP](image)

4. **Development projects of Bozshakol and Aktogay quarries and their discussion**

Bozshakol and Aktogay quarries are unique. They develop ore formations of huge sizes lying in the vicinity of the day surface. As a result, the sizes of the examined quarries along the day surface, respectively, are 4378x1175, 2500x950 and 2545x2250 m, and the area is 3770, 2045 and 3932 thousand square meters (see the table 1.). By a depth of quarries of 340, 360 and 340 m, 1416, 892 and 1875 million tons of rock mass are extracted from the quarry fields, including 851, 602 and 1500 million tons of ore and 565, 290 and 375 million tons of stripping rock. The average stripping
ratio is 0.66, 0.5 and 0.25. This is an exceptional case in the history of open development of mineral deposits.

According to the schedule of mining at the Bozshakol quarries, starting from the second year of operation, an average of 34 million tons of ore and 25 million tons of stripping rocks are produced annually during 20 years. In 2036-2054 these indices respectively are at the level of 30.1 and 18.2 million tons. etc. (see the table 2). Almost during the entire service life, the annual volume of rock mass undergoes minor fluctuations.

During the last 20 years 16-17 thousand tons of off-balanced ore have been mined annually. This provides a rhythmic work of the quarry. In general, from the ore mass for the entire life of the deposit 3888 thousand tons of copper, 70.23 thousand tons of molybdenum, 140.84 tons of gold and 2834 tons of silver is produced.

In accordance with the mining schedule at the Aktogay quarry, at the period from 2017 to 2035, 35 million tons of ore and 10 million tons of stripping are mined annually in average. From 2036 to 2068, these indicators will be at the level of 25 million tons and 8 million tons, respectively (see the table 3). In the specified stages it is possible to make corresponding modernization of technical means and technologies of mining works and to provide rhythmic work of the enterprise. The total volume of production of copper, molybdenum, gold and silver for the entire life of the quarry are respectively at the level of 5233.3 thousand tons, 114.2 thousand tons, 39.142 tons, 1370.7 tons.

The expected mining and technical results by exploitation of Bozshakol and Aktogay quarries are the highest among the mining enterprises of the CIS and the world. This confirms the experience of these quarries in 2015-2016. They are provided with favorable conditions for occurrence of ore formations, involvement in operation of the off-balance ores and integrated use of mineral raw materials.

At the same time, the height of the ledge (10 m) adopted in the project is not rational, it is greatly understated. With a significant size of the working areas, the use only of automobile vehicles raises many questions.

5. Conclusions
1. Bozshakol and Aktogay deposits of copper ores of stockwork type of Kazakhstan are unique in scale and geometric characteristics of occurrence of ore formations in the earth's crust.
2. The squares of quarry fields on the daily surface of the Central, Northern quarries of Bozshakol and the mine of the Aktogai field are respectively 3770, 2045 and 3,932 thousand square meters.
3. By a depth of quarries of 340,360 and 340 m, 1416, 892 and 1875 million tons of rock mass are extracted from the quarry fields, including 851, 602 and 1500 million tons of ore and 565, 290 and 375 million tons of stripping rock. The stripping ratios are 0.66, 0.5 and 0.25.
4. The average annual productivity of the Bozshakol MPP for oxidized ore is 5 million tons, for sulphide ore - is 25 million tons. Annual output of copper in cathode equivalent is 75 thousand tons for 46 years. The output of Aktogay MPP for ore is 28 mln tons with an annual copper production in the cathode equivalent of 100 thousand tons for 54 years.
5. In total 3888 thousand tons of copper, 70.23 thousand tons of molybdenum, 140.84 tons of gold, 2834 tons of silver will be extracted from ores of Bozshakol MPP. From the ores of Aktogay MPP 5233.3 thousand tons of copper, 114.2 thousand tons of molybdenum, 39.142 tons of gold, 1370.7 tons of silver will be produced.
6. These high results of the operation of Bozshakol and Aktogay MPP will be achieved due to favorable conditions for the occurrence of ores in the earth's crust, the involvement of off-balance ores and the integrated use of mineral raw materials.
7. Modernization of mining technologies at these enterprises with the transition to the combined road and rail transport is undoubted.

Acknowledgement
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References
Landscape heterogeneity – the quantitative attribute of a new landscape of reclaimed areas

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Abstract — Rehabilitation of post-mining landscapes is a multidisciplinary issue. It is a great challenge to create a new landscape that can improve the landscape character and ecology of the region. Restoration of a landscape that was destroyed by mining or similar activities is very often understood primarily as a technical problem with the goals of finding economical solutions for achieving a few simple objectives: (1) stabilization of land surfaces, (2) pollution control, (3) visual improvement, and (4) creating a general amenity in order to preserve the composition of interest and to prevent the land from being unpleasant for the people that use it. However, successfully restoring the original ecosystems or creating new ecosystems involves re-introducing biodiversity and spatial composition (landscape heterogeneity).

In the long term, the major contribution of landscape ecology is to provide a wise way to mesh nature and culture. This way should recognize the pillars of sustainability (Equity, Environment, and Economy) and balance human use with the protection of resources for future generations. For successful revitalization of post-mining landscapes, the reintroduction of target species is a basic issue. Generally speaking, the total species diversity is higher in heterogeneous landscapes that provide many ecosystem types; each of them with characteristic distinctive biota.

Many authors refer to the importance of landscape heterogeneity for the biotic well being of the territory. From the ecological point of view, landscape heterogeneity can be defined by the following relevant attributes:

- Diversity of types of landscape elements;
- Intensity of interaction among landscape elements;
- Size and form of these elements;
- Spatial configuration of the elements;
- Character of relation of the elements; and
- Dynamics of the changes in the characteristics mentioned above.

Our case study (the North Bohemian brown coal basin located near to the frontier between the Czech Republic and Germany) presents various designs of land-use pattern and process for the study area using the criterion of landscape heterogeneity based on detailed knowledge of the present state of the surrounding area and the historical state of the territory. The information gained from the studies of adjacent areas, and from the historic study of the study area itself, was used not in order to reconstruct the area as it had once been, nor even in order to reproduce a situation similar to that in nearby areas.

Keywords: landscape heterogeneity, biodiversity, landscape ecology, Czech Republic.

1. Introduction

Open cast mining leads to the loss of the man-made cultural landscape above the deposits, which is the result of centuries of development [1] [2]. The re-cultivation of abandoned opencast coal mines (post-mining landscapes) is very difficult. It is a great challenge to create a new landscape that can improve the design of the region, or even become a nature reserve [3]. Restoration of landscape destroyed by mining or similar activities is very often understood as a technical problem only, a matter of finding economical ways of achieving a few simple objectives: (1) stabilization of land surfaces, (2) pollution control, (3) visual improvement, and (4) general amenity, in order to preserve the structures of interest and to prevent the land from being unpleasant for the people that use it (Bradshaw, 1987). However, successfully restoring the original ecosystems or creating new ecosystems involves adding biodiversity and spatial composition (landscape heterogeneity).

Forman [4] believes that the long term the major contribution of landscape ecology will be to provide a wise way to mesh nature and culture. This way should accept the concept of sustainability and balance human use with the protection of resources for future generations [5] [6]. Reintroduction of target species is a basic issue for successful revitalization of post-mining landscapes. Generally speaking, the total species diversity is higher in heterogeneous landscapes that provide many ecosystem types, each of them with characteristic distinctive biota [7]. The theory of island biogeography may also be relevant as a theoretical background [8], and the concept of territorial systems of ecological stability can provide a practical basis for recreating a functional network of large heterogeneous patches and connected corridors [9].

Many authors refer to the importance of landscape heterogeneity for the biotic well being of the territory, for example: [10] [11] [12] [13] [14]. From the ecological point of
view landscape heterogeneity can be defined by the following relevant attributes [15]:

- Diversity of types of landscape elements;
- Intensity of interaction among landscape elements;
- Size and form of these elements;
- Spatial configuration of the elements;
- Character of relation of the elements; and
- Dynamics of the changes in the characteristics mentioned above

Having access to current and historical land-use information is vital for landscape assessment and for the formulation of meaningful management plans [16]. Statistical land use data only gives information about landscape macrostructure; it does not provide an accurate idea of an actual spatial composition on landscape elements [17].

1.1 Mining Reclamation Practices

The different phases of mining and reclamation of an open cast coal mine actually begin with the phase of technical mining; the stages of reclamation begin simultaneously with the development plans for the eco-technical phase, the biological phase and finally post reclamation. During the phase of technical mining, surveys of the mining area are made to save as much of the useful topsoil through selective stripping and storage of the fertile top layers. Both hydrological and archaeological surveys are made at this time are critical in the role that they will play in final reclamation process and the associated benefits for creating future diverse heterogeneity in the restoration of the landscape.

Ecotechnical Phase

During the Ecotechnical phase, reshaping of the post mining landscape is conducted to actually re-sculpt the ground plane. Technical erosion control modification, creation of drainage for surface water runoff, establishment of new roads and pathways, and then replacement with the former topsoil layer which was stored nearby during the phase of technical mining is conducted.

Biological Restoration Phase

During the Biological Restoration phase, one or more of four different types or techniques are typically used. These include:

1. Forest reclamation: planting of 2-4 year old new tree seedlings in a grid of 1 x 1.2 meter square patterns with a density of approximately 8,300 trees per ha. Only endemic tree species are selected and receive up to nine-years of active management, however it is typical that they experience a 40% mortality rate due to browsing by deer species, dry soils and other environmental factors.
2. Agricultural reclamation;
3. Hydric or water reclamation; and
4. Other types of reclamation.

Agricultural Reclamation

Agricultural Reclamation is a common technique employed in the restoration of former mining sites, however it is typically used for period of just five to eight years. Crop rotation involves the use of a clover and grass combination intended to add organic matter and nitrogen to soil structure. A variety of agricultural land structure includes:

1. Meadows;
2. Arable land with typical row-crops;
3. Fruit bearing orchards;
4. Vineyards; and
5. Energy crops for bio-mass combustion such as rape wheat.

Hydric Reclamation

Hydric Reclamation is perhaps one of the most commonly employed techniques for restoration of former open cast pit brown coal mines in the Czech Republic. The desire to create large new lakes, ponds and other water features is perhaps understandable in a land-locked country. Of note is Lake Most, with a surface area of nearly 335 ha and a depth of 150 meters is mammoth new water body that now occupies the former Vrbensky and Ležáky mine that occupied the former historic city of Most; the city was entirely eradicated from the landscape with a new modernist urbanized development created nearby. The large lake at Most is intended to serve as a national recreation and scenic amenity with future economic development potential. Other locations include Michal lake, Medard lake and Chabařovice lake.

Whereas diverse or meaningful heterogeneity is not present within a water body, the landscape immediately surrounding the lake certainly can and does support diversity of plans and animal species.

Other Types of Reclamation

Other types of reclamation are of a non-productive (agricultural) character aimed at increasing landscape diversity, providing ecological stability, recreation activities and economic development. Specifically, some of these other types of reclamation techniques include the following:

1. Public green areas, such as vegetation in sports and recreational locations such as a hypodrome for horse racing events;
2. Buffer areas along streams, ponds and lakes;
3. Larger recreation areas such as play grounds, stadiums and swimming pools;
4. Cultural and education areas such as zoological parks and botanical gardens; and
5. Study areas such as an environmental education center, etc.

1.2 Near Natural Restoration vs. Technical Reclamation of Mining Sites

Recent trends in restoration of post mining sites are now leading in a new direction, which employs the use of Ecological Succession as the primary driver. In Germany, the Netherlands, the UK and other western EU countries, Ecological Succession is commonly used as part if not all of the landscape reclamation process. In Germany, for example, 15% of the disturbed land area is intentionally left to naturally restore itself over time. However in the Czech Republic, the process of Ecological Succession has not been fully respected as a viable alternative to the more commonly practiced restoration techniques described above – biological, hydric and agricultural site remediation.
With any new approach to solving a problem, such as how one restores a very large land mass that has been severely compromised or completely destroyed from what might have existed prior or mining extraction, there exist strengths and weaknesses in both application and within the scientific community of experts.

**Strengths**
- Ecologically valuable habitats are created by means of natural succession within the landscape and not through highly mechanized processes;
- Succession helps to increase local biodiversity as opposed to large single-species tree and grassland plantations.
- A greater level of valuable ecosystems and their associated ecosystem services are created in comparison with technical reclamation (biological, hydric and agricultural);
- The financial requirements are significantly less with Ecological Succession. For example, restoration can average from between 400 to 2000 Euro per/ha vs. 4000 to 30,000 Euro per/ha.

**Weaknesses**
- Leaving the post mining landscape in close to the same conditions that exists once mining operations have concluded is requires much more time before nature can begin to heal the site and natural succession processes are well established.
- Is the terrain now so irregular that it limits the potential for some forms of passive recreation?
- How might humans use this landscape?
- What are the management requirements for the future landowners?
- Does this approach represent a simple, cheap way out of what should be a much more complex and active process of site restoration?
- Has Ecological Succession in post mining landscapes been actively practised long enough to adequately access the success rate of species regeneration of plans and the associated animal habitat?

In the Czech Republic, there is an effort by scientists, non-governmental organizations and mining companies to increase the proportion of near-natural restoration measures in post-mining sites, but legislative barriers limit its broad scale application. In spite of these barriers toward broader implementation, there are a number of post mining landscape sites in the Czech Republic where ecological succession was the intended practice and as such, the landscape displays a high degree of heterogeneity. Likely the future will see a compromise between technical reclamation and ecological succession with approximately 25% of the disturbed area allowed to naturally regenerate based on the principles of biological succession.

2. **Landscape heterogeneity** – a key to higher biodiversity of reclaimed sites (Case study)

### 2.1. Study area

The North Bohemian brown coal basin is located near the frontier between the Czech Republic and Germany. It forms part of a wider region known as the Black Triangle.

The post-mining study area of Chabarovice (1400 ha) is situated in the eastern part of the North Bohemian brown coal basin (Figure. 1). Mining was abandoned in 1990. At present, the final arrangement of the relief is under discussion. The selected proposal will be a wet variant, involving flooding an area of 248 ha around the disused pits.

![Figure 1: Location of the study area](image)

### 2.2. Analyzed attributes

GIS tools were used and aerial photographs and maps were interpreted for the following attributes:

- Relative length of edges;
- Average size of landscape elements;
- Relative frequency of landscape elements;
- Land use type richness; and
- Establishing an index of landscape heterogeneity.

The relative length of edges (km.ha\(^{-1}\)) was calculated as the total length of the boundaries between the different land-use types (km) divided by the total area (ha). The average size of landscape elements is the average area of the patches and corridors (ha). The relative frequency of landscape elements is the total number of patches and corridors (No.) divided by the total area (ha). Land-use type richness (No.) is the number of different land-use types within the study area.

### 2.3. Data collection and analysis

Analysis of aerial photographs and maps provide good information about the structure of landscape mosaics [18] [16]. Aerial photographs, 1: 10,000 topographic maps, and 1: 2,880 historical maps were interpreted for land uses. The interpretation of the aerial photographs was checked by terrain reconnaissance of the study area.

Historical stable cadastre maps were used to interpret the landscape structure in 1842. They represent the earliest source of relevant landscape structure attributes for the study area. Aerial photographs and 1: 10,000 topographic maps represent the most recent state of the study area. The landscape structure...
was assessed using ten land-use types from the official land-use classification scheme of the Czech Republic (Table 1).

Table 1 – Official land-use classification modified for interpretation of aerial photographs and 1:10 000 maps using GIS tools.

<table>
<thead>
<tr>
<th>Land-use categories</th>
<th>1. Arable land</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>2. Hop-field</td>
</tr>
<tr>
<td></td>
<td>3. Vineyard</td>
</tr>
<tr>
<td></td>
<td>4. Gardens</td>
</tr>
<tr>
<td></td>
<td>5. Orchard</td>
</tr>
<tr>
<td></td>
<td>6. Grassland</td>
</tr>
<tr>
<td>7. Forest land</td>
<td></td>
</tr>
<tr>
<td>8. Water area</td>
<td></td>
</tr>
<tr>
<td>9. Urban area</td>
<td></td>
</tr>
<tr>
<td>10. Others</td>
<td></td>
</tr>
</tbody>
</table>

The historical maps and aerial photographs were scanned as raster images, transformed and digitised as polygon boundaries and labels into the Geographic Information System Topol for Windows vs. 5.5. The map and aerial photograph features were assigned to ten land-use types as polygon attributes, and the boundaries between the different land-use types were generated as line attributes. The data was checked for errors and processed to give a polygon topology. The values of landscape structure attributes for the present state of the two comparative areas and also for the historical state of the study area were analysed and calculated.

3. Results and Discussion

3.1. Analysis of land-use pattern in three comparative areas

Figure 2 shows the location of the study area and two comparative areas. Seven land-use categories were identified from the aerial photographs and 1: 10,000 maps in the territory surrounding the study area. Seven land-use categories were identified from the aerial photographs and 1: 10,000 maps in the territory surrounding the study area. Seven land-use categories were identified from the stable cadastre 1: 2,880 historical map. This map shows the state of the landscape in 1842 prior to any significant urban development, land reforms during the post World War II era, and the opening of the open cast coal mine (Figure 3 and Figure 4).

Figure 3: Raster image of stable cadastre map 1842 [19].

For each of the three comparative areas, arable land is the dominant land-use type (matrix) because it represents the majority of the total land area and because it demonstrates landscape connectivity. The second most abundant land-use type is grassland (C1 and stable cadastre) or forest land (C2). Table 3 shows the values of the attributes of landscape heterogeneity for each of the three comparative areas.

The results confirm that in 1842 arable land accounted for the highest percentage, and that there was much greater landscape fragmentation meaning that large functional units of biological centers and connectiveness were already compromised by the practice of agriculture. In contrast, the percentage of edges is lower in the stable cadastre than in the present state (Only the border lines between the different land-use types were taken into account). However, the most relevant comparative data is provided by the analysis of the present state of areas, C1 and C2, because the adjacent landscape is
ecologically well balanced and because it presents recent natural and cultural conditions of the study area.

3.2. Proposal for the land-use pattern in the study area

After analyzing the land-use pattern in the three comparative areas, we have proposed a new character of landscape for the study area using three distinctive variants (Figures 5, 6, and 7). Although arable land is a dominant land-use type in each of the three comparative areas, grassland was selected as the land-use type for the matrix in our proposals (more than 40%). It allows free spreading of target organisms across re-cultivated area and it is an efficient remedy for water erosion. The second most abundant land-use type is forest (about 30%). The three variants (V1-V3) are graduated according to land-use pattern fragmentation. While the percentage of land-use types is almost equal, the number of elements increases from V1 to V3 (Table 4).

Figure 4: Land-use types interpreted from the stable cadastre map (1842)

Figure 5: Three proposed variants for landscape structure. Variant 1

Figure 6: Variant 2

Figure 7: Variant 3
Table 2: Summary of land-use analysis showing the number, area and percentage for each of the land-use categories.

<table>
<thead>
<tr>
<th>Land-use categories</th>
<th>1999 Comparative area C1</th>
<th>1999 Comparative area C2</th>
<th>1842 Stable cadastre</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Number</td>
<td>Area (ha)</td>
<td>%</td>
</tr>
<tr>
<td>Arable land</td>
<td>30</td>
<td>700.7</td>
<td>64.0</td>
</tr>
<tr>
<td>Gardens</td>
<td>7</td>
<td>11.6</td>
<td>1.1</td>
</tr>
<tr>
<td>Orchards</td>
<td>7</td>
<td>36.7</td>
<td>3.4</td>
</tr>
<tr>
<td>Grassland</td>
<td>59</td>
<td>205.2</td>
<td>18.6</td>
</tr>
<tr>
<td>Forest land</td>
<td>12</td>
<td>49.6</td>
<td>4.5</td>
</tr>
<tr>
<td>Ponds</td>
<td>14</td>
<td>89.3</td>
<td>8.2</td>
</tr>
<tr>
<td>Rivers, streams</td>
<td>28</td>
<td>2.1</td>
<td>0.2</td>
</tr>
<tr>
<td>Others</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

Table 3: Landscape heterogeneity expressed by some attributes for three comparative areas.

<table>
<thead>
<tr>
<th>Attributes of Landscape heterogeneity</th>
<th>1999 Comparative area C1</th>
<th>1999 Comparative area C2</th>
<th>1842 Stable cadastre Sc</th>
</tr>
</thead>
<tbody>
<tr>
<td>Relative length of edges (km.ha⁻¹)</td>
<td>0.10</td>
<td>0.12</td>
<td>0.08</td>
</tr>
<tr>
<td>Average size of landscape element (ha)</td>
<td>6.98</td>
<td>6.93</td>
<td>6.71</td>
</tr>
<tr>
<td>Relative number of elements (per ha)</td>
<td>0.14</td>
<td>0.14</td>
<td>0.15</td>
</tr>
<tr>
<td>Number of land-use types</td>
<td>7</td>
<td>7</td>
<td>7</td>
</tr>
<tr>
<td>Index of landscape heterogeneity</td>
<td>3.31</td>
<td>3.28</td>
<td>3.74</td>
</tr>
</tbody>
</table>

Table 4: Land-use characteristics of the study area - three proposed variants.

<table>
<thead>
<tr>
<th>Land-use categories</th>
<th>Variant V1</th>
<th></th>
<th>Variant V2</th>
<th></th>
<th>Variant V3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Number</td>
<td>Area (ha)</td>
<td>%</td>
<td>Number</td>
<td>Area (ha)</td>
</tr>
<tr>
<td>Arable land</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Gardens</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Orchards</td>
<td>4</td>
<td>55.7</td>
<td>3.9</td>
<td>9</td>
<td>57.2</td>
</tr>
<tr>
<td>Grassland</td>
<td>10</td>
<td>636.1</td>
<td>44.3</td>
<td>23</td>
<td>596.1</td>
</tr>
<tr>
<td>Forest land</td>
<td>10</td>
<td>451.3</td>
<td>31.4</td>
<td>63</td>
<td>423.9</td>
</tr>
<tr>
<td>Ponds</td>
<td>7</td>
<td>292.0</td>
<td>20.3</td>
<td>7</td>
<td>292.0</td>
</tr>
<tr>
<td>Rivers, streams</td>
<td>29</td>
<td>2.1</td>
<td>0.2</td>
<td>29</td>
<td>2.1</td>
</tr>
<tr>
<td>Others</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

Table 5: Landscape heterogeneity expressed by some attributes for the study area - three proposed variants.

<table>
<thead>
<tr>
<th>Attributes of Landscape heterogeneity</th>
<th>Variant V1</th>
<th></th>
<th>Variant V2</th>
<th></th>
<th>Variant V3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Relative length of edges (km.ha⁻¹)</td>
<td>0.06</td>
<td>0.08</td>
<td>0.09</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Average size of landscape element (ha)</td>
<td>23.95</td>
<td>10.73</td>
<td>7.22</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Relative frequency of elements (per ha)</td>
<td>0.04</td>
<td>0.09</td>
<td>0.14</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Land use type richness</td>
<td>4</td>
<td>6</td>
<td>6</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Index of landscape heterogeneity</td>
<td>1.14</td>
<td>2.16</td>
<td>3.01</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
For each of the variants the values of the landscape heterogeneity attributes were calculated (Table 5). Only the third variant V3 is characterized by values that approximate those in the comparative areas.

The final arrangement of the landscape design for the study area was based on aesthetic criteria. The new form of man-made relief takes into account, above all, the relation to adjacent landscapes and the accentuation of the surrounding mountain peaks. The main visual corridors were defined and visual diagrams were calculated and constructed by analyzing a 3D model of the study area. Woodlands were eliminated from visually highly exposed places.

4. Conclusions

Re-cultivation of post-mining areas is a specific application of landscape planning. Using landscape heterogeneity as a criterion for analysis leading to a proposal for renewal of the landscape design, we quantified the relevant characteristics of the proposed landscape structure. The aim was to harmonize the re-cultivated post-mining landscape with the ecologically valuable surrounding area.

From the analysis of landscape heterogeneity it was clear that the first and second variants (V1 and V2) for the proposed land-use pattern of the study area do not fit in with the adjacent areas (see Table 5 and Figure 5). Our final proposal (variant V3) places special emphasis on the creation of spatial and qualitative (land-use types) conditions for successful reintroduction of target species in the post-mining area. The arable land was identified as the predominant matrix in all three comparative areas. However, it is not a suitable land use characteristic of the matrix for the re-cultivated area because it can be a barrier for some wildlife species and limits overall connectivity. Arable land on unconsolidated slopes is also much more threatened by water erosion than grassland. For this reason we proposed grassland as a land-use type for matrix that will improve the biodiversity of the region as a whole.

Variant V3 complements the present state of adjacent environment, which provided very significant quantitative data. The historical state of the study area provided information of limited present-day relevance, because it cannot take into account recent natural (relief, soil), economic, and social conditions. However, the historical maps were useful for locating some new landscape elements in variant V3 and for interpreting original land-use types in the study area.

The landscape heterogeneity of the adjacent environment is a relevant quantitative criterion for restoring post-mining areas, provided that the surrounding landscape is ecologically well balanced. If not, as in our case study, we have to attempt to improve the ecological conditions of adjacent environment too.

The development and application of GIS tools allows a thorough analysis of spatial attributes and formulation of multi-variation scenarios. This approach enables proposals to be made on the basis of detailed knowledge of a wide range of factors, and could usefully be applied also to planning and decision processes.

References

[18] Ihse, M., (1988). Air photo interpretation and computer cartography - tools for studying the cultural landscape. In:

Geostatistical Simulation of Cross-Correlated Variables: a Case Study through Cerro Matoso Nickel-Laterite Deposit

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²The Robert M. Buchan Department of Mining, Queen’s University, Kingston, ON, Canada

Abstract—Geostatistical methods have been increasingly used as powerful techniques for predicting spatial attributes and modelling the uncertainty of predictions in un-sampled locations, especially through multi-element deposits. Independent Gaussian simulation constructs precise outputs over each variable, in most cases by simulating using the multi-Gaussian assumption. However, this approach does not consider the underlying correlations between the variables. Spatial uncertainty can also be quantified by co-simulation, where the relationship of the co-regionalized variables is accounted for and the spatial relationships between variables are reproduced. In this study, we apply the two aforementioned approaches (independent simulation and co-simulation) for modelling two correlated elements (Fe & MgO) at Cerro Matoso S.A. Nickel laterite deposit located in Colombia. Results show that co-simulation provides a reasonable outcome in regards to the correlation coefficient parameter and relative error as expected.

Keywords: Multivariate geostatistics, Co-simulation, Nickel-Laterite Deposits.

1. Introduction

Geostatistical methods have been increasingly used as powerful techniques for predicting spatial attributes and modeling the uncertainty of predictions in un-sampled locations, especially through multi-element deposits, which are important in mineral resource estimation and ore reserve evaluation [1, 2, 3, 4, 5, 6, 7, 8, 9 and 10]. Independent Gaussian simulation construct precise outputs over each variable [11, 12, 13], and most of them can be simulated separately by transformation to the Gaussian (or multi-Gaussian) distribution. But the problem of applying this approach in multi-element deposits is that it does not consider the intrinsic correlation coefficients between co-regionalized variables. In a nutshell, ore body evaluation in multi-element deposits requires considering the characterization of cross-correlated variables observed at available datasets. Quantifying the uncertainty at un-sampled locations encourage geostatistical modeling of these co-variables. This modeling can be divided into two parts. The first one uses cokriging methods [14, 15] and the second one assessing the local uncertainty by applying co-simulation. The later generates some realizations, in which they can reproduce the desired spatial continuity and the desired correlation. The objective of this work is to assess the performance and check the accuracy of independent simulation and co-simulation for modeling two co-regionalized attributes (Fe & MgO) meanwhile they are cross-correlated significantly, in a nickel-laterite actual case study located in Colombia.

2. Material and Methods

2.1. Methodology

Several Gaussian simulation algorithms have been developed. Generally speaking, they are divided into two types; exact and approximate algorithms [16]. The applied simulation and co-simulation algorithm in this paper is turning band proposed by Emery (2008) following Matheron (1973) and Mantoglou (1987). This method was first introduced by Chentsov (1957) in a special case of Brownian random functions, but has been extended for the Gaussian simulation of stationary and intrinsic random functions by Emery and Lantuejoul (2006) for independent simulation and by Emery (2008) for co-simulation. These methods aim at simplifying the Gaussian simulation problem in multidimensional spaces, using simulations in one dimension and spreading them to 2-D or 3-D spaces. This method is extremely fast with parallelizable computations and one can simulate as many locations as desired. The Gaussian simulation also exactly reproduces the desired covariance model [21, 22, 23, 24 and 25]. In co-simulation, the relationship is being characterized by examining the cross-variogram together with direct-variogram. There exist a range of various methodologies for modeling such a variogram [26, 27, 28, 29, 30, 31 and 32]. In this research, the linear model of coregionalization has been proposed of the following form [14]:

\[ C(h) = \sum_{n=1}^{N} B_n \rho_n(h) \]  

where \( \{\rho_n, n = 1 \ldots N\} \) is a set of positive semi-definite covariance functions and \( \{B_n, n = 1 \ldots N\} \) is a set of symmetric positive semi-definite matrices.
2.2. Presentation of dataset

The Cerro Matoso S.A. Nickel laterite deposit is an important resource of Ni in the world located in northwest Colombia [32 and 33]. Cauca ophiolite complex belonging to Cretaceous age shows some peridotite outcrops in the region which is the main house of the Cerro Matoso deposit [34]. The principal tectonic feature of this deposit is Romeral fault system with approximately 500 km that hosted this ophiolitic complex in the time of Pre-Andean orogeny. For instance, the regional boundary among accreted pre-Tertiary ophiolitic sequences and polymetamorphic core of the Central Cordillera can be detected through this structural discontinuity. Furthermore, the Geophysical surveys confirm that the Romeral fault system separate continental crust to the east from oceanic crust to the west. This deposit manifests itself through a hill range about 2.5 km length and 1.5 width and is evolved during a variably serpentinized ultramafic body [35]. The mineralogical and chemical alteration system is Lateritization and Saprolitization according to the geographical extent of the deposit. The dataset is composed of 3000 records of blasting holes belongs to this deposit. In which seven variables are assayed for each sample. In this research, two co-regionalized variables (Fe and MgO) out of these seven variables have been selected due to the presence of the satisfactory correlation coefficient. The sampling zone covers approximately 180×200×70 meters. Table 1 summarizes statistical parameters of these variables through collecting the corresponding samples in entire deposit. The frequency of each variable has been also depicted in Figure 1 to assess the shape and spread of the sample data. This bar chart is necessary before or during any analysis. As can be seen from the figure, Fe shows a distribution with two peaks, pretending some bimodality. The MgO seems to bear lognormal distribution with one peak which demonstrates that the low values are further fluctuating in the region.

Correlation coefficient as another important yardstick in multivariate analysis turns out the linear relationship (proportionality relationship) enclosed by Fe and MgO through a value between -1 and 1. This coefficient for Fe and MgO is -84.04%, which means that there is a considerable relationship between these two variables. Scatter plot (Figure 2) is an intuitively determination of the dependence relationship between two variables that can also be used to detect the possible anomalous data.

<table>
<thead>
<tr>
<th>Statistical Parameter</th>
<th>Fe</th>
<th>MgO</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>27.95%</td>
<td>8.95%</td>
</tr>
<tr>
<td>Median</td>
<td>26.1</td>
<td>3.6</td>
</tr>
<tr>
<td>Std. Deviation</td>
<td>15.74</td>
<td>10.23</td>
</tr>
<tr>
<td>Variance</td>
<td>247.839</td>
<td>104.699</td>
</tr>
<tr>
<td>Minimum</td>
<td>5.2</td>
<td>0.1</td>
</tr>
<tr>
<td>Maximum</td>
<td>57.2</td>
<td>39.43</td>
</tr>
</tbody>
</table>

2.3. Geostatistical Modeling

The initial analysis parts of the modeling are, first, implementing independent simulation for each variable and, second, co-simulating both variables simultaneously. For this purpose, the primary data should be declustered and transformed to the normal score Gaussian distribution [1]. The technique allows us assign each datum a weight based on closeness to surrounding data for alleviating the high pseudo frequency occurring in high graded areas [36]. These declustered data are then should be transformed to the
standard normal distribution. So the analysis data (Fe & Mgo) have been transformed to normal score N(0,1) by nscore subroutine of Gslib [11]. Each variogram-based geostatistical modeling including these two simulation and co-simulation approaches require learning the model of spatial continuity [1]. In simulation, direct-variogram plays an important role and in co-simulation; the cross-variogram is needed to be modeled as well. Hence, the initial step is to analyze the direct experimental variogram for deriving the potential isotropy or anisotropy of the variables of interest in the region. One technique is to calculate the direct experimental variogram in alternative directions with narrow tolerances. The differences in range values through the variograms give the idea of geometric anisotropy. So, for this purpose, the experimental variograms are calculated along the specific directions. The results showed that there is isotropy variability in the horizon and anisotropy in the vertical direction. As mentioned earlier, spatial continuity structure for employing the co-simulation can be represented by cross-variogram associated with the information obtained from the direct-variograms for co-regionalized variables. The cross-variogram was first introduced by Matheron (1965) as the natural generalization of the variogram [1]. This coregionalization matrix can be modeled by several methods. The most common and widespread approach is known as “Linear Coregionalization Model (LCM)” applicable to any multivariate spatial data analysis [37, 14]. In this model, the sample variograms should be fitted by semidefinite coregionalization matrices which indeed are mathematically consistent [14]. The difficulty of obtaining a semidefinite model has been covered somewhat by automatic or semi-automatic fitting method which are often used for modeling this type of spatial correlation structure [38]. This protocol makes the process of fitting somehow convenient. In this study, direct and cross-variogram model for Fe and MgO are calculated and depicted in Fig. 3. The theoretical model of LCM is also fitted to the experimental ones:

\[
\begin{pmatrix}
\gamma_{Fe}(h) & \gamma_{Fe-Mgo}(h) \\
\gamma_{Fe-Mgo}(h) & \gamma_{Mgo}(h)
\end{pmatrix} = \\
\begin{pmatrix}
0.05 & 0.05 \\
0.05 & 0.05
\end{pmatrix} N_{uggets} + \\
\begin{pmatrix}
0.45 & -0.25 \\
-0.25 & 0.25
\end{pmatrix} Sph(80m, 27m) + \\
\begin{pmatrix}
0.65 & -0.65 \\
-0.65 & 0.75
\end{pmatrix} Sph(\infty, 27m)
\]

Figure 3. Direct and cross-variogram analysis (top: Fe; middle: MgO; bottom: cross-variogram

In this part, our objective is to model the Fe and MgO via two commented methodologies. The first one considers the independent simulation for each variable and the second one acknowledges the correlation coefficient between two dependent variables by co-simulation. So, for independent simulation, one just needs to apply the direct-variograms of Fe and MgO, while the co-simulation is dealing with the cross-variogram as well as the direct ones. The applied simulation and co-simulation methodologies as explained above; are turning band simulation and co-simulation in which they have priority to other approximate approaches of simulation [39]. In Figure 4 and 5, one realization of simulation and co-simulation for each, have been provided. Visually consideration, one is not able to find out bolded diversity among the simulation and co-simulation results. But
in the upcoming sections, we will discuss about the statistical parameters of them.

3. Results and Discussion

In this part, for making a comparison between two approaches statistically, Table 2 is presented to show the correlation coefficient as an important key factor calculated over 100 realizations for each variable. As can be acquired from this table, the obtained correlation coefficient from co-simulation in average; is closer to the correlation coefficient of primary data.

<table>
<thead>
<tr>
<th>Variable</th>
<th>Primary data</th>
<th>Simulation</th>
<th>Co-Simulation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fe vs. MgO</td>
<td>-0.84</td>
<td>-0.53</td>
<td>-0.76</td>
</tr>
</tbody>
</table>

Based on the definition of relative error which is an absolute measure of difference between the true value and its approximation divided by the magnitude of that true value, one should consider these true and approximation values. For this purpose, the dataset of this case study are divided into two parts, one analysis dataset (30% of all the primary data) and one test data set (70% of all the primary data). In Geostatistical literatures, this methodology is known as jackknife [11]. In a nutshell, the test dataset are quantified by these two methodologies. As the true value of these test dataset are known, the approximation values can be obtained from this quantification. As can be seen from the Table 3, the absolute relative error for simulation is higher than the co-simulation.

<table>
<thead>
<tr>
<th>Variable</th>
<th>Simulation</th>
<th>Co-Simulation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fe</td>
<td>0.278</td>
<td>0.175</td>
</tr>
<tr>
<td>MgO</td>
<td>0.756</td>
<td>0.659</td>
</tr>
</tbody>
</table>

4. Conclusions

Multivariate Geostatistics offers a flexible framework for modeling the continuous variables when the variables convey a satisfactory correlation. The current methodology such as independent simulation in this case suffers from reproducing the expected correlation among the co-regionalized variables. Co-simulation as an alternative flexible technique, is dealing with quantifying linear coregionalization model which hold much acceptable results statistically. This approach overcomes the limitation of independent simulation and increases the model versatility. The priority of this method is attractive because of its capability to accept any number of nested structures. However, this method is concerned with quantifying linear coregionalization model, in which the process of modeling is somewhat time consuming and not very easy to use.

References


Abstract — The Narbaghi Copper deposit is located in the middle part of Urumieh-Dokhtar magmatic arc. That is situated 22 km northeast of Saveh City. The dominant lithology of this region consists of: silty tuff with interbedded tuff breccia and andesite flow ascribed to Eocene. Hydrothermal alteration zones are commonly associated with mineralization zones, recognizing these alteration areas for regional mineral exploration is much valuable. Remote sensing techniques were performed in the Narbaghi area to detect hydrothermal alteration zones. The study of the alterations of the Narbaghi mining area carried out using ETM+ and ASTER multispectral imagery data and different methods of image processing. The hydrothermal alterations were detected using the two methods false colour composite and band ratio on the both ETM+ and ASTER images. Then comparison of the results of each data set that were used, was performed. In order to extract the fractures and tectonic lineaments of the area and assess their relationship with the altered zones, the Sunangle filter on digital elevation models (DEM) and directional filter on the band 8 of ETM+ was applied. The comparison of the results deriving from the two methods indicates that the outcomes of the application of both methods with different Data (ASTER and ETM+ images) are in compliance and remote sensing methods are accurate. Finally, the comparison between the distribution of hydrothermal alteration zones and the map of the fractures shows that the alteration areas are related to fractures and they also overlap, particularly in the Narbaghi mine site.

Keywords: remote sensing, alteration, Narbaghi, Saveh, Iran.

1. Introduction

In 1972, the first satellite recognizing the earth resources was launched into space and after that more satellites were located in orbit by other countries and organizations that numerous satellite images of the earth's surface were collected. Since then satellite data has been widely used and remote sensing techniques became essential tools in map covering, environmental monitoring and ecological processes [6]. Today the importance of remote sensing in geological studies is increasing. Techniques used in this science to identify faults and geological structure, geomorphological phenomena, detect all types of material on the ground, identify different alteration and separation of lithological units, making it a valuable tool for researchers, geological and natural resources experts [15]. Extensive use of remote sensing in mapping lithological and geological structure and mineral exploration has widely reduced costs in desert exploration [16].

Remote sensing methods can be used to detect altered areas and mineral deposit associated easily. In addition to reduce cost, using remote sensing in mapping areas with limited access such as mountains and forests, it is a highly helpful solution [2]. One of the most usable of remote sensing resources for mining exploration and earth resources is ASTER sensor. In December 1999, this sensor abroad the Tera satellite was launched into space by NASA. ASTER is able to produce spectral images ranging from earth resources in fourteen different bands including three visible bands near the infrared (SWIR) with the spectral resolution of 30 meters, and five band in the range of the thermal infrared wavelength (TIR) [7]. The ASTER sensor has two telescopes which capture images on VNIR band. Both telescopes are angled forward the Zenith and a front facing telescope (Relative to the direction of circuit of satellite) and the rear facing telescope. The combining images of telescopes are used to create stereo images and digital elevation model (DEM) [18]. ASTER DEM sensor has a spatial resolution of 30 meres.

Another common source for the exploration of earth resources is Landsat 7 ETM+ instrument that was launched into space in 1999. This sensor is capable of providing 8-band spectral images that three bands of them are in the range of
visible light wavelength with the spatial resolution of 30 m (bands of 1 to 3), three bands are within the range of infrared with the spatial resolution of 30 m (band 4, 5 and 7), one band in the range of infrared heat with the spatial resolution of 60 m (band 6) and the panchromatic band with the spatial resolution of 15 (band 8) [17]. In this study the remote sensing techniques has been widely used to identify hydrothermal alteration in mining areas, Iran, North Saveh, Markazi Province. To evaluate these techniques, the images of two different sensors used and results of the same process on different data (ETM+ and ASTER) were compared. The images used in this study are related to the ASTER sensor and ETM+. In addition, the geological structures especially faults and fractures have been mapped by using remote sensing techniques. By comparison spatial distribution of faults and fractures to the hydrothermal alteration areas in a GIS, their relationships were examined.

2. Geological setting

The study area is part of the Zavieh 1:000,000 geological map [12]. The Narbaghi Copper index is located 22 Km north east of Saveh in the Markazi province. This area is in the central part of the magmatic belt of Oroumieh-Dokhtar belt (fig. 1). Iran copper belt with abundant intrusive and volcanic rocks is in the Oroumieh-Dokhtar belt. This belt also hosts many porphyry copper deposits and mines. The formation of this belt is related to the subduction of Neo-Tethys beneath Iranian plate. Based on the main characteristic of this belt, the majority parts of this area is covered by the sequence of volcanic and sub-volcanic igneous rocks and sedimentary rocks [4,9].

The complex of Eocene volcanic-sedimentary rocks includes dacite tuffs, trachyte and andesite volcanic rocks with pyroclastic interbeds that are the oldest known unit in Narbaghi area. Plutons introduced into Eocene volcanic-sedimentary rocks and have deformed them. It is probably intrusive bodies that caused copper mineralization and alteration in the region of Narbaghi [13]. Existing limestone is massive and with no distinctive layers. Recent alluviums include unconsolidated and poorly cemented deposits and composed from silt and clay particles and rubbles. Alluvium can be seen within the hornblende- andesite unit and around the limestone unit as well as (fig. 2).

3. Material and Methods

The images of ETM+ taken in September 17st, 2002 are used in this case study. The correction of images up to 1T-level has been already done. 1T-level corrections include the radiometric correction, geometric correction and increasing accuracy, using ground control points and DEM. ASTER images used in this study are form 6st, October 2006 and required radiometric and geometric corrections have been done on them in the EROS centre of American Geological Survey (USGS). In this study, to separate the alteration, the method of colour composite and the band ratio were used. The best possible colour composite was achieved by calculating OIF. Different band ratio images are created according variety of minerals associated to each hydrothermal alteration. Two methods were used for extracting and mapping the faults and fractures. First, DEM of the area is used to provide hillshade model and assess topographic features to the enhancement of linear features such as fractures and faults. The faults and fractures were extracted by applying Sun Angle Filter on DEM of the area. GIS was used to plot the map of lineaments and alterations and to examine the relationship between their disorders.
4. Discussion

4.1. False colour composite

False colour composite of the band multispectral images is provided from common remote sensing techniques. This method is based on the theory of colours. According to this theory, all the colours that human eyes can see are made from different composites of the three primary colours: red, green and blue. In this technique, three different bands are used in the role of three primary colours: red, green and blue [17]. The concept of optimum index factor is used to select the appropriate bands which is developed by Chavez et al. 1982 and Sheffield in 1985. The basis of this method is selecting the band which has the highest spectral information on their own [8]. For this purpose, the standard deviation of the spectral data in each band is measured by the intended satellite images. Similarly, the spectral correlation between the spectral band data is measured. The three bands that have the most sum of standard deviation and have the least sum of correlation coefficient provide the best colour composite [20].

$$OIF = \frac{\sum_{j=1}^{k} s_k}{\sum_{j=1}^{r} |r_j|}$$ \hfill [21]

In the above equation, $s_k$ is the standard deviation of band $k$, and $r_j$ is the correlation coefficient for each pair of bands [14]. High levels of OIF represent bands with highest data and the minimum of duplicate data. To create a colour composite, ASTER and ETM+ images of the area were analysed. For ETM+ images the thermal band (band 6) and the panchromatic band (band 8) is avoided. For other bands, standard deviation of spectral date and the correlation between each pair of bands were measured (table 1 and 2).

From 6 usable ETM+ bands, 3 bands of them have been used to make the composite colour. There are twenty possible ways to create different composites of these bands. By using the data in table 1 and 2, the amount of OIF for 20 states was calculated. Table 3 shows that most OIF is for band 7, 5 and 3 with the amount of 11.1. So these three bands were used to create false colour composite images. Figure 3 shows the false colour composite image of band 7 as red, band 5 as green and band 3 as blue. In this image, altered areas appeared with bright white colour.

### Table 1– shows the statistical data in different spectral bands of ETM+ images

<table>
<thead>
<tr>
<th>Basic Stats</th>
<th>Min</th>
<th>Max</th>
<th>Mean</th>
<th>Standard deviation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Band 1</td>
<td>46</td>
<td>221</td>
<td>64.50642</td>
<td>4.605048</td>
</tr>
<tr>
<td>Band 2</td>
<td>34</td>
<td>214</td>
<td>60.27435</td>
<td>6.369071</td>
</tr>
<tr>
<td>Band 3</td>
<td>34</td>
<td>254</td>
<td>74.22342</td>
<td>9.553062</td>
</tr>
<tr>
<td>Band 4</td>
<td>24</td>
<td>203</td>
<td>57.99232</td>
<td>8.219825</td>
</tr>
<tr>
<td>Band 5</td>
<td>17</td>
<td>255</td>
<td>61.2803</td>
<td>11.43545</td>
</tr>
<tr>
<td>Band 7</td>
<td>13</td>
<td>255</td>
<td>54.42949</td>
<td>9.992137</td>
</tr>
</tbody>
</table>

### Table 2 – shows the correlation coefficient between the spectral bands of ETM+ images

<table>
<thead>
<tr>
<th>Correlation</th>
<th>Band 1</th>
<th>Band 2</th>
<th>Band 3</th>
<th>Band 4</th>
<th>Band 5</th>
<th>Band 7</th>
</tr>
</thead>
<tbody>
<tr>
<td>Band 1</td>
<td>1</td>
<td>0.945</td>
<td>0.886</td>
<td>0.771</td>
<td>0.76</td>
<td>0.78</td>
</tr>
<tr>
<td></td>
<td>1</td>
<td>024</td>
<td>739</td>
<td>01</td>
<td>7111</td>
<td>943</td>
</tr>
<tr>
<td>Band 2</td>
<td>0.945</td>
<td>1</td>
<td>0.966</td>
<td>0.882</td>
<td>0.88</td>
<td>0.88</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>024</td>
<td>735</td>
<td>524</td>
<td>0233</td>
<td>020</td>
</tr>
<tr>
<td>Band 3</td>
<td>0.886</td>
<td>0.966</td>
<td>1</td>
<td>0.917</td>
<td>0.90</td>
<td>0.90</td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>739</td>
<td>735</td>
<td>247</td>
<td>8125</td>
<td>595</td>
</tr>
<tr>
<td>Band 4</td>
<td>0.771</td>
<td>0.882</td>
<td>0.917</td>
<td>1</td>
<td>0.90</td>
<td>0.85</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>01</td>
<td>524</td>
<td>247</td>
<td>6405</td>
<td>693</td>
</tr>
<tr>
<td>Band 5</td>
<td>0.767</td>
<td>0.880</td>
<td>0.908</td>
<td>0.906</td>
<td>1</td>
<td>0.97</td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>111</td>
<td>233</td>
<td>125</td>
<td>405</td>
<td>484</td>
</tr>
<tr>
<td>Band 7</td>
<td>0.789</td>
<td>0.880</td>
<td>0.905</td>
<td>0.856</td>
<td>0.97</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>7</td>
<td>437</td>
<td>208</td>
<td>958</td>
<td>937</td>
<td>4847</td>
</tr>
</tbody>
</table>

### Table 3– calculating OIF index for triple composite of ASTER images bands.

<table>
<thead>
<tr>
<th>RGB</th>
<th>OIF</th>
<th>RGB</th>
<th>OIF</th>
</tr>
</thead>
<tbody>
<tr>
<td>123</td>
<td>7.335071</td>
<td>234</td>
<td>8.726516</td>
</tr>
<tr>
<td>124</td>
<td>7.386383</td>
<td>235</td>
<td>9.929822</td>
</tr>
<tr>
<td>125</td>
<td>8.644439</td>
<td>237</td>
<td>9.413441</td>
</tr>
<tr>
<td>127</td>
<td>8.018704</td>
<td>245</td>
<td>9.750006</td>
</tr>
<tr>
<td>134</td>
<td>8.690474</td>
<td>247</td>
<td>9.383259</td>
</tr>
<tr>
<td>135</td>
<td>9.989777</td>
<td>257</td>
<td>10.16224</td>
</tr>
<tr>
<td>137</td>
<td>9.352825</td>
<td>345</td>
<td>10.69206</td>
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<tr>
<td>145</td>
<td>9.242346</td>
<td>347</td>
<td>10.35953</td>
</tr>
<tr>
<td>147</td>
<td>9.43872</td>
<td>357</td>
<td>11.10844</td>
</tr>
<tr>
<td>157</td>
<td>10.28391</td>
<td>457</td>
<td>10.82738</td>
</tr>
</tbody>
</table>
False colour composite images were made with ASTER band 4 to 9, and thermal bands (bands 10~14) and NVIR bans (bands 1~3) were discarded. Table 4 shows the statistical data of these 6 bands including standard deviations and table 5 shows the correlation of these bands mutually. Table 6 shows the results of the OIF calculation and different modes of ASTER band composite. According to table 6, the composite of bands 4, 5 and 6 shows the most OIF and the best possible composite to display a variety of lithology. In this image, the Phyllic alteration zone can be seen in slim green.

**Table 4** shows the statistics from different spectral bands of ETM+ images.

<table>
<thead>
<tr>
<th>Basic Stats</th>
<th>Min</th>
<th>Max</th>
<th>Mean</th>
<th>Standard deviation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Band 4</td>
<td>0</td>
<td>55.219601</td>
<td>11.619035</td>
<td>5.935453</td>
</tr>
<tr>
<td>Band 5</td>
<td>0</td>
<td>17.6784</td>
<td>3.596279</td>
<td>1.806332</td>
</tr>
<tr>
<td>Band 6</td>
<td>0</td>
<td>15.875</td>
<td>3.39205</td>
<td>1.719672</td>
</tr>
<tr>
<td>Band 7</td>
<td>0</td>
<td>15.1638</td>
<td>2.997191</td>
<td>1.511143</td>
</tr>
<tr>
<td>Band 8</td>
<td>0</td>
<td>10.591801</td>
<td>2.156207</td>
<td>1.089937</td>
</tr>
<tr>
<td>Band 9</td>
<td>0</td>
<td>8.077201</td>
<td>1.601749</td>
<td>0.797437</td>
</tr>
</tbody>
</table>

Table 5 shows the correlation coefficient between the spectral bands of ASTER.

<table>
<thead>
<tr>
<th>Correlation</th>
<th>Band 4</th>
<th>Band 5</th>
<th>Band 6</th>
<th>Band 7</th>
<th>Band 8</th>
<th>Band 9</th>
</tr>
</thead>
<tbody>
<tr>
<td>Band 4</td>
<td>1</td>
<td>0.9816</td>
<td>0.9706</td>
<td>0.9537</td>
<td>0.9600</td>
<td>0.9763</td>
</tr>
<tr>
<td>Band 5</td>
<td>0.9816</td>
<td>1</td>
<td>0.9855</td>
<td>0.9511</td>
<td>0.9593</td>
<td>0.9737</td>
</tr>
</tbody>
</table>

Table 6- calculating the OIF index for the triple composite of ASTER bands.

<table>
<thead>
<tr>
<th>RGB</th>
<th>OIF</th>
<th>RGB</th>
<th>OIF</th>
</tr>
</thead>
<tbody>
<tr>
<td>456</td>
<td>3.220833</td>
<td>567</td>
<td>1.751764</td>
</tr>
<tr>
<td>457</td>
<td>3.205826</td>
<td>568</td>
<td>1.596123</td>
</tr>
<tr>
<td>458</td>
<td>3.044524</td>
<td>569</td>
<td>1.480573</td>
</tr>
<tr>
<td>459</td>
<td>2.912903</td>
<td>578</td>
<td>1.520956</td>
</tr>
<tr>
<td>467</td>
<td>3.201395</td>
<td>579</td>
<td>1.420889</td>
</tr>
<tr>
<td>468</td>
<td>3.03874</td>
<td>589</td>
<td>1.267514</td>
</tr>
<tr>
<td>469</td>
<td>2.906814</td>
<td>678</td>
<td>1.503557</td>
</tr>
<tr>
<td>478</td>
<td>2.942426</td>
<td>679</td>
<td>1.40297</td>
</tr>
<tr>
<td>479</td>
<td>2.841542</td>
<td>689</td>
<td>1.248358</td>
</tr>
<tr>
<td>489</td>
<td>2.681324</td>
<td>789</td>
<td>1.155975</td>
</tr>
</tbody>
</table>

**4.2. Band ratio**

Band ratio is one of most widely used technique for exploration of hydrothermal alterations in many parts of the world [2]. Band ratio causes to decrease the effects of shadows, effects of topography on the brightness and the reduction effect of the variation of intensity and angle of
sunlight due to the seasonal changes [19]. Each satellite image is made from elements or pixels and each pixel represents a certain area of the earth. These pixels show the value of the reflected light on the earth in the form of an 8-bit number from 0 to 255 [1]. The band ratio come from the dividing brightness value of corresponding pixels in one band to the other band and are used for the separation of minerals with a distinctive spectral characteristic [3]. Band selection will be done based on the spectral behaviour of targets. Clay mineral alteration zones in the range of 1.65 μm of the electromagnetic wavelength show the highest reflectance that is equivalent to band 5 of the ETM+ and also in the range of 2.2 μm of waves that is equivalent to the band 7 shows the highest absorption [10,1].

In the band ratio techniques, dividing the pixel brightness for band 5 and 7 gives the largest quotient. As the difference in the brightness of numerator and denominator of these pixels are the maximum value. As a result, the image output of the band 5 and 7 of the ETM+, areas with clay minerals create the brightest pixel. The 7/5 band ratio images of ETM+ are used for the detection of alteration zones containing clay minerals such as Alunite, Kaolinite, Illite and Montmorillonite [1]. Iron oxide and sulphates are from other minerals related to the hydrothermal alteration zones. These materials in band 1 of the ETM+ show the highest reflection and the highest absorption in band 3 and in 1/3 band ratio images, has a bright tone [1]. With ratio images created by this technique, we can produce false colour composite. To achieve this purpose, the band ratio of 7/5 is used as red, band ratio 1/3 is used as green and band ratio 5/3 is used as blue colour. Figure 5 shows the results of the bands composites. In this image, the altered areas are shown from yellow to orange colour.

ASTER band 5 and 6 detect absorb electromagnetic radiation of AL-OH that caused by clay minerals. In the range of band 7, Fe-OH cause to absorb radiation. Band ratios 5/4 and 6/4 are used for identifying minerals that are containing Al+OH component, and ratio 7/5 is to identifying minerals that has Fe-OH [5]. These three ratios are used to create false colour composites and ratios 5/4, 6/4 and 7/4 respectively, are used in the colours red, green and blue. In figure 6, the result of the hydrothermal alteration is shown with bright tone and bright white.

Figure 6: shows false colour images of the band ratios 5/4, 4/6 and 7/4 of the ASTER sensors.

4.3- Extracting faults and fractures
In order to assess and study the relationship between fractures and faults with the alternation zones on the digital elevation model (DEM), Sanangle filters are applied. In this method, light is stimulated on the area from certain azimuth and certain height and the resulting shadows and brighten images. In this study, the light was simulated on the area at four cardinal directions and four intercardinal directions with an elevation angle of 45. The obtained images were used to identify linear topographic features available in the area. Alternatively, by applying the directional filters on band 8 ETM+, the linear features in the images were detected. Finally, by extracting the lineaments and faults of the resulting images and incorporation of them, a map for the fractures was made (fig. 7). Using the GIS, the resulting map of the identification of the altered areas and the map of faults and fractures was overlaid on each other (fig. 8). Comparing these maps show that alteration zones and fractures have overlapping in some
places specially in the mine site Narbaghi and location of the intrusive body.

Figure 7- shows the resulting images of applying filters on ETM+ sensors band 8 and extracted fractures.

Figure 8- comparing altered zones with fractures and faults

5- Conclusions
The study of the Narbaghi mineral alteration area has been carried out via remote sensing techniques and by using images of ETM+ and ASTER data. In this study, the detection and separation of different alteration types including Phyllic, Argillic and Propylitic were performed by providing false colour composite images and band ratio. The distribution of these alterations in the area shows a zoning pattern where the altered Phyllic zone is located in the centre and is surrounded by propylitic zone. The argillic zone is diffused and it is mostly along the faults and fractures. Spatial distribution of altered zones was similar in both ETM+ and ASTER processed images. Comparing the results of two methods with different date (ASTER and ETM+ images) indicates compliance with both the results and accuracy of remote sensing methods. The comparison the distribution of hydrothermal alteration with fractures map shows the spatial relationship between the altered areas and fractured areas and these areas are overlapping each other.

References


Extraction of zinc from dump dust of ferrous metallurgy in the way their joint processing with the oxidized zinc ore

M.A. Naimanbayev, N.G. Lokhova, Ye.K. Markayev, Zh.A. Baltabekova

«Institute of Metallurgy and Ore Benefication» JSC, Almaty, Kazakhstan

Abstract - One of the resource in the production of non-ferrous metals is the use of waste iron in steel industry, in which the content of non-ferrous metals is up to industrial conditions. Thus, in dusts of gas purification of some plants of ferrous metallurgy the zinc content is as high as 15%. The results of the study of the binder agent effect during the briquetting of charge, the type of the carbonaceous reducing agent, the consumption of reducing agent, fineness of charge components on the process of carbothermal reduction of zinc from oxidized zinc ore with the addition of stale dust of gas cleaning of blast furnace smelting has been addressed. Bentonite, hydrated lime and treacle were tested as binding agent when briquetting of charge. It is established that optimum binding agent is treacle in an amount of 4.5-5.0% by weight of the mass of the ore. It is shown that the residual zinc content in a product of the reductive roasting with the special coke received from coal of Shubarkol deposit is 1.9 times less, than with anthracite and 3.3 times less, than with metallurgical coke, i.e. special coke is the most fissile reducing agent. The carbon consumption during carbothermal reduction of zinc from oxide ore with the addition of dust is 22-24% lower than in case of zinc recovery from ore. It was found that crushing of charge to class + 0.071-0.04 microns reduces the degree of zinc sublimation. If the size of charge is 1.0 micron, then the residual zinc content is increased in the cinder. High recovery efficiency is achieved with the following composition of charge, wt. %, oxidized zinc ore is 53.8; dust of gas purification of blast furnace smelting is 26.9; special coke is 21.0; treacle is 5.3.

Keywords: zinc, charge, oxidized zinc ore, dust of gas purification of blast furnace smelting, binding agent, carbothermic reduction

1. Introduction

The annual volume of world production of zinc exceeds 10 million tones. Half of this volume is used to protect steel against rust. Environmentally attractive fact in favor of use of zinc is that 80% of it is used again, and it does not lose its physical and chemical properties. Protecting steel against corrosion, zinc helps to preserve natural resources, such as iron ore and energy. Extending the service life of steel, zinc increases the life cycle of goods and capital investments, i.e. buildings, bridges, power and water distribution, telecommunications, thus protecting investment and helping to reduce repair and maintenance costs [1].

It is known [2], that 85% of the total world production of zinc is obtained from concentrates, and the rest from sub-standard and secondary raw materials and wastes of chemical and metallurgical industries.

Increased release of zinc through the processing of substandard raw materials is an important economic and environmental objective.

The main zinc-containing waste from metallurgical industries are blast and steel smelting dusts. Analysis of scientific, technical and patent literature has shown the possibility of processing zinc-containing waste from iron and steel production. Difference of pyrometallurgical methods of extraction of zinc is in the parameters of technological mode, types of fuel and reducing agent used and the constructive design of systems. [3]

Industrial pyrometallurgical processing of dust and sludge of gas purification systems of blast furnaces and converter shops have been mastered by Germany company named AG Krupp, recycling dust and sludge of steel production in rotary tube furnaces [4].

Information on the joint recycling of oxidized zinc ore with zinc-containing waste from ferrous metallurgy is not available in the literature. At the same time, during the recycling of oxidized zinc ore with a low iron content it is necessary to introduce an iron product in the charge, promoting zinc stripping. Dust of ferrous metallurgy retaining high profitability of Waelz process, may be used as the flux instead of the ballast iron concentrate. In addition, the high zinc content in the blast furnace sludge will significantly extend the life of the deposit of oxidized zinc ore. Such type of deposit is Shaimerden deposit.

Application of charge briquetting in the reduction roast is one of the cost-effective processing methods [5-8]. High reactivity of charge is necessary for the process without slag with the recovery rate of oxides exceeding the speed of their melting. Use of active carbonaceous reducing agent, fine grinding of materials their careful hashing will meet this requirement. As a result of the production practice it is confirmed that joint briquetting of the crushed raw materials and a reducing agent is the best way of preparation of charge.
Addition of binders into the charge is intended to increase the strength of the briquettes. [10]

The objective of this work is to study the effect of the binder during the charge briquetting, the amount of the carbonaceous reducing agent, the type of reducing agent, the particle size of charge on the indices of high temperature reductive roasting of briquetted charge consisting of oxidized zinc ore, dust of gas purification of blast furnace smelting and solid reducing agent.

2. Material and Methods

Methods of analysis. X-ray fluorescence analysis was performed with the help of a wave dispersion spectrometer Venus 200 PANalytical B.V. (PANalytical B.V., Holland).

The chemical analysis of samples was performed with the help of an optical emission spectrometer with inductively coupled plasma Optima 2000 DV (Perkin Elmer, USA).

Materials and Instruments. Materials for the study were the state dust of gas purification of blast furnace smelting from the sludge of the storage area in the West Siberian Metallurgical Plant (Novokuznetsk, Russian Federation), containing, wt. %: Zn 12.3; Fe 40.1; Ca 3.4; Si 4.3; Mg 0.7; Mn 0.2; Al 1.6; Ti 0.3; Pb 0.8; S 1.0; ore of the Shaimerdyn deposit provided by “Kazzinc” Ridder Metallurgical Complex, containing weight %: Zn 21.4; Fe 2.6; Si 11.4; Al 5.0; Ca 6.7; Mn 0.8; Ti 0.2; Pb 0.5; Mg 0.3; F 0.2; Cu 0.01; As 0.02.

Studies on the effect of reducing agent on the high-temperature reduction process is conducted using carbonaceous materials available and used in the industry (Table 1).

Shubarkol deposit refers to low-ash gas coals. The special coke received by thermal-oxidative coking of Shubarkol deposit coal represents a solid carbonaceous reducing agent with grain size of 5-25 mm, possessing the developed steam structure formed as large pores, 150-300 microns in size, and less – 0.5-1.0 microns. [11].

Bentonite, hydrated lime and treacle were tested as a binding agent during the briquetting of the charge.

Table 1 - Properties of Carbonaceous Reducing Agents

<table>
<thead>
<tr>
<th>Indices</th>
<th>Anthracite coal</th>
<th>Metallurgical coke</th>
<th>Special coke</th>
</tr>
</thead>
<tbody>
<tr>
<td>A (% ash content)</td>
<td>3.73</td>
<td>16.30</td>
<td>2.41</td>
</tr>
<tr>
<td>W (%) (humidity)</td>
<td>2.31</td>
<td>1.05</td>
<td>3.19</td>
</tr>
<tr>
<td>V (volatile matter)</td>
<td>5.10</td>
<td>2.67</td>
<td>24.91</td>
</tr>
<tr>
<td>Carbon content, wt. %</td>
<td>90.2</td>
<td>79.7</td>
<td>69.2</td>
</tr>
<tr>
<td>Sulfur content, wt. %</td>
<td>0.18</td>
<td>0.96</td>
<td>0.10</td>
</tr>
</tbody>
</table>

Great demands along with giving high mechanical strength to briquettes are placed on binders, they should not bring harmful or ballast impurity [12], and the value of binders should be comparable with the value of raw material agglomerates, otherwise the value of a binder will make briquetting as a noncompetitive method. Hydrated lime is one of binders having high bind and flux possibility as well as cheap and abundant. The main disadvantage of hydrated lime is that at temperature is above 570°C, it decomposes and loses its strength.

The bentonite is the most widely used binder in the ferrous metallurgy, because it has a great capacity to absorb water and consists of layers of aluminum silicates.

The feed treacle is an advanced binder. This type of treacle is a waste from sugar-beet production, syrupy liquid of dark brown color with a water content of 22.7% and carbohydrates of 58-60%, mainly sugar, easily soluble in any proportion of cold and hot water and with a low value. Treacle is non-toxic, i.e. an environmentally friendly industrial product.

The high temperature roasting was performed at a temperature of 1250°C in a horizontal furnace “Nabertherm”-1300 (Germany) with automatic temperature control. The accuracy of temperature measurement was 5°C. Experimental Method. Due to the fact that the consumption of the solid reducing agent which provides maximum extraction of zinc fumes in each case varies depending on the kind of raw material, we selected the consumption of a solid reducing agent as obviously providing fullness of zinc sublimation.

Briquetting was performed on a laboratory press. The charge for pressing was mixed in a mixing cup with a binder, then the sample was maintained under the charge compression within 1-2 min and after that briquettes were extracted from a forming block by means of extrusion. Briquettes have a cubic shape with an edge size of 19 mm.

The briquettes volume of 6.86 cm³ was selected based on the results of thermophysical calculations of heating and reducing in a horizontal tube furnace. Drying of briquettes was performed at 130°C.

The quality of finished briquettes was evaluated for mechanical strength with the help of stroke method, with repeated dropping on a concrete slab from a height of 1 m to the briquette split.

The briquetted charge was placed in an alundum crucible which was moved into a working zone of a furnace. The furnace heating was stopped after reaching a temperature of 1250°C. The crucibles were covered by an alundum cover. The formed fumes were collected on the inner surface of the cover. Determination of mass of a crucible containing calcined and fumes, was performed after cooling the furnace to room temperature.

3. Results and Discussion

During experiments on study of the effect of a carbonaceous reducing agent type on high-temperature zinc reduction, calculation of the amount of solid reducing agent necessary for reduction of oxides under the equal
conditions, was performed with regard to the carbon content in the material. The ratio in the charge before briquetting was:

ore : dust : special coke : treacle = 1 : 0.5 : 0.39 : 0.1;
ore : dust : anthracite : treacle = 1 : 0.5 : 0.35 : 0.1;
ore : dust : metallurgical coke : treacle = 1 : 0.5 : 0.4 : 0.1.

Effect of the binder on the indices of high temperature reductive roasting. Correctly chosen method of preparation of the charge is an initial prerequisite for the achieving of high indices of zinc extraction. One of the components constituting the charge is a binder used during briquetting.

Results of the reductive roasting of charge briquettes with various binders are given in Table 2.

Table 2 - Results of the reductive roasting of charge briquettes with various binders

<table>
<thead>
<tr>
<th>Components</th>
<th>Quantity, wt. %</th>
<th>Number of strokes of the briquette</th>
<th>Calcine efficiency, %</th>
<th>Zink content in calcine, wt. %</th>
<th>Degree of metallization Fe, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore</td>
<td>50.5</td>
<td>6.0</td>
<td>52.2</td>
<td>0.50</td>
<td>19.7</td>
</tr>
<tr>
<td>Dust</td>
<td>25.3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Reducing agent*</td>
<td>19.5</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>The binder is treacle of 4.7 wt. %</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore</td>
<td>52.2</td>
<td>7.0</td>
<td>53.0</td>
<td>0.79</td>
<td>20.3</td>
</tr>
<tr>
<td>Dust</td>
<td>26.1</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Reducing agent*</td>
<td>20.3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>The binder is bentonite of 1.4 wt. %</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore</td>
<td>52.2</td>
<td>9.0</td>
<td>53.3</td>
<td>0.90</td>
<td>18.5</td>
</tr>
<tr>
<td>Dust</td>
<td>26.2</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Reducing agent*</td>
<td>20.3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>The binder is hydrated lime of 1.3 wt. %</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

*The content of carbon in the charge taking into account carbon in dust is 25.0%.

The data in Table 2 shows that the mechanical strength of the briquettes increases in the row treacle <bentonite <hydrated lime and the residual content of zinc in a cinder submits to this regularity. Such behavior of the studied binders is caused by the fact that binding agents in the form of solutions, in comparison with powder, are more evenly distributed in volume of the briquetted material and have high reactivity [13].

The highest degree of metallization of iron during high-temperature roasting of briquetted charge is achieved using bentonite as a binder - 20.3%, and the lowest when using the hydrated lime -18.5%.

Effect of the binder on the reduction of zinc and iron to metal during the roasting of the briquetted charge is determined by the briquette permeability, the denser briquette, the greater difficulty during the reduction reactions.

Results of roasting of briquettes of charge with treacle showed that the reduction reaction of zinc to gaseous state proceeds with greater speed, than iron metallization reaction in the received briquettes.

Effect of the type of the carbonaceous reducing agent on the residual zinc content in the product of roasting. Studies have shown that the reactivity of the subjects of the carbonaceous reducing agents have a significant effect on the zinc sublimation (Figure 1).

Shubarkol special coke was the most active reducing agent. The residual content of zinc in the calcine after reduction roasting of charge with this type of coke is 0.49 wt. %. Metallurgical coke has a much lower activity. The residual content of zinc in the calcine is 1.61 wt. %. The activity of anthracite has intermediate value. The residual content of zinc in the calcine is 0.93 wt. %.

The difference in the effect of the test carbonaceous reducing agents at the zinc sublimation due to the fact that according to [11], the reactivity of the special coke is 5.2 ml/g/s, and, for example, metallurgical coke is 0.62 ml/g/s only. Furthermore, the reactivity of special coke is 3.82 ml / m²/sec, and the reactivity of metallurgical coke is 0.05 ml / m²/s.

Effect of the amount of solid reducing agent in a charge on the of high temperature roasting process. The carbon content in the charge has a great effect on the process of high-temperature reduction of oxidized metals. The amount of reducing agent should be determined as the need to ensure the required concentration and getting loose, crumbling cinder.

Studies on the effect of the solid reducing agent as part of the charge was performed using special coke derived from the coal of Shubarkol deposit.

It is known [11] that, the share of costs for coking of coal in the structure of value is 85-9 %, and the share of the value of coke in the production value of the most non-blast furnace users as high as 40 - 50 % [1]. Therefore, it is necessary to attract non-deficient and cheap coal for the increasing of the efficiency of production that use the coke. For example, brown, long-flaming, gas, petrographically homogeneous, fortified coal of Shubarkol deposit. The coke cost received from such coals, is below the coke cost, received from hard coals.
Calculation of an expense of a solid reducing agent was performed in relation to amount of the oxidized zinc ore in charge. Results of experiments are given in Table 3.

Table 3 - Results of the reducing roasting of briquettes of charge with various quantity of a solid reducing agent

<table>
<thead>
<tr>
<th>Charge</th>
<th>Special coke amount, wt. %</th>
<th>Calcine efficiency, %</th>
<th>Zink content in calcine, wt. %</th>
<th>Degree of metallization Fe, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore</td>
<td>50.5</td>
<td>45.3</td>
<td>53.1</td>
<td>0.66</td>
</tr>
<tr>
<td>Dust</td>
<td>25.3</td>
<td>38.7</td>
<td>52.2</td>
<td>0.50</td>
</tr>
<tr>
<td>Treacle</td>
<td>4.70</td>
<td>32.0</td>
<td>53.7</td>
<td>0.79</td>
</tr>
<tr>
<td></td>
<td></td>
<td>25.3</td>
<td>55.3</td>
<td>0.92</td>
</tr>
</tbody>
</table>

The data in Table 3 shows that the residual zinc content in the calcine is reduced with increasing expense of a special coke from 25.3 to 38.7%. A further increase in the expense of solid reducing agent increases the degree of metallization and residual zinc content in the calcine.

It should be noted that in case of special coke expense of 25.3% the melting briquettes was observed, at an expense of 32% the degree of fusion is less, but calcine is dense and difficult to destroy. In case of special coke expense of 38.7% and above, calcines are loose, friable.

Effect of charge particle size on the residual content of zinc in the calcine. The effect of material size was studied in order to study the effect of variables of process parameters on the residual zinc content in the calcine. Because the size of the material largely affects on the efficiency of the recovery process. The finer the material and the lower the porosity, worse conditions for heat transfer and the probability of a material baking is higher.

In the study of the effect of particle size of charge on the high-temperature zinc reduction, all components of the charge were crushed to a uniform particle size. Content of the test charge, weight. %: oxidized zinc ore is 50.5; dust of gas purification of blast furnace smelting is 25.3; special coke is 19.5; treacle is 4.7.

Figure 2 shows the effect of charge particle size on the residual content of zinc in the calcine.

4. Conclusions

The treacle in an amount of 4.5-5.0 wt. % is the best binder during the briquetting of charge consisting of oxidized zinc ore, dust of gas purification of blast furnace smelting and solid reducing agent.

It is found that the optimal consumption of solid reducing agent is 38-39% of the oxidized zinc ore amount in the charge.

Studies of the effect of charge particle size have shown that the presence of dust reduces the gas permeability of charge, increases the probability of sintering of material, which has a negative impact on the performance of high temperature zinc reduction and necessitates briquetting.

Study of zinc reduction from various types of reducing agents showed prospects of special coke derived from brown coal of Shubarkol deposit for the high reduction of zinc oxide from the charge and dust of blast gas purification of furnace smelting gas.

References


Research on the landscape attractiveness of the selected abandoned quarries

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The publication presents the research results concerning determination of the level of landscape attractiveness of the abandoned quarries. The research concerned 20 structures located within the area of Poland, Great Britain and Austria. The research used procedures of landscape attractiveness assessment which includes three research methods: survey method with the use of the semantic differential, the method of point bonitation and the landscape entropy method. Throughout the conducted analyses, the level of landscape attractiveness of the quarries was determined through including them into one of four classification groups. The main motif showing quarries' attractiveness (group I) is their uniqueness, differentiation, as well as interest and curiosity evoked. What is more, these structures are characterised by a good state of preservation, not an advanced level of natural succession, good road accessibility, and occurrence of surface waters. They may be used for universal social education, e.g. making a didactic place. Quarries showing landscape attractiveness (group II) are usually characterised by slightly lower parameters compared to group I. They usually have a lesser vertical differentiation, and the pace of natural succession is a bit faster, and thus some precious and interesting geological profiles are covered. On the other hand, slightly attractive quarry landscapes (group III) have elements which decrease their attractiveness evaluation (e.g. little height differences, hindered access to the structure). Unattractive quarries (group IV) do not contrast surrounding areas, remain hardly visible and are not accessible along the set routes. What is more, they have very little vertical differences, preservation state is very bad while the pace of natural succession is meaningful. These structures are not perceived as interesting or useful.

Keywords: quarries, landscape, post-mining areas, rock mining, reclamation

1. Introduction

The necessity of acquisition of resources indispensable for production of materials used in many branches of industry has become the main factor determining the landscaping [6]. The landscape forms that occurred as the result of rock mining, which caused interference in the environment, create new values that could constitute an important element of tourist interest. Obviously, quarries, as quite specific landscape forms, might be perceived by individuals as more or less attractive. Supposing they may be perceived attractive in many respects, then their location in the vicinity of key tourist routes may occur to raise the region tourist attractiveness. They may be a sort of a side attraction, used successfully to liven up the local tourism. For among the newly created modern tourist forms, there is also geotourism and industrial tourism. People interested in such a kind of sightseeing, look for new stimuli, challenges as well as interests [5, 10]. New modern attractions aim to face all of these. Cultural objects like monuments and museums are not enough in this case. The new tourism refers to the new form of attractions evolution, which is the 3xE rule (entertainment, excitement, education) that is displacing the traditional 3xS formula (sun, sea, sand). Another important element here are tourist attractions built up on their cultural heritage. Heritage is such an element of legacy that may be evaluated to assess its value. The UN Convention concerning the Protection of the World Cultural and Natural Heritage (1972) recognizes the cultural heritage as monuments of architecture, sculpture, painting, objects, archaeological structures, cave dwellings and highly valuable elements. Using cultural heritage as an attraction is continually more frequently practiced. This is a background for geoparks and geotourism popularization, which are forms of cognitive tourism also called global tourism [9, 10, 12].

Geotourism is related to exploring and means recognising geological attractions and active participation in discovering interesting forms, rocks, minerals and landscape features [10]. Moreover, naturalists claim that morphologically differentiated landscape is the most precious for tourism and recreation [4]. Thus in the areas of abandoned quarries with unique forms and geological constructions discovered during material exploitation as well as located in natural protected areas, there is a combination of natural, geological and industrial values, as well as of industrial tourism (post-industrial). At the same time there is a lack of research
supporting such reasoning, as there was not proper investigation recognizing such elements as: landscape features, state of quarry preservation, its contrast, road accessibility, succession progress as well as society preferences and social acceptance of such structures. The aforementioned researches are the aim of the publication.

2. Material and Methods

To conduct the research concerning the landscape attractiveness of the abandoned quarries, a procedure of the quarry attractiveness assessment was used [1]. This is the procedure that includes three research methods subjected to proper modifications:

- the survey method with semantic differential that is a kind of measuring scale used for the assessment of connotations and allows to conduct the quantitative assessment in the meaning of defined concepts for various groups [7, 11],
- evaluation of the number of signals coming from the landscape through the assessment of landscape entropy conducted directly on the basis of the field research, which results allowed to group the units according to the number of existing stimuli in the area,
- the method of point bonitation which was supposed to separate physical environment and anthropogenic elements that were carriers of possible values from the quarry landscape area and to determine their attractiveness.

In the survey method, the questionnaire aimed at: determining the connotations of the word quarry, determining the distance covered by the surveyed to get to the abandoned quarry area, frequency of using, diagnosing the values through the attractiveness evaluation based on the choice of negative or positive features of the researched structures. According to the entropy, landscape should be perceived as a multi-sensory unit. The multi-sensory landscape is an existing structural and territorial reality perceived with senses and it provides a set of signals through channels, making them the stimuli for receptors [2, 3]. On the grounds of the entropy research, three types of landscape were isolated (strongly stimulating, moderately stimulating and slightly stimulating). Due to that, one may deduce if a potential landscape, given the number of its stimuli, may appear attractive for tourists and encourage them to come back to a particular place. The method of point bonitation adopts the bonitation scale that shows the relation between the adopted natural or landscape variable and the number of points [8]. Generally, points are assigned to imaginary fields.

Finally, the procedure of evaluation of the landscape attractiveness of the abandoned quarries consists of the summary assessment of individual criteria, according to the formula:

\[ A_{K,K} = \left( W_p + W_{sn} + W_{sz} + W_g + W_w + W_d + W_a + W_e + W_\alpha \right) - W_n \]

where:
- \( A_{K,K} \) – quarry landscape attractiveness
- \( W_p \) – vertical differentiation indicator
- \( W_{sn} \) – natural succession influence indicator
- \( W_{sz} \) – quarry preservation state indicator
- \( W_g \) – boundary contrasts indicator
- \( W_k \) – indicator of the number of adjacent area types
- \( W_w \) – surface water presence indicator
- \( W_d \) – route accessibility indicator
- \( W_a \) – the respondents’ evaluation indicator
- \( W_e \) – entropy evaluation indicator
- \( W_\alpha \) – indicator of areas subjected to legal protection
- \( W_n \) – indicator of human unfavourable activity influence

The research on the landscape attractiveness of the abandoned quarries was conducted in 20 structures located in the areas of three countries: Poland, Great Britain and Austria. For the specific research there were selected 10 Polish units (quarries) located in the area of the Ślęża Landscape Park and its buffer zone where minerals like granite, gneiss, gabbro or serpentinite were exploited (figure 1). In order to analyse the issue more deeply, comparative studies were conducted in another 10 quarries located in Great Britain on the Jurassic Coast in Dorset county (3 limestone quarries) and in Austria in the area of town Adnet in Austria (7 units where limestone was exploited) (figures 2−3). Moreover, they are all protected by various forms of landscape protection.
The structures subjected to the research are all located in the Ślęża Region. Eight of the selected quarries are located in the Ślęża Landscape Park, and two within the borders of its buffer zone. The units have a differentiated state of preservation which is stated below:

1. Former gneiss quarry covering about 4500 m$^2$. Forceful natural succession almost completely covered the tracks of the quarry. In the area there is a small natural pond. It is located in the buffer zone of the Ślęża Landscape Park, and one can get there along a countryside path.

2. Prehistoric quarry „Pod Wieżycą” covering about 5500 m$^2$ and one can get there along a few tourist routes. It is the oldest prehistoric quarry of two-mica granite. The beginnings of the mineral extraction date 3500 years back. It was extracted especially intensively in the bronze and iron ages by the tribe of Ślężanie, who lived there. Currently, fast and forceful natural succession covers the tracks of the quarry existence. At the bottom of the quarry there is a little stream.

3. Mediaeval granite quarry covering the area of about 1000 m$^2$. Progressing natural succession almost completely covered the evidence of extracting the mineral, and now only few quern-stones left around mark some intensive labour in the past. Not far from that site there is a dirt forest road. It is located in the Ślęża Landscape Park.

4. Granite quarry in Chwałków - called Emerald Lake. It was naturally filled with water and now covers about 10,000 m$^2$, is a part of Ślęża landscape and an attraction for a number of those who visit this place to swim, dive and climb rocks. Its characteristic features are almost vertical walls of the excavation site and a good access from the local road (figure 4).

5. Granite quarry in Chwałków. It is remote about 4 km west from Sobótka centre. Its surface is about 5,000 m$^2$ altogether. This was also naturally filled with water. Similarly to the previous one, this quarry has also got very steep walls of the excavation site, which still remain well preserved. Very good access along the local road.

6. Gabbro quarry with quartz veins. It is covered with very dense vegetation and remains hardly visible, its area is about 3000 m$^2$. Numerous trees and shrubs obstruct the access to the quarry. It is located in the Ślęża Landscape Park.

7. Granite quarry located in the Ślęża Landscape Park, covers the area of about 1200 m$^2$. In its immediate vicinity is a forest road. The average state of preservation.

8. Former serpentinite quarry located by a local road about 1 km from Tapadla Pass. Forcefully progressing natural succession makes the quarry covering about 1200 m$^2$ hardly visible. The quarry is located in the Ślęża Landscape Park.

9. Former serpentinite quarry located on the slope of Mount Słupicka (335 m above sea level) covers about 2500 m$^2$. The good state of preservation. It is located in the Ślęża Landscape Park and is accessible by a forest road. It is protected as an ecological area.

10. Former serpentinite quarry located within the Ślęża Landscape Park. It covers the area of about 3700 m$^2$ and is accessible by a local road. There is a small natural pond. It is additionally protected as an ecological area (figure 5).
history of rock minerals excavation with similar methods and are subjected to many forms of landscape protection. Additionally, in Great Britain and Austria they are precious for tourism and are local attractions. As for the quarries of the Ślęza Region, they were also assigned with consecutive numbers:

11. Winspit limestone quarry covering the area of about 10,000 m$^2$ and is accessible by a local road. It is legally protected by the National Trust (figure 6).

12. Limenstone quarry located in picturesque Portland Island has the area of about 20,000 m$^2$ and from the east is adjacent to the English Channel. Because of its close vicinity to the ocean, it is a major tourist attraction. Additionally, some of its part is characterized by almost vertical walls, which is eagerly used for climbing. It is legally protected by the National Trust (figure 7).

13. Limenstone quarry also located on Portland Island and covering about 20,000 m$^2$. Its characteristic feature are slight slopes, is accessible by a public road. In its area there is a school of sculpting in stone. Legally protected by the National Trust.

14. Limestone quarry from the 15th C. Consists of visibly market terraces. It is a small site covering about 200 m$^2$. It has the average state of preservation and inconsiderable markings of natural succession.

15. Quarry where the white kind of “marble” used to be extracted. It covers the area of about 5000 m$^2$. Well preserved geological forms make the site extraordinarily picturesque (figure 8).

16. Limestone quarry covering about 1500 m$^2$. The bad state of preservation due to the fast progress of natural succession, additionally, it is naturally filled with water

17. Quarry where the red kind of limestone was extracted. It covers the area of about 7500 m$^2$. The very good state of preservation with perfectly presented geological structure.

18. Ancient quarry covering about 600 m$^2$, with precisely marked terraces. Naturally filled with water. In its vicinity there is a museum dedicated to extraction of rock minerals with an exhibition of extraction techniques and tools (figure 9).

19. Limestone quarry covering about 1500 m$^2$. It has got steep slopes and the average state of preservation. Natural succession is slowly camouflaging interesting geological profiles.

20. Quarry where also the red kind of limestone was extracted, has got only one picturesque wall. Its other parts have already been covered with vegetation. It is about 900 m$^2$ large.

3. Research result and Discussion
The sample for the survey were tourists and inhabitants of the neighbouring towns visiting the selected quarries. The questionnaire aimed to: define the connotations of the word
quarry, determine the distance covered by the surveyed to get to the abandoned quarry area, frequency of using, diagnosing the values through the attractiveness evaluation based on the choice of negative or positive features of the researched quarries. In the end of the questionnaire there was a blank for filling in with sex, age, social status, that let gather the information about the respondents.

In regard to the survey answers, polarized graphic profiles were created for evaluative characteristics of every researched structure, as well as an average mark for all researched quarries from every single region (Fig. 10-12).

The research survey proved that in the Ślęża Region the lowest mark was achieved by the quarries covered with dense vegetation and without high excavation walls. Furthermore, their accessibility for tourists is hampered too. They are evaluated as monotonous, unpleasant, boring, useless and discordant. Different marks got the quarries which excavation walls are high, are scarcely influenced by natural succession, and are easily accessible by tourists - they are assessed as differentiated, interesting, intriguing, distinctive, concordant and natural.

The quarries in Dorset county were marked high by the respondents. First of all, they were evaluated as differentiated, pleasant, intriguing natural and concordant. One of the quarries got poor marks and was perceived as dangerous due to its location by the shoreline of steep cliffs.

The evaluated quarries in the Adnet Region in Austria, where limestone was extracted, have educational paths and are protected as the Historical Heritage monuments. The areas were assessed by the respondents positively and got high marks, especially for differentiation and concordance with the neighbourhood. Furthermore, they were assessed as pleasant, relaxing and natural. Only two quarries got poorer marks. The first one due to its average state of preservation and meaningful influence of natural succession masking interesting forms, and the other one, due to its bad condition and lack of characteristic features, was perceived as monotonous and unpleasant.

Basing on the marks of the respondents, seven point groups were created. Then the 14 studied units were assigned to the partition between 0.60 points and 1.29 points, getting high scores, and another six to the partitions: -0.60 points to -0.09 points, and -0.10 points to 0.59 points. None of the units was qualified to the lowest point group with solely negative features (table 1).

<table>
<thead>
<tr>
<th>The respondents’ average evaluation</th>
<th>Ślęża Region</th>
<th>Dorset county</th>
<th>Adnet</th>
</tr>
</thead>
<tbody>
<tr>
<td>from 2.00 to 1.30</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>from 1.29 to 0.60</td>
<td>-</td>
<td>5</td>
<td>6</td>
</tr>
<tr>
<td>from 0.59 to -0.10</td>
<td>-</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>from -0.09 to -0.60</td>
<td>-</td>
<td>3</td>
<td>-</td>
</tr>
<tr>
<td>from -0.61 to -1.30</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>from -1.31 to -2.00</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

The survey based on the evaluation of the entropy of information sources let qualify the quarries into three groups, which was related to the intensity of emitted stimuli. The biggest number of the quarries (12) were in the second group – moderately stimulating landscapes, whereas another seven were qualified to strongly stimulating landscapes (table 2). The results show that the quarries are areas emitting many stimuli and thus affecting all the senses.

<table>
<thead>
<tr>
<th>The number of units resulting from the entropy divisions</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shane</td>
</tr>
<tr>
<td>12</td>
</tr>
<tr>
<td>7</td>
</tr>
<tr>
<td>5</td>
</tr>
</tbody>
</table>

Table 1 – The number of units resulting from the respondents’ average evaluation

Table 2 – The number of units resulting from the entropy divisions
The aim of using the point bonitation method in the procedure of evaluation of the landscape attractiveness was to isolate from the area of the quarry landscape the elements of physical and anthropogenic environment being the carriers of possible values, and evaluate their attractiveness. The evaluation criteria accepted for the method include:

- vertical differentiation of the area (0-5 points)
- the percentage of natural succession (0-5 points)
- contrast with surrounding areas for a particular land cover (0-3 points)
- number of neighbouring land types (0-3 points)
- occurrence of surface waters (0-4 points)
- state of quarry preservation (0-2 points)
- roads accessibility, routes (0-2 points)
- evaluation by the surveyed (0-5 points)
- entropy evaluation (0-2 points)

Additional points were also assigned for protection forms existing in the area. The protection forms existing in the area may meaningfully affect the will and frequency of visiting the quarries by tourists, and treating them as another attraction worth seeing. Thus every quarry subjected to a form of protection was assigned with an additional point. The research also took into consideration the negative anthropogenic impact that may considerably affect the perception and evaluation of the quarry. Due to that, every researched structure where disfiguring architectural elements were marked, like remains of fencing or household rubbish, got -1 point. The same evaluation was implemented for the structures located by busy roads or by operating processing plants.

Detailed results of the conducted survey research, entropy measurement and criteria measurement with point bonitation are presented in Table 3. Basing on the analyses, the quarry landscapes were assigned to four classification groups (table 4).

<table>
<thead>
<tr>
<th>Entropy divisions</th>
<th>Ślęża Region</th>
<th>Dorset county</th>
<th>Adnet</th>
</tr>
</thead>
<tbody>
<tr>
<td>Strongly stimulating landscapes over 6.00</td>
<td>4</td>
<td>3</td>
<td></td>
</tr>
<tr>
<td>Moderately stimulating landscapes from 6.00 to 3.00</td>
<td>5</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td>Slightly stimulating landscapes below 3.00</td>
<td>1</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The very attractive quarry landscapes (group I) have elements like: meaningful vertical differentiation, the good state of quarry preservation, not advanced level of natural succession, and thus huge contrast with surrounding areas, very good roads accessibility, occurrence of surface waters, which is another element increasing the attractiveness and augmenting chances for the quarry utilization. Furthermore, the survey results also point to a meaningful interest in these areas and perceiving them as interesting, differentiated and being natural landscape elements. Their characteristic feature is the biggest entropy indicator, which classifies them as strongly stimulating objects. Further, they are located in areas protected in one or a few ways.

The attractive quarry landscapes (group II) usually have a lesser vertical differentiation, and the pace of natural succession is a bit faster, and thus some precious and interesting geological profiles are covered. Furthermore, not all the quarries in the group have surface waters in their area. Road accessibility is good, marks of the respondents and entropy are also on a high level. Additionally, they are legally protected.

The little attractive quarry landscapes (group III) have elements which decrease their final evaluation. They have little heights differences; access to such a quarry is hindered due to the pace of natural succession. The majority of them do not possess surface waters, and are not contrastive to surrounding areas. Marks of the respondents are much lower, and entropy indicator also points to a much lower probability of occurring various stimuli. However, the areas have a great potential if right efforts are undertaken, aiming at, for example, decreasing the pace of natural succession or improving road accessibility.

The unattractive quarry landscapes (group IV) have the lowest parameters. They have very little vertical differences, their preservation state is very bad, while the pace of natural succession is meaningful. They are not contrastive to surrounding areas to such a degree that they remain hardly visible, and a layman cannot distinguish them from the neighbouring area at all. Furthermore, they are not accessible along the set routes. Marks given by the respondents and entropy marks are very low. The quarries from this group are not vividly differentiated and are not perceived as interesting or useful.

Table 4 – Classification groups for the evaluation of the landscape attractiveness of the abandoned quarries

<table>
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<tr>
<th>Group</th>
<th>Qualification categories</th>
<th>Total points</th>
<th>Number of units</th>
</tr>
</thead>
<tbody>
<tr>
<td>I</td>
<td>Very attractive quarry landscape</td>
<td>over 24</td>
<td>2</td>
</tr>
<tr>
<td>II</td>
<td>Attractive quarry landscape</td>
<td>from 24.00 to 16.00</td>
<td>2 2 5</td>
</tr>
<tr>
<td>III</td>
<td>Little attractive quarry landscape</td>
<td>from 15.99 to 8.00</td>
<td>3 2</td>
</tr>
<tr>
<td>IV</td>
<td>Unattractive quarry landscape</td>
<td>below 8</td>
<td>3</td>
</tr>
</tbody>
</table>
Table 3 – Detailed results of the evaluation of the landscape attractiveness of the abandoned quarries

<table>
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<th>Evaluation criteria</th>
<th>Quarry No</th>
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<tr>
<td></td>
<td>1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19 20</td>
</tr>
<tr>
<td>Vertical differentiation</td>
<td>0 2.2 0.75 5 3.75 0 1.1 0.7 2.15 0.45 4 3 1.25 1 2 1.5 2.75 1 2 1.25</td>
</tr>
<tr>
<td>Natural succession</td>
<td>0 2 0 5 5 0 2 1 2 1 5 4 3 5 4 1 4 5 2 3</td>
</tr>
<tr>
<td>Preservation state</td>
<td>0 1 0 2 2 0 1 0 1 1 2 2 1 1 2 0 2 2 1 1</td>
</tr>
<tr>
<td>Contrast</td>
<td>0 0 0 3 3 0 1 0 0 0 2.25 2.25 2.25 3 1 0 2 3 1 2</td>
</tr>
<tr>
<td>Neighbouring lands</td>
<td>0 1 0 1 1 0 1 1 1 0.3 2 2 2 1 1 1 1 1 1</td>
</tr>
<tr>
<td>Surface waters</td>
<td>1 1 0 4 4 0 0 0 0 1 3 3 3 3 0 0 1 0 1 0</td>
</tr>
<tr>
<td>Accessibility</td>
<td>0 2 0 1 2 0 1 1 2 2 2 1 2 1 2 1 2 2 2 2</td>
</tr>
<tr>
<td>Evaluation by the surveyed</td>
<td>3 3 2 4 4 2 4 2 4 4 4 4 4 4 4 3 4 4 4</td>
</tr>
<tr>
<td>Entropy evaluation</td>
<td>1 2 1 2 2 0 1 1 1 2 2 2 2 1 1 1 1 1</td>
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<tr>
<td>Protected areas</td>
<td>- +2 +2 - - +1 +2 +2 +3 +2 +1 +1 +1 +1 +1 +1 +1 +1</td>
</tr>
<tr>
<td>Unfavourable impact</td>
<td>- - - -1 - -1 - - - - - - - - - -</td>
</tr>
<tr>
<td>Total</td>
<td>5.0 16.2 5.75 26.0 26.75 2.0 14.1 8.7 16.15 13.75 27.25 23.25 20.5 18.00 18.00 10.5 19.75 21.0 15.00 16.25</td>
</tr>
</tbody>
</table>
The attractiveness studies in the Ślęża Region proved that two units are the attractive quarry landscapes, and another two were evaluated as the very attractive quarry landscapes. Three units were evaluated as the little attractive quarry landscapes due to major influence of natural succession that covers the evidence of the quarries, and for general negligence of the area. Negatively evaluated the unattractive quarries (3 units) received low marks due to major natural succession influence, lack of accessibility and difficulties with their localization. In my opinion, minor works like setting routes, marking the place, introducing some information in guides would dramatically raise the final mark.

The quarries selected for the study from Great Britain and Austria are very popular among tourists, and they have major landscape attractiveness: as many as seven units were marked as the attractive quarries, and one as the very attractive. None of the quarries was evaluated as having the unattractive landscape. However, two units were qualified as the little attractive. The quarries in Great Britain for years have been used successfully as climbing destinations, walking areas or didactic places. Local powers that noticed their potential, use them successfully and encourage people to visit the quarries during events, concerts, art exhibitions, and other that are organized there. In the meantime, tourists may admire the extracted material – in the quarries in Austria it was limestone of unique colours and structure. Moreover, the signboards inform the visitors about the history of extraction, geological structure, existing species of fauna and flora, as well as the influence and meaning of the quarries on local life.

4. Summary

Taking up the issue of the landscape attractiveness of the abandoned quarries was intended to get information in as broad area as possible if the landscape attractiveness of quarries exists and how it may increase the attractiveness of the region. The researched units located in the Ślęża Landscape Park are its integral part. They are the evidence of rich mining history in the area and its historic heritage. Unique wildlife, unusual geological phenomena, historic and cultural values determine the bright future of the area as a tourist resort. The attractive quarry landscapes may be used as additional elements or side attractions appealing to more tourists. Additionally, the areas may be used for universal social education as a particular feature, make a didactic place that may be incorporated into programs of recreational and educational trips, science and ecology classes. Implementing this idea requires the promotion of quarry landscape, elaborating a guide and proper marking of the places in the area. Similarly to Austria, a route network along the former quarries may come into existence, with the history of extraction, technology and tools, etc. In Adnet the idea is vividly recognized and popular among tourists.

Acknowledgement

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References


3D Geostatistical Modelling of a Metalliferous Mine for Resource Assessment

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Abstract The Balya district is located near the town of Balya in the province of Balıkesir, North-West of Turkey. The aim of the study is to estimate grade and modelling of reserve distribution of complex base metals mining as Balya lead and zinc mine by using geostatistical methods respect to the distribution of variables with the lowest standard error. A French mining company produced lead and zinc between 1881 and 1938 from Balya mines. Currently Eczacibasi Esan operates the Balya mine adjacent to study area since 2008. Study strives to align to international best practice for Mineral Resource reporting and it completed by considering JORC code implement. JORC reporting standards, The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The ‘JORC Code’) sets out minimum standards, recommendations and guidelines for Public Reporting in Australasia of Exploration Results, Mineral Resources and Ore Reserves. The Joint Ore Reserves Committee (‘JORC’) published their first report in 1971 and published the first edition of report in 1989. After that version, they revised and updated edition of the Code in 1992, 1996, 1999, 2004 and 2012. The code endorsed by Mineral Council of Australia and the Financial Services Institute of Australasia as a contribution to good practice. The Code have adopted by and included in the listing rules of the Australian Securities Exchange (ASX) and the New Zealand Stock Exchange (NZX). The most important aim of this study is to investigate the advance geostatistical methodologies to evaluate lead and zinc orebody located in Balya district. Lead and zinc are the two most widely used non-ferrous metals after aluminum and copper and are vital materials in everyday life.

Keywords: resource, estimation, kriging, base metal, JORC code.

1. Introduction

Geostatistical methods were found when classical statistics unsuitable for estimating disseminated ore reserves. Mineral resources and their subsequent conversion to ore reserves are of key importance to mining companies. Over time only changes has been in difference between methods using for estimation. In the beginning of mining activities, miners made basic interpretations like sampling, chemical analyses and estimation of ore thicknesses for day-to-day operations of a mine. Beside the advantage on day-to-day operations, mining companies’ reliable estimation is critical also in a feasibility studies. There is thus a requirement for high-quality interpretation and estimation to be supported by high-quality data. Any company expecting to make sound investment or operational decisions must base this on both relevant and reliable information.

The reliable estimation of mineral resources and ore reserves is critical to all mining operations irrespective of size or commodity (Annels, 1991).

Resource estimation is the most important process of creating a three-dimensional model of in situ mineralization based on drillhole samples and current geological knowledge. This process is principal for mining engineers plan, design and extract mineralization economy. Geostatistical analysts based on the ore modelling to define volumes and so the geological understanding is crucial.

The most important aim of this research is to investigate the advance geostatistical methodologies to evaluate lead and zinc orebody located in Balya district. Lead and zinc are the two most widely used non-ferrous metals after aluminum and copper and are vital materials in everyday life.

Application of 3 dimensional research approaches is used as an innovative investigations based on geostatistics to decrease the variance of estimated values obtained from kriging methods.

2. Study Area and Geology

Study area is located near the town of Balya in the province of Balıkesir, North-West of Turkey (Figure 1). Balya mines have been worked since the time of Pericles, but regular operations with the application of scientific and modern mining methods were started about 1880. A French mining company (Societe Miniere Metallurgique de Penarroya) produced 3.5 to 4 Mt of lead and zinc between 1881 and 1938, additionally produced 1.000 ton of silver and 3.000 kg of gold. After 1978
the Balya area has been divided in two licenses, which belong to ESAN Company (Old Balya) and Dedeman Mining (Hastanetepe). Currently Eczacibasi-Esan operates the Balya mine adjacent to study area since 2008.

![Figure 1. Google Earth view of Study area.](image1)

Balya region is located along the Sakarya Zone of the Pontides (Figure 2). PermoTriassic aged highly deformed and partly metamorphosed clastic and volcanic series of the Sakarya Zone is called as Karakaya Formation. Okay & Güncüoğlu (2004) gives a comprehensive review on the Karakaya Complex. They have described the unit as: “It is generally subdivided into two parts: The structurally lower part, called the Lower Karakaya Complex, consists of a mafic lava-mafic pyroclastite-shalelimestone succession metamorphosed in the greenschist and blueschist facies during the Late Palaeozoic or Triassic. The structurally upper part containing highly deformed Permian and Triassic clastic, volcaniclastic and volcanic rocks with exotic limestone blocks. There are currently two different models for the depositional setting and tectonic evolution of the Karakaya Complex. The rift model, which assumes the Karakaya Complex, formed in a Late Permian rift, which developed into a marginal oceanic basin and closed by the latest Triassic. The subduction-accretion model regards the Karakaya Complex as subduction-accretion units of the PalaeoTethys. The non-metamorphosed equivalence of the Karakaya Formation is called Balya Formation by MTA (Turkish Geological Survey) geologists. In Balya region, both Karakaya & Balya formations are cut by Tertiary volcanics (i.e. Şapçı volcanic).

![Figure 2. Tektonic map of western Anatolia showing the distribution of the Karakaya complex and related units (Okay&Güncüoğlu, 2004).](image2)

2.1. Mineralization

Kovenko (1940) studied about the mine geology and formations consisting ore in Balya district and it was identified four different ore types as porfiric ore, ore in limestone, ore between layers in sedimentary rocks and lastly as contact veins. Besides redefining the ore formations fault sets and andesite dikes were also defined in this study. Gjelsvik (1962) brought up that the Permian limestone above the Triassic formations discordantly and the ore occurrences are located in dacite and intense folded limestone. Akyol (1979) divided ore in three types as a contact type, vein type and disseminated ore. In addition, it was mentioned in this study that when the research studies developed along north, south and east side of the study area, reserve would increase in high possibility and there is acidic intrusive rock in depth or near the mine.

Study area is dominated by Tertiary volcanic, Permian limestone and Upper Triassic sedimentary formation. At a prospect scale, distinction can be made between a Neogene dacite and andesite magmatic units. Magmatic rocks are mainly dacite, rhyolite and andesite, rarely basalt.

Major fault system is closely associated with the mineralization. The general structural orientation within the prospect area is NE-SSW to N-S, with crosscutting E-W structures. The surface expression of the structures can generally be observed sub-vertical and has been interpreted to dip steeply to the north and west.

The modeling of the ore body was performed using Surpac 3D software and sectional method was used. An area of mineralization in each section is outlined considering by grade value of samples, geological formations and both previous and next sections. Mineralization extends in the middle of two adjacent cross-sections when the continuity ends in the subsequent cross-sections (Figure 3).
3. Data Management and Geostatistics

3.1. Data Management

The quality of the resource estimation is directly depending on the quality of data procedures. Drill hole sample data will be used to predict tonnages and grades. Statistical analyses will be used with geological and other technical information to make inference decisions.

3.1.1. Data Determination

All of the data examined and the studies that completed by company authorized person reviewed. Company gave the following information about data sharing procedure and it deducted that data have transparency and can be reach by anyone works related with Balya (Figure 4). All required data and findings were determined from the study of Toka (2015).

3.1.2. Density

A geological model is used to predict the mineralized volume, and this volume in turn is multiplied by its in-situ density to obtain an estimated tonnage for deposit. Any error made in density determination and estimation is directly incorporated into tonnage estimates (Rossi and Deutsch, 2014).

86 core samples had submitted to a laboratory by the company to determine the specific gravity value. After investigation into the density values considering rock type and grade values, it is concluded that the amount of density changes related with grade of the ore body. And reserve tonnage calculations has been made considering mean density values for different Pb% grade for Pb<2% ore, 2<Pb<8% ore and Pb>8% ore respectively (Table 1).

Table 1 – Density based on Pb grade

<table>
<thead>
<tr>
<th>Pb (%)</th>
<th>Specific Gravity</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt;2</td>
<td>2.71</td>
</tr>
<tr>
<td>2&lt;Pb&lt;8</td>
<td>2.87</td>
</tr>
<tr>
<td>&gt;8</td>
<td>3.76</td>
</tr>
<tr>
<td>Mean</td>
<td>2.40</td>
</tr>
<tr>
<td>Minimum</td>
<td>2.33</td>
</tr>
<tr>
<td>Maximum</td>
<td>4.21</td>
</tr>
</tbody>
</table>

3.1.3. Descriptive Statistics

Descriptive statistic completed using sample transformed data for each ore occurrences on each drillhole which population is 588. According to that basic statistic evaluation, lead values change from 0.01 to 40.60 and its mean percentage is 3.07% with a standard deviation of 4.38; zinc values are between 0.01-27.60 and its mean percentage is 1.82% with a standard deviation of 3.05; silver values between 0.25-931.24 and its mean percentage is 50.07% with a standard deviation of 85.67.

3.1.4. Histograms and Probability Plot

As briefly and remarkable explained by Isaaks and Srivastava (1989), data speak most clearly when they are organized. Histograms and probability plot graphs are one of the most useful presentations of data sets. After examining 1-m composited data’s probability plot graphs, it can deducted that lead values have seven population, zinc values have six population and silver values have ten different data population in study area as it seen their inflection points.

3.1.5. Semivariogram Functions

Omnidirectional, anisotropic and downhole variograms were determined. An anisotropic condition exists when the rate of change data values is different for different directions. Downhole variables determined by considering data for -90° dip direction. Omni-directional variogram calculated for each of the variables. Subsequently considering anisotropy maps given in Figure 5, it concluded that the best.
Figure 5. Anizotropy maps for (a) Pb, (b) Zn and (c) Ag.

4. Search Ellipsoid and Cross Validation

Dip and bearing dimension of search ellipsoid is taken as constant value as 45° as per the similarity of the ore deposit. Different values for ellipsoid geometrical dimensions are also considered from 15 m to 150 m to understand the effect of search ellipsoid dimension for the estimation methodology. In order to optimize the estimation procedure results obtained from ordinary kriging, different search ellipsoids were used for estimation. Cross validation results were also used to select the best optimize search ellipsoid. In this step of the geostatistical application, 90 different search ellipsoid were applied to gain cross validation results for three different variables that are Pb, Zn and Ag. Correlation coefficient and average kriging error parameters were used to decide the success of the estimation procedure.

After all of the analyses carried out, number of sample to krig a point is selected as 40 while ellipsoid width and length is selected as 50 m due to the high correlation coefficient as well as low kriging error variance. In order to use the most available data for the estimation, sample number for the estimation is selected as 40 m although with the less number of samples there are also quite well results in terms of higher correlation.

Cross validation results for the selected search ellipsoid parameter are given in Figure 6 for Pb, Zn, and Ag. Highly correlated relations between estimated and actual values of ReVs encourage applying geostatistical methods for the study area to obtain distribution maps for each ReVs individually.

5. Kriging Results

Ordinary kriging method was used for the estimation and omnidirectional variogram parameters used for each base metal estimation.

According to the results determined from the geostatistical applications, more than 4 million tons of ore having more than 1% Pb that is selected as geological cut-off grade that has average grade as per 4.06% Pb content (Fig 7).
production that has average grade as per 2.87% Zn content (Fig 8).

According to the results determined from the geostatistical applications, more than 4 million tons of ore having more than 10% Ag that is selected as cut-off grade for the production that has average grade as per 63.19% Ag content (Fig 9).

![Figure 9. Reserve and grade relation for Ag (gr/ton)](image)

6. Conclusions
This study covers evaluation of Balya-Hastanetepe lead and zinc ore body using geostatistical methods. Dedeman Mining company hold the license of study area and the company have 33,000 m of core drilling that totally 184 diamond drillholes up to date of study started.

Data validation and determination and was main step of the study. Considering the fact that quality of the resource estimate is directly depend on the quality of data procedures, it took into deeply.

Solid model of the ore body created using sectional method as a base of 3 dimension geostatistical investigations. So, total volume of the solid model created in this study is 1,503,167 m³.

This study mainly covers geostatistical approaches for resource estimations. Statistical investigation was then followed by spatial descriptive statistics completed. Related with spatial descriptive statistic studies semivariograms examined therefore the data distribution observed as isotropic.

Ordinary kriging method is applied to estimate the ReVs magnitude for unsampled points in the study area. Hence, distribution maps for each ReVs are then determined for the study area that shows the magnitude for each desired locations in 3-D environment. Finally, reserve and grade relations are obtained to demonstrate reserve quality of the ore deposit that is vital for ore body validation as well as mine planning.

<table>
<thead>
<tr>
<th>Resource (t)</th>
<th>Pb (%)</th>
<th>Zn (%)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>4,314,312</td>
<td>3.92</td>
<td>1.91</td>
<td>62.18</td>
</tr>
</tbody>
</table>

Table 2 – Resource results for each commodity

Acknowledgement
The Authors would like to express deepest appreciation to Dedeman Mining Company who provided the required data for this research. All findings presented in this paper is a part of MSc thesis successfully passed successfully of MSc. Eng. Ezgi Toka.

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