A total of 205 students successfully completed in 2013 the two courses that comprise the thesis or research project. This is one of the capstone projects undertaken by students over successive semesters in the final year of the MEA program, the other being mine feasibility studies. The latter is undertaken by students working in teams whereas the research project is often undertaken by students working individually. A capstone project is usually multifaceted and complex in nature. It has been defined as

...involving an authentic, project-based activity that closely relates to professional work in the field. In completing it, students must apply the discipline knowledge and capabilities they have learned, as well as generic skills.1

Its purpose is to

...synthesise their learning across the program, demonstrate holistically their development of graduate capabilities and successfully negotiate the transition to their next career stage.2

The papers in this, the third volume of the Journal of Research Projects Review, were prepared jointly by students, their academic supervisor and occasionally the student's industry supervisor. All the papers in this volume were peer-reviewed by academics from the member universities of MEA.

The MEA program is an education initiative between the four major universities in Australia that offer a degree program in Mining Engineering: Curtin University through the Western Australian School of Mines (WASM), University of Adelaide (UoA), University of New South Wales (UNSW) and University of Queensland (UQ). The program, which was launched in 2007, was collaboratively developed and is continually updated by academic staff from each of the partner institutions with advice from education consultants to ensure the latest in teaching and learning practices were embedded into it. The program is supported by the Minerals Council of Australia (MCA) who, through its various industry partners, has helped ensure MEA Graduates will be able to meet current and future challenges of the global minerals industry.

The 2013 MEA Student Conference hosted by UNSW was the largest on record with 20 students from each of the four MEA universities presenting a paper on their research project, a copy of the program is shown in Appendix 1. The list of presenters at the one-day conference is shown in Appendix 2. The 2013 conference prize winners were:

- First Prize – Kirstan LEE (Curtin)
- Second Prize – Jake SMALL (UNSW)
- Equal Third Prize – Stefan SKORUT (UNSW) and He REN (Curtin)
- Fourth Prize – Nicholas BUTEL (UQ).

The quality of the student papers and presentations at the conference was a marked improvement on previous years making even more difficult the process of marking by the judging panel comprising the Program Directors from each university. The five students who attended the conference from each university were selected on the basis of being the best achievers in the
research project. Success in the research project is assessed not only achieving its objectives but also in the manner in which the project was undertaken: the level of innovation in the approach used by the student; how they managed the project; how they dealt with issues as they arose during the course of project; the insights gained from their analysis and how these were translated to solving issues facing the mining industry; and, finally how well they were able to communicate what they did and the project outcomes.

Members of the MEA course team who coordinated the research project in 2013 are acknowledged for their efforts and contributions to improving the student experience. This team included Associate Professor Paul Hagan (UNSW) (Course Leader), Professor Roger Thompson (WASM), Associate Professor Chaoshui Xu (UoA) and Associate Professor Mingxing Zhang (UQ). It is also important to acknowledge the important role of all the academic staff who supervised the students as well as the professional staff who supported the students in completing their project.

Finally, the contribution made by the mining industry and especially those persons who assisted the students with their research projects must be acknowledged. Many of the projects were initiated while students were on vacation employment which emphasises the important contribution that work experience makes to student education. It is important that industry recognises the role it plays in contributing to student education and ensuring the continual supply of high quality graduate engineers.

Paul Hagan
Mining Research Project Course Leader
MEA STUDENT CONFERENCE ORGANISING COMMITTEE

Associate Professor Emmanuel Chanda (UoA)
Associate Professor Paul Hagan (UNSW) (Chair)
Dr Adrian Halim (Curtin)
Associate Professor Mehmet Kizil (UQ)
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CONTENTS

Block Size Distribution Analysis of a Fractured Rock Mass – Case Study: OZ Minerals’ Carrapateena Project  
G H Ball, J B Clark, M Gifford, R Rathod, C Xu and D Goodchild  
1

Progressive Damage of Hawkesbury Sandstone Subjected to Systematic Cyclic Loading  
T J Bastian, B J Connelly, C S Lazo Olivares, N Yfantidis and A Taheri  
7

A Study of On-bench Explosives Reloading at the Foxleigh Coal Mine  
E L Berriman and P C Hagan  
15

Using Artificial Neural Networks to Predict Stope Overbreak at Plutonic Underground Gold Mine  
D Boxwell, H Jang and E Topal  
21

A Comparison of Methods to Create the Initial Void for Longhole Open Stoping Blasting  
K Lee and A Halim  
27

A Maturity Scale for Indigenous Employment in the Australian Minerals Industry  
T Lucas and P Knights  
33

Comparison of Two Excavations Systems for the Mining of Lunar Regolith  
M T Lucas and P C Hagan  
39

Using Clustering Methods for Open Pit Mine Production Planning  
H Ren and E Topal  
45

Optimising Cut-off Grade Strategy for Seabed Massive Sulfide Operations  
S A Skorut and S Saydam  
51

A Critique of Selective Mining Unit Sizing at Century Mine to Optimise Productivity with Dilution  
C B Smith, M Nehring and M K Bosompem  
63

Selection of Haulage Fleet at Daisy Milano Gold Mine  
B Yiu and A Halim  
69

Appendix 1 – Program 2013 MEA Student Conference  
77

Appendix 2 – Presenters at MEA Student Research Conferences, 2009–2013  
79

Author Index  
81
INTRODUCTION

The aim of this research is to assess in situ block size distribution using the discrete fracture network (DFN) approach for the proposed OZ Minerals’ Carrapateena block caving project. The financial viability of any block caving operation is determined by the ability of the orebody to cave naturally once the fracture propagation has been initiated. It is important to identify at the mine design stage the in situ block size distribution as this will determine the need for preconditioning and the suitability of drawpoint and layout design.

The literature review conducted revealed that DFN has major applications in the field of fluid flow through fractured reservoirs (e.g. Dershowitz, la Pointe and Doe, 2004; Xu, Dowd and Wyborn, 2013). Some researchers claimed their focus on the use of DFN to conduct block size analysis for mining applications, but there were in fact very limited related publications available. This represents a unique opportunity to apply the basic principles of DFN and to develop our own method for the analysis of block size distribution that can then be used for our case study.

The application of the proposed method to OZ Minerals’ Carrapateena project will provide an insight into the in situ block size distribution within the orebody due to natural fractures. It will also provide the company with valuable information that can be used to design and assess the necessity and suitability of preconditioning for the orebody. Chosen drawpoint dimensions can also be assessed for their suitability for the corresponding block size distribution.

IN SITU FRACTURES AND BLOCK CAVING

The success of block caving mining depends heavily on the ability of the rock mass to cave naturally under the action of gravity. Ideally the caved blocks should be small enough to pass through the ore drawing structures easily. The size distribution of the caved blocks is mainly determined by the existing fractures in the rock mass when it caves. To assess
the performance of block caving these fractures have to be modelled first. The common approach to model the spatial distribution of rock fractures is to use DFNs. A brief overview of previous research in DFN is presented further to provide a background for this research, including a review of current techniques and strategies being used for the assessment of block size distribution of a fractured rock mass.

### Discrete fracture networks

DFN is a stochastic modelling technique that is used to represent the geological and various other properties of a fracture system within a certain body of a rock mass using statistical methods (Moffitt *et al.*, 2007). This technique involves constructing a number of statistically equivalent DFNs based on models derived from field observations. These observations include fracture size (fracture trace length), fracture orientation as well as fracture intensity and they are conducted either by borehole fracture mapping or scanline or window survey of exposed rock faces. The main objective of constructing a DFN model is to represent the natural fractures in a statistical framework that can then be used in numerical simulations (Jin, 2007).

In the past, DFN models have been applied in various fields to simulate the transport of fluids through fractured rock mass (eg Xu, Dowd and Mohais, 2012). In recent developments with regard to the ability to model rock mass behaviour, DFN models have been implemented in the mining industry to analyse fragmentation and wedge or key block formation for stability assessment of slopes and other rock excavations (Rogers and Moffitt, 2006; Hadjigeorgiou and Grenon, 2005). DFN is generally regarded as a preferred modelling technique as it is capable of representing the rock mass more realistically so that an analysis closer to reality can be achieved (Rogers and Moffitt, 2006).

However, very limited research has been found in using DFN modelling directly to help the design process in fractured rock masses, either for a rock excavation structure or for a mining operation. This research will explore the opportunity to expand DFN modelling as an analytical tool to help the design of block caving operations at the feasibility study stage of the mining operation.

### Mining by block caving

Block caving is becoming an increasingly popular mining method because of its features of low operation cost and high production rate. The method can be used for low-grade massive orebodies which are becoming more common as high-grade deposits are getting scarce and more difficult to find (Ford, Pine and Flynn, 2007). Block caving mining method takes advantage of the natural fractures as well as the fractures created by the preconditioning treatment in the rock mass. The rock coves gradually due to gravity once an undercut, or void, has been driven underneath the orebody (Laubscher, 2000).

The critical aspects of block caving include caving initiation, propagation and fragmentation. The fragmentation of the rock initially occurs due to the extension of fractures that form blocks once the undercut is made. Drawpoints are constructed underneath the orebody and act as collection points for the fragmented ore. Therefore it is important to understand the fracture network of the orebody and assess the potential block size distribution during the caving operation.

### Pretreatment methods

The success of the caving process is highly dependent on the existing fractures as mentioned above and if the rock mass does not contain sufficient *in situ* fragmentation, a pretreatment technique can be used. Various pretreatment techniques have been studied and implemented in the industry (Catalan *et al.*, 2012; Jeffrey, 2000; Laubscher, 2000).

Pretreatment of the rock mass typically is carried out before caving is initiated and is done in order to change the characteristics of the rock mass and to induce fractures that will enhance the caving process (Catalan *et al.*, 2012). The technique commonly includes hydraulic fracturing, blasting and/or a combination of both. Hydraulic fracturing changes the rock mass characteristics by injecting fluid into packed boreholes to create additional fractures in the rock mass. Blasting also changes the rock mass characteristics by localised damages due to the explosion. The combination of these two pretreatment techniques is known as ‘intensive preconditioning’. The success of a pretreatment program depends on the existing fractures and the borehole design (including pattern and spacing) used (Catalan *et al.*, 2012).

### Software packages

Multiple computer programs can be used to analyse removable blocks and wedge stability in rock excavations and those suggested in the literature include Dips (by Rocscience Inc), FracSim3D (Xu and Dowd, 2010), FracMan® (by Golder Associates Inc) and Resoblok (Baroudi *et al.*, 1990). For this study, the programs that were chosen to create DFNs and to conduct the analysis were FracSim3D and FracMan because of their accessibility. A comparison will be made between the results obtained from these two programs.

FracSim3D is a software package that can be used for stochastic simulation of fractures within rock mass in either two-dimensional plane or three-dimensional volume (Xu and Dowd, 2010). DFNs are created by the program through the use of Monte Carlo sampling of the probability density functions of fracture properties. The program also allows the statistical analysis of the DFN through the use of sampling planes, windows, and scanslines (Xu and Dowd, 2010).

FracMan is another software package that can be used for the generation and storage of stochastic simulations of fracture networks. The program also allows the incorporation of raw fracture data to derive parameters that are required to build DFNs. The main advantage of FracMan is its ability to analyse block volumes that are formed within the DFN using the Block Size Analysis (by Golder Associates Inc) add-on and the ability to analyse rock wedges that form on excavation boundaries and their volumes using the RockBlock (by Golder Associates Inc) add-on. FracMan has been cited numerous times in various studies (Dershowitz, La Pointe and Doe, 2004; Tollenaar, 2008; Starzec and Andersson, 2002; Merrien-Soukatchoff, Korini and Thoraval, 2011).

### CARRAPATEENA PROJECT

OZ Minerals’ Carrapateena Project, located in South Australia, is an iron oxide copper-gold (IOCG) deposit that has been extensively studied based on a 75 km drilling program undertaken at the site between 2007 and 2009. The deposit is hosted in a very strong hematite breccia complex that is overlain by 500 m of a sedimentary sequence. Copper is mostly in the form of chalcopyrite (OZ Minerals, 2013). The deposit can be split into three main sections: the main lens, the north-east lens and the north-west lens. The top of the...
A cylindrical deposit is located 470 m below the surface and the mineralisation extends over a vertical height of approximately 1000 m (OZ Minerals, 2013).

The considered options for mining the deposit included sublevel caving, sublevel open stoping and block caving. SRK Consulting was contracted by OZ Minerals to conduct a block caving scoping study for the Carrapateena Project. Within the scoping study, two methods were considered: one being a single-lift block caving option and one with a double-lift block caving option. It was concluded by SRK Consulting that the double-lift option was the optimal design for mining the Carrapateena deposit because of the vertical extent of the orebody and geotechnical considerations. The double-lift option involves splitting the deposit into two separate caving operations with the maximum height of 500 m (Figure 1).

The design for the extraction level and the drawpoints for each lift were also outlined in the study. To ensure that the blocks can be extracted from the drawpoints, the maximum size of each caved block needs to be 39 m following the primary fragmentation and 17 m after secondary fragmentation.

**METHODOLOGY**

To generate the stochastic fracture models a large amount of data needs to be collected and analysed in order to derive the parameters required. Two approaches are then used for block size distribution analysis of the rock mass. The first method involves the construction of fracture networks using FracSim3D and exporting the generated fracture network data for analysis with a visual basic for applications (VBA) macro developed in this research. The second method uses FracMan to generate DFNs from the same fracture data and then to conduct the assessment using the inbuilt block analysis tools. The process of data acquisition, analysis and the construction of the fracture models is described in details in the following sections.

**Analysis of existing fracture data**

Data is needed to derive statistical distribution parameters to be used as inputs for the DFN model. In general, fracture data can be collected using various methods such as scancial surveys, areal (window) surveys, oriented core fracture mapping and borehole imaging.

In our study, the first step taken for this project is a review of company technical reports to obtain the relevant geotechnical data for the Carrapateena site. This involved the acquisition of available geotechnical data from OZ Minerals which was primarily oriented core fracture mapping. This data set was processed with the relevant fracture set information being transferred into a spreadsheet for further analysis. The dip and dip direction of each of the identified fractures are calculated along with the trend and plunge recorded for the borehole.

**Classification of fracture sets**

Given that the deposit is at some depth below the surface without any of the lithology of interest exposed at the surface, data collection methods such as areal surveys cannot be used. Instead the data relating to fracture orientation and density were sourced from geotechnical surveys undertaken by OZ Minerals and other previous leaseholders. As mentioned, these surveys mainly involved oriented diamond drilling cores which can provide critical information about fracture location and orientation. Given that approximately 500 m of overburden consisting of sedimentary lithology covers the orebody of interest, boreholes within this section were omitted from the fracture assessment. Therefore the final fracture model was constructed from the data of 43 boreholes over a total length of 45 km. Note: the omission of the overburden in the analysis could potentially be an issue as the overburden could be the potential sources of unexpected cave propagation and ore sterilisation. Further assessment of its influence on the caving operation is recommended.

Fracture locations were found by calculating the corresponding depth of the borehole at the point where a fracture has been observed in the core sample. Similarly fracture dip direction and dip angle are calculated based on the recorded alpha and beta angles from the survey plus the trend and plunge of the borehole. The alpha angle recorded in the fracture mapping is the acute angle between the core axis and the long axis of the ellipse, between 0–90°. The angle which forms between the reference line along the core and the ellipse apical trace is the beta angle, measured in a clockwise sense (0–360°).

Unfortunately fracture persistence can not be measured by oriented cores and in the absence of a scanline or an areal survey, four lognormal distributions were used for the DFN models as lognormal distribution is regarded as the most common type of distribution for rock fracture sizes (Xu and Dowd, 2010). Following the advice from the geotechnical department of OZ Minerals, a mean fracture size of 4.0, 7.5, 10.0 and 15.0 m were tested. Each of these mean fracture sizes were modelled with an identical standard deviation of 1.82 m.

**Fracture intensity calculation**

Fracture intensity is an important input parameter needed to construct a DFN model. After the fracture set classification, the data was then used to calculate a linear fracture intensity (or spacing) for each individual fracture set. This process
involved initially partitioning the data into their respective identified fracture sets. A correction was then made to the recorded downhole depth to convert it to the true vertical depth using the plunger angle of the borehole. Histogram analysis was then used to determine the number of fractures interpreted within 50 m intervals from 500–2500 m. The number of fractures was then divided by the bin width to generate the plot of linear fracture intensity (# fractures/m) against depth. A further correction is needed to the calculated figures that must be divided by the number of boreholes where the data was taken from for each interval. Failure to carry out this step would result in the linear fracture intensity being grossly overestimated. As was anticipated, the linear fracture intensity decreased with depth and the actual relationships vary depending on fracture sets.

In order to calibrate the fracture model generated in FracSim3D, the relationship between one-dimensional fracture intensity and three-dimensional fracture density needed has to be found for each fracture set. This relationship was found by generating each fracture set with varying volumetric density values of 0.0001, 0.001, 0.01 and 0.1 (# fractures/m³) and taking scanline samples using FracSim3D’s plane sample function. This involved taking three vertical scanline samples within two planes both oriented parallel to the z-axis, one perpendicular to the x-axis and one perpendicular to the y-axis.

In order to minimise the sampling influence due to edge effects, these scanlines were 36 m in length and offset by 15.0 m. The average number of fractures intercepted by the scan lines was then divided by the length of the scan lines to give the linear intensity (# fractures/m) for each set of volumetric densities. This provides the relationship which in turn is used to interpolate the volumetric density for the measured linear intensity required as the input for each fracture set at a particular depth for the DFN model. Figure 2 shows a calibration example for Fracture Set 3.

**Construction of fracture models**

Once the fracture orientation properties and intensities had been sourced and corrected the construction of the fracture models using both FracSim3D and FracMan could be conducted.

**FracSim3D**

To assess the influence of models on the analysis results, a total of 12 models were generated. Each model replicates an individual scenario using the four different mean fracture sizes and fracture intensities for each individual fracture set calculated at the top, middle and bottom of the formation. An example of the generated fracture model is shown in Figure 3.

**FracMan**

The volumetric fracture intensity found in FracSim3D was then input into FracMan for all 12 models. This process generated statistically equivalent models to those generated in FracSim3D. An example of the generated FracMan fracture model is shown in Figure 4. The block size analysis was then performed using the Sybil-Frac algorithm available in the FracMan software. The Sybil-Frac algorithm overlays fine grid cells throughout the volume of investigation. The number of grid cells present within a closed polyhedron (i.e. a formed block) are counted and then multiplied by the grid cell volume to determine the volume of each polyhedron.

As part of the case study, the developed VBA code was used to analyse wedge volumes intersecting a selected plane within the model and the results were compared with those calculated from FracMan. The relationships between block
volume and fracture intensity are found to be consistent between the two methods. The results presented here are those based on the FracMan output.

RESULTS
The following results were summarised using FracMan’s block analysis tool applied to the fracture models generated using the derived relationships between fracture intensity, size and block volumes. From FracMan’s block volume analysis it was found that when the mean fracture size was 10.0 m or less, a majority of the rock mass was intact. This indicated that only large blocks were formed and therefore the average block volume is extremely high. Table 1 shows the percentage of intact rock mass for each mean fracture size at different depths. This is calculated by dividing the volume of the largest identified block by the total volume of the model. As an example, for a mean fracture size of 15.0 m at the top and middle sections of the orebody, the intact rock proportion was 10 per cent and 23 per cent respectively.

Figure 5 shows the relationships between average block volumes and the mean fracture size used for the model. As expected, the average block volume decreased with increasing mean fracture size. This is consistent with other analysis as increasing fracture sizes will indirectly cause the increase in fracture intensity and therefore the average block volume will decrease. The average block volume also decreases from the bottom of the orebody upwards as fracture density increases (top part of the orebody is more fractured).

ANALYSIS AND DISCUSSION
As demonstrated, an increase in mean fracture size leads to an increase in fragmentation in the rock mass, hence reductions in the proportion of intact rock and the average block size. For example, an increase in mean fracture size from 10.0 to 15.0 m causes a significant decrease in intact rock mass from 74.6 per cent to 10.3 per cent at the top of the formation. Additionally the average block volume for the larger mean fracture size (15.0 m) model is 5 m³ compared with 14 m³ for the model with the smaller mean fracture size (10.0 m). This demonstrates the sensitive influences of the mean fracture size on the final block size distribution within the orebody. As the mean fracture size is the only parameter that can not be derived from oriented core fracture mapping and therefore its value can only be assumed in this case, the final mean fracture size for the DFN model to be used to help the mine design must be chosen with great care. It is recommended that further investigations are necessary to derive a more reliable estimate for the mean fracture size.

Block size distribution
As an example, Figure 6 shows the cumulative distribution of the block volumes formed at top section of the orebody when mean fracture size of 15 m is used. This curve accounts for each of the blocks generated in the model and is a better representation of the whole rock mass at that level in the formation in comparison to a single deterministic average block volume value. Note the maximum possible block volume is 1600 m³, which suggests a minimum side length of approximately 12 m for possible large caved blocks. This assessment can help to determine the adequacy of drawpoint design or the necessity of orebody preconditioning. Note this curve is derived from only one simulation (realisation), ie one possibility. In practical applications, many simulations will be required in order to derive a more reliable estimation of the characteristics. Another point worth mentioning is that a comprehensive sensitivity study is necessary in practice to assess the influences of various fracture parameters, particularly the mean fracture size as it is a parameter with the greatest uncertainty in this case. This study is beyond the scope of the current research due to time constraint. However, the framework and the assessment procedure have been developed in this work.

From our analysis, the majority of the mean fracture sizes correspond to a rock mass which was primarily intact. For example, if the in situ rock mass contains discontinuities with a mean fracture size less than 10.0 m, block caving may not be suitable in this case without first conducting a preconditioning fracturing program. The best scenario modelled was for a mean fracture size of 15.0 m at the top of the formation. This resulted in the smallest intact rock mass of 10.3 per cent with 80 per cent of the block volumes being less than 1000 m³.

Impact of results on mine design
Based on our initial and naïve assessment, the majority of the fracture models explored suggest that the orebody has high proportion of intact rock mass. The proportion of the intact rock increased with depth. The current design of a block caving with two lifts could remain feasible with the
introduction of a form of preconditioning fracturing plan to increase the intensity and size of fractures. The extent of the pretreatment program will be related to the amount of artificial fracturing required to reduce the average block volume to a satisfactory level. A balance will need to be found between the costs of pretreatment and the economic benefits of reducing the block volume. Details of the pretreatment design is beyond the scope of our research, however the technique developed in this work can be incorporated in the process to assess the suitability of a treatment program. If a hydraulic fracturing of packed borehole is used, it is possible to monitor the seismic events generated during the hydraulic fracturing process. These events can then be used to provide a more reliable estimate of the fracture size which is the critical parameter that needs to be determined in this case. The seismic events can also be used to construct more realistic conditional fracture models using technique such as the Markov chain Monte Carlo simulation (Xu, Dowd and Wyborn, 2013) so a more realistic design and assessment of the caving operation can be achieved.

CONCLUSIONS AND RECOMMENDATIONS

For this case study, fracture models were created using both FracMan and FracSim3D. Analyses of these generated models provided information on the distribution of block volumes and their dependence on fracture parameters such as the mean fracture size of the DFN model. The analyses were completed at different depths within the orebody and the results indicated that a smaller percentage of rock mass was intact at shallower depths. In addition to this, it was found that block volume distribution is sensitive to the mean fracture size used in the model and large fracture sizes resulted in significantly small average block volumes.

Based on the mean fracture sizes modelled, the block volume analysis indicated that a preconditioning treatment might be necessary in this case to ensure an efficient block caving operation. A more intense preconditioning will be needed in the lower part of the orebody due to the larger proportion of intact rock.

Further investigation could be undertaken to analyse variation in fracture size distribution with depth as it has been demonstrated to be a critical parameter in this case. It is possible that in conjunction with fracture intensity, fracture size will also decrease with depth. This could alter the fracture model and resultant distributions of block sizes significantly.

ACKNOWLEDGEMENTS

We acknowledge Golder Associates for giving us the permission to use their FracMan software. Without this software, our research would not have progressed as far as it did.

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REFERENCES


Progressive Damage of Hawkesbury Sandstone Subjected to Systematic Cyclic Loading

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ABSTRACT
An experimental investigation was carried out on the Hawkesbury sandstone to identify and predict the change in mechanical properties of the rock during uniaxial and triaxial cyclic compressive testing. Cyclic tests were completed at different stress levels and unloading amplitudes. Damage increased with an increase in unloading stress level and unloading amplitude. Results indicate that leading up to unstable crack propagation, approximately 65 per cent of the cumulative axial strain (measured at the peak of each cycle) occurs prior to the critical damage point with a rapid increase after this point. The rate at which strain accumulated after the critical damage point is dependent on the stress level at unloading and unloading amplitude. A preliminary damage model was proposed to predict reduction in the peak strength and tangent Young’s modulus of a rock due to cyclic loading. For future development of this work, a methodology to conduct cyclic loading tests was proposed.

INTRODUCTION
The underground excavation of rock results in the disturbance and redistribution of the in situ stress field. As a result, the surrounding undisturbed rock compensates for the excavated rock by supporting the redistributed stresses. With continuous mining activity, the burdened rock mass experiences further loading and unloading in the form of cyclic loading due to drilling and blasting, truck haulage vibrations and mine induced seismicity. This study concentrates on the changes in the strength and deformability of a brittle rock subject to systematic cyclic loading.

The notion of fracture damage and quantification of its effects on the mechanical properties of a material has developed into a study known as damage mechanics (Eberhardt, Stead and Stimpson, 1999). Stress-induced brittle fracture damage is caused by the initiation, propagation and coalescence of microfractures within a rock during compression. Brace (1964) and Bieniawski (1967) define the five stages of brittle fracture during compression. These stages have since been discussed in a number of publications (Goodman, 1989; Lajtai, Carter and Scott Duncan, 1991; Martin and Chandler, 1994; Gatelier, Pellet and Loret, 2002).

This study primarily focuses on the two stages, crack damage threshold and ultimate rock strength. The crack damage threshold, \( \sigma_{\text{cd}} \), is the critical value at which cracks begin to propagate in an unstable manner. Unstable crack growth continues until numerous microfractures have coalesced and the rock can no longer support any load, whether it be constant or increasing and results in failure of the material (Eberhardt et al, 1998; Eberhardt, Stead and Stimpson, 1999). In this study the crack damage threshold \( \sigma_{\text{cd}} \) was determined by monitoring the secant Young’s modulus \( E_{\text{sec}} \); the crack damage threshold occurs when \( E_{\text{sec}} \) reaches a maximum as shown in Figure 1.

To date, a number of models have been established to predict damage of a rock as a result of compressive cyclic loading (Eberhardt, Stead and Stimpson, 1999; Xiao et al, 2009, 2010; Chen et al, 2006; Taheri and Tani, 2013). However, each model fails to consider cyclic loading and when it is considered, the models do not take into account critical mechanical properties and cyclic loading parameters which play a significant role in the progressive damage of a rock. Therefore it is required to

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develop a model for brittle rocks which considers the various cyclic loading parameters.

In this study, it is aimed to study the effect of various parameters including the magnitude of the stress level at the initiation of cyclic loading, \( q_{un} \), the unloading amplitude of the cyclic loading, \( q_b \), damage increment (the number of cyclic loads), \( i \) and the confining pressure/depth of the rock mass, \( \sigma_3 \) on mechanical properties of Hawkesbury sandstone. It is also intended to develop a preliminary damage model to predict reduction in the peak strength and tangent Young's modulus due to cyclic loading.

**METHODOLOGY**

Uniaxial and triaxial compression testing was used to replicate systematic cyclic loading in the laboratory. To perform uniaxial and triaxial tests, a closed-loop servo-controlled testing machine with a loading capacity of 250 kN and a loading rate capability in the range of 0.001–10 mm/s was used. A high-pressure Hoek cell and hydraulic pressure system were used to apply and control the confining pressure for triaxial tests. The machine is equipped with a linear variable differential transformer (LVDT) to measure axial displacement and control axial loading. All samples were cored and prepared according to International Society for Rock Mechanics (ISRM) standards with approximate dimensions of 100 mm in height and 42 mm in diameter. A ramp waveform was selected for the machine to apply cyclic loading. Pairs of axial and lateral strain gauges were secured on opposite sides of the specimen to measure axial and lateral strains as shown in Figure 2.

The following procedure was followed to protect strain gauges from being damaged in the Hoek cell:

- all the voids on the specimen surface that would otherwise be in contact with the strain gauges were filled with epoxy precoating (PS XH107F) to eliminate rough surfaces, left to dry for 24 hours and sanded back to a smooth surface
- once the strain gauges were secured to the specimen surface, they were coated with polyurethane for protection
- immediately before conducting triaxial tests, microcrystalline wax was then applied over the strain gauge to allow movement between the strain gauge and the triaxial Hoek cell jacket.

**RESULTS**

A total of 24 tests were conducted for this study. This consisted of 12 uniaxial tests (three monotonic, nine cyclic)

\[ q_f : \text{peak strength}; \quad q_{un} : \text{stress at which cyclic loading begins}; \quad q_b : \text{stress at which cyclic load is bound}. \]
and 12 triaxial tests (four monotonic, eight cyclic). The test results were compiled to plot graphs of the applied deviatoric stress ($q$) against the axial strain measured by the LVDT, the axial ($\varepsilon_{ax}$) and lateral strain ($\varepsilon_{lat}$) recorded by strain gauges and the volumetric strain ($\varepsilon_{vol}$), which was calculated using strain gauges results.

**Monotonic compressive loading**

Figure 4 displays uniaxial monotonic testing results for samples HS1 and HS3. Monotonic testing was used to obtain the failure strength ($q_f$) of an unconfined specimen to determine the stress level at which cyclic loading should be initiated ($q_{un}$). The average peak monotonic strength for uniaxial loading was 47.1 MPa. The amount of axial strain measured by the axial external LVDT is larger than that recorded by strain gauges. This is because of the bedding errors and also deformations in loading piston, cap and axial loading system, which were included in the recorded displacements by the axial external LVDT. Figure 5 displays triaxial monotonic testing results for samples HS15 and HS18. These tests were used to obtain the failure strength ($q_f$) of a specimen confined at $\sigma_3 = 4$ MPa to determine the stress level at which cyclic loading should be initiated ($q_{un}$). Due to sample variability a wide range of monotonic strengths were measured, ranging between 61 MPa and 82 MPa. The average peak monotonic strength for triaxial loading was 69.2 MPa.

**Cyclic compressive loading**

Figure 6 summarises results for uniaxial cyclic loading tests HS4 and HS7. Both tests were conducted under the full unloading (100 per cent unloading amplitude) cyclic testing regime, requiring 70 and 106 cycles to failure respectively. Cyclic loading was initiated at 42.6 MPa and 40.3 MPa respectively causing a failure of the samples at 87 per cent and 82 per cent of the peak monotonic strength.

Figure 7 summarises results for triaxial cyclic loading tests HS12 and HS16. HS12 was conducted with unloading amplitude of 80 per cent and HS16 conducted under full unloading (100 per cent). Cyclic loading was initiated at 74.6 MPa and 68.8 MPa, requiring 42 and 129 cycles until failure respectively.

A summary of the successful systematic cyclic loading tests can be found in Table 1.

**ANALYSIS OF RESULTS**

**Predicting $q_f$**

The highest stress experienced by samples that underwent cyclic loading was $q_{un}$, ie the peak strength was never reached and therefore was not known for any of the samples which underwent cyclic loading. Due to the large inevitable variation among samples tested in this study, the peak monotonic strengths ($q_f$) of samples subject to cyclic loading were unknown. However, to quantify the degradation in strength due to cyclic loading, the peak monotonic strength ($q_f$) was required; therefore a method for predicting the peak strength of a specimen was devised.

By using monotonic test results, the ratio of $q_{cd} / q_f$ was determined for $\sigma_3 = 0$ MPa and 4 MPa as 0.988 and 0.973 respectively. Once this was established, the crack damage threshold $q_{cd}$ could be determined (stress corresponding to the maximum secant Young’s modulus, Figure 8) for cyclic loading tests which had a $q_{un} \geq q_{cd}$. The predicted peak strength, $q_{fp}$, could then be calculated from the following formula:

\[
q_{fp} = \frac{q_{cd}}{q_{un}} \cdot q_f
\]
Mining Education Australia – Research Projects Review

T J Bastian et al

For cyclic loading tests which had a $q_{\text{fp}} < q_{\text{cd}}$, a curve of best fit was established for the secant Young’s modulus ($E_{\text{sec}}$) versus $\varepsilon_{\text{ax}}$ curves in order to predict the maximum secant Young’s modulus, $E_{\text{sec(max)}}$. An example of this is shown in Figure 8. Maximum $E_{\text{sec(max)}}$ which is measured during axial loading in monotonic test, has been used previously as a critical point in which extensive damages happen in a rock specimen after this point (Taheri and Tani, 2008; Taheri and Chanda, 2013). This concept is used in this study as well to determine critical

Uniaxial: $q_{\text{fp}} = \frac{q_{\text{cd}}}{0.988}$

Triaxial: $q_{\text{fp}} = \frac{q_{\text{cd}}}{0.973}$

where:

$q_{\text{cd}}$ = crack damage threshold

$q_{\text{fp}}$ = predicted compressive strength of the specimen

FIG 6 – Uniaxial cyclic loading test results for HS4 and HS7.

FIG 7 – Triaxial cyclic loading test results for HS12 and HS16.
Progressive Damage of Hawkesbury Sandstone Subjected to Systematic Cyclic Loading

It was observed that during cyclic loading the rock accumulated strain relatively uniformly followed by a rapid strain increase as it headed towards unstable crack propagation. This rapid accumulation in strain occurred on average at approximately 65 per cent of the cumulative axial strain. The point at which the rapid strain accumulation occurred was termed the critical damage point \( \omega_c \). Figure 10 shows an example of results which in this test \( \omega_c \) has been measured at 58 per cent of the cumulative axial strain.

It was noted that at higher unloading stress levels the more rapid the accumulation of strain was after the critical damage point. It was also observed that the larger the amplitude of cyclic unloading the greater accumulation of damage experienced by the specimen and a slower accumulation of strain beyond the critical damage point. The same observations in the trend of strain accumulation were observed in the axial, lateral as well as the volumetric strain graphs. Therefore, it was concluded that there is an increase in damage with an increase in stress level at unloading and unloading amplitude.

**Deformability**

To quantify the effects of cyclic loading on the deformability of brittle rock, Young's modulus and Poisson's ratio were calculated. The tangent Young's modulus and Poisson's ratio were determined at 50 per cent of \( q_{un} \) for each damage increment.

In all tests there is a moderate decline in the tangent Young's modulus of each sample until reaching its elastic limits, after which there is a rapid decrease in stiffness until unloading. Figure 11 show an example of results for a cyclic triaxial test. Poisson's ratio initially increases rapidly, followed by a stage of slower accumulation. During this second stage, the rate of increase remains relatively constant as shown by the linearity of graphs in the middle cycles. As the sample begins

<table>
<thead>
<tr>
<th>Test type</th>
<th>Test #</th>
<th>Unloading amplitude (%)</th>
<th>Initiation stress, ( q_{un} ) (MPa)</th>
<th>Failure stress, ( q_f ) (MPa)</th>
<th>Strain at failure, ( \varepsilon_{ax(f)} ) (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Uniaxial HS2</td>
<td>100</td>
<td>47</td>
<td>44.2</td>
<td>0.48</td>
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<tr>
<td>Uniaxial HS4</td>
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<td>42.6</td>
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<td>0.44</td>
<td></td>
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<tr>
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<td>44.1</td>
<td>42.7</td>
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<tr>
<td>Uniaxial HS7</td>
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<td>40.3</td>
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<tr>
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<td>74.8</td>
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<tr>
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<td>69.4*</td>
<td>73.1</td>
<td>0.504</td>
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<tr>
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<td>74.6</td>
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<tr>
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<td>0.49</td>
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<td>74.6</td>
<td>68.9</td>
<td>0.45</td>
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<tr>
<td>Triaxial HS16</td>
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<tr>
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<td>61.5</td>
<td>61.2</td>
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<td></td>
</tr>
<tr>
<td>Triaxial HS22</td>
<td>50*</td>
<td>61.8*</td>
<td>68.6</td>
<td>0.61</td>
<td></td>
</tr>
</tbody>
</table>

* This value was altered during the test to cause the specimen to fail.

**Cumulative strain**

When a sample experiences a stress that exceeds the yield stress of the sample, it will incur an incremental cumulative permanent strain which increases with each damage increment (or cycle). This cumulative strain (\( \omega \)) is the difference in strain value from the peak of each successive loading cycle to the peak of the primary loading curve (Figure 9). It can be used to represent the non-visible damage incurred by a specimen.

TABLE 1

Summary of uniaxial and triaxial cyclic loading testing.  

<table>
<thead>
<tr>
<th>Test type</th>
<th>Test #</th>
<th>Unloading amplitude (%)</th>
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FIG 9 – Cumulative strain measurement during systematic cyclic loading.

FIG 10 – Cumulative axial strain graph for the uniaxial test, HS7. The critical damage point occurs after damage increment 86.
to approach failure, Poisson’s ratio begins to rapidly increase until failure.

**Unloading with respect to the crack damage threshold**

The results indicated that when unloading commenced above the crack damage threshold $q_{cd}$, the specimen attained a rapid accumulation of damage as it approached failure (Figure 12, HS8). In contrast, when specimen was cyclic loading starts below $q_{cd}$, quite large number of cycles required to create damage into the rock and specimen showed to reach a standstill after a certain amount of cycles (Figure 12, HS22). It can be concluded that if a specimen is unloaded below $q_{cd}$ after a certain amount of cycles it will not incur further permanent damage with repeated exposure to cyclic loading.

**Confinement**

Results presented in Figure 13 show that for an increase in confining pressure a higher stress level at start of cyclic loading required. As well as this, it was found that for rock specimens under the same unloading amplitude the one at higher confinement required a greater amount of cycles for failure to occur. Note that in this study only cyclic triaxial tests at $\sigma_3 = 4$ MPa were performed. More tests at different confining pressures are required to completely validate this conclusion.

**DAMAGE MODEL**

A preliminary cyclic loading damage model is proposed for the Hawkesbury sandstone with reference to the change in the peak strength and the reduction in stiffness. The change in these properties was explored by considering controlled cyclic loading parameters such as $q_{un}$ and $q_b$ as well as the axial strain and damage increment at failure. The results of this study resulted in the conceptual relationships shown in Figure 14.

Figures 15 and 16 represent examples of relationships obtained from the experimental results. It can be concluded that for a high stress level at unloading, a specimen will experience a smaller axial strain at failure (Figure 15) as well as a smaller reduction in the tangent Young’s modulus (Figure 16).

From the developed conceptual model represented in Figure 14, it can be seen that for a higher stress level at unloading, a smaller number of damage increments are required to fail a specimen. Therefore, there will also be a larger reduction in the tangent Young’s modulus (Figure 14f).

**PROPOSE A CYCLIC LOADING TEST METHOD**

The difficulties experienced in this study primarily revolved around the high variance among the samples tested. In multiple cases the variability in strength and stiffness of a specimen required a change in one or more cyclic loading parameters in order to fail the specimen (loading rate, unloading amplitude). As a result, a systematic cyclic loading method is proposed to overcome these issues. The method is outlined as follows:
determine the extent of specimen heterogeneity from the relationship between \( q_f \) versus \( \varepsilon \)
- ease of calculating \( q_f \) without the need of a prediction curve for heterogeneous samples through knowledge of \( q_{cd} \)
- eliminate time issues associated with failing the rock as it is loaded at or beyond the unstable crack growth region (\( \geq q_{cd} \))
- cyclic loading parameters remain constant; testing method is consistent, producing more reliable data.

**CONCLUSIONS**

The following conclusions were drawn from analysis of systematic cyclic loading:
- the higher the stress level of unloading the more rapid the accumulation of strain after the critical damage point
- the larger the amplitude of cyclic unloading the greater accumulation of damage experienced by the specimen and less rapid accumulation of strain beyond the critical damage point
- rock under confining pressure requires a greater number of cycles to failure and higher initiation stress of unloading in comparison to unconfined rock
- cyclic unloading below \( q_{cd} \) will result in rock acquiring limited amounts of permanent damage
- cyclic unloading above \( q_{cd} \) will result in permanent deformation of the rock through degradation of stiffness and accumulation of irreversible strain
- a preliminary damage model was developed to predict the degradation of the mechanical properties of Hawkesbury sandstone subjected to cyclic loading.

**ACKNOWLEDGEMENTS**

The authors wish to give thanks to Ian Cates and Simon Golding for improving the understanding of the laboratory machinery and test preparation, and conducting laboratory tests.

**REFERENCES**


A Study of On-bench Explosives Reloading at the Foxleigh Coal Mine

E L Berriman¹ and P C Hagan²

ABSTRACT

Foxleigh Coal Mine currently has a fixed plant explosives reloading facility that services its four active mining pits. For the majority of pits, the distance between the plant and pit results in excessive cycle times due to lengthy reload times creating a bottleneck in the explosives loading capacities of the mobile manufacturing units (MMUs). An analysis was undertaken to assess the potential benefits of on-bench reloading in terms of explosives loading rates and financial savings.

A time and motion study was conducted to collect data on the cycle times for the two available Orica Bulkmaster MMUs operating on-site. The data was used to assess the potential loading rates that could be achieved and cost reductions associated with three different on-bench reloading scenarios. The results were compared to the current fixed plant reloading arrangement to demonstrate the potential increase in loading rate and financial savings that could be achieved from on-bench reloading.

A unit basis comparison showed that all on-bench reloading scenarios resulted in an improvement in terms of cost per bank cubic metre for an average loading day at Foxleigh. The best scenario resulted in an increase in daily explosives loading rates as well as a reduction in costs. A risk analysis identified a number of potential risks that on-bench reloading poses to the operation, but with the management of ammonium nitrate prill supply and constant preventative maintenance of the MMUs, these risks can be substantially mitigated.

INTRODUCTION

Foxleigh Coal Mine, located in the Bowen Basin of Central Queensland, is a small open cut coal mining operation with a current annual production capacity of 3.3 Mt of product coal. It is managed by Anglo American Metallurgical Coal (AAMC) and is a joint venture between AAMC and two international organisations – POSCO and Itochu – with AAMC being the majority shareholder (Anglo American, 2012). As is common with most open cut coal operations, it is first necessary to blast and remove the overburden in order to expose the underlying coal seam. The blasting process generally uses bulk explosives vehicles to transport the explosives products from on-site storage facilities to the blast site. Foxleigh currently has two Orica Bulkmaster mobile manufacturing units (MMUs) on-site which transport, mix and deliver explosives products to the pits.

The current explosives reloading arrangement has created a bottleneck in the explosives loading capacities of the MMUs. The distance that the MMUs are required to travel to reload with explosives is quite long for some pits, resulting in long reload turnaround times that adversely impact on the loading capacities of the MMUs.

The current mine layout at Foxleigh is shown in Figure 1. Shown within each of the rectangles are the four current active pits within the mining lease, these being: WC, One Tree (OT), Pipeline (PL) and Carlo Creek (CC). Servicing these four pits is a single fixed plant ammonium nitrate (AN) reload facility located in the centre of Figure 1 designated as Orica Reload.

An analysis of the 2012 loading performance data showed that the WC Pit had the highest daily loading rate. This is

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due primarily to its close proximity to the reload facility. The aim of the study was to assess the benefits of reducing the distance of the reload facility to the active loading bench by considering on-bench reloading.

On-bench reloading involves purchasing or hiring mobile storage containers to temporarily store the explosives products. These mobile storage containers, with the aid of a prime mover/tipper, can be relocated to the bench that is currently being loaded with explosives therefore decreasing the distance from the loading bench to the reload facility. The decrease in travel distance should significantly reduce the reload turnaround times and therefore increase the daily loading rates of the MMUs.

**TIME AND MOTION STUDY**

**Data collection**

A time and motion study was conducted over two consecutive days at the PL and WC pits. Data collected included hole load times, travel times to and from the explosives reload and reload times. The four sequential steps in the process are shown in Figure 2.

Although data was collected on MMU performance for the two pits, they provide a snapshot of the loading rates for the entire operations as these pits are the furthest from and closest to the Orica reload facility respectively.

**Data analysis**

The data collected from the time and motion study was analysed to find the average reload time, production time, speed of the MMUs and on-bench delays in each cycle. Values for these parameters are provided in Table 1.

**ON-BENCH RELOADING**

To introduce on-bench reloading at Foxleigh mine, the following extra equipment would be required:

- AN storage container
- diesel storage container
- auger (delivery mechanism for both AN and diesel)
- prime mover/tipper (to relocate the storage containers).

**Scenarios**

The four scenarios that were analysed included:

- **S1** current fixed plant arrangement
- **S2** one MMU with two operators
- **S3** two MMUs with four operators
- **S4** one MMU with three operators.

Scenarios 2 to 4 involve the acquisition of the same on-bench reloading equipment, only the use of current equipment and the number of operators are altered. Scenarios 2 and 3 involve alternating even time crews with no overlaps, therefore having one operator on-site per MMU. Scenario 4 involves two operators working overlapping rosters, therefore having two operators on-site the majority of the time with only one MMU. This scenario allows for fewer delays as the second operator can take care of such issues as prill deliveries and administration paperwork, which results in higher overall loading capacity for an MMU.

**RESULTS AND ANALYSIS**

**Productivity**

Productivity refers to the daily loading rates of the MMUs on-site as measured in units of kilograms of explosives loaded per shift. This calculation was based on the data analysed from the time and motion study that included:

- average production time
- average reload time
- average travel speed of the MMU
- average on-bench delays.

These factors remain unchanged in the analysis. However, two other factors were also considered in the productivity calculation, these being:

1. the distance of the loading bench to the reload
2. the operating hours of the MMU(s) per day.

The distance between each pit and the reload for the current fixed plant setup was measured using Minescape. For simplicity, the distance from the loading bench to the on-bench reload was held constant at 1 km. The distance to each pit is listed in Table 2. The number of operating hours per day was calculated for each scenario based on a 12 h shift taking into consideration the delays which would be experienced throughout the day. These delays include pre-start meetings, emulsion prill re-load times, prill delivery escort and wait times, and pre-shift meetings.

**TABLE 1**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Mean value</th>
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<tbody>
<tr>
<td>Average speed</td>
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<tr>
<td>Delays</td>
<td>9.6 min</td>
</tr>
<tr>
<td>Reload time</td>
<td>21.4 min</td>
</tr>
<tr>
<td>Production time</td>
<td>27.5 min</td>
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</table>

**TABLE 2**

<table>
<thead>
<tr>
<th>Pit</th>
<th>Distance to fixed plant reload (km)</th>
<th>Distance to on-bench reload (km)</th>
</tr>
</thead>
<tbody>
<tr>
<td>WC Pit</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>One Tree (OT)</td>
<td>8</td>
<td>1</td>
</tr>
<tr>
<td>Pipeline (PL)</td>
<td>10</td>
<td>1</td>
</tr>
<tr>
<td>Carlo Creek (CC)</td>
<td>10</td>
<td>1</td>
</tr>
</tbody>
</table>

**FIG 2** – Steps in the mobile manufacturing unit (MMU) loading process.
times, crib, wait on blast crew, wait on other MMU and administration. The operating hours per day can be seen in Table 3.

The method used to calculate the daily loading rates for each scenario is shown in Equations 1, 2 and 3.

\[ \text{cycle time} = l + t_r + r + t_b + d \]  
\[ \text{loads/day} = \frac{\text{op hours/day} \times 60}{\text{cycle time}} \times 2 \text{ for 2 MMUs} \]  
\[ \text{t/day} = \text{loads/day} \times t/load \]

where:
- \( l \) = load time
- \( t_r \) = travel time to reload
- \( r \) = reload time
- \( t_b \) = travel time back to bench
- \( d \) = delays

The amount of explosives per load refers to the combined explosives products that the MMUs can discharge per load. This was calculated as an average from the data collected and was found to be 10.5 t.

Comparing the productivity of the fixed plant reloading setup and the on-bench reloading scenarios provides a good measure of the production benefits of moving to on-bench reloading. Figure 3 shows a comparison of the potential loading rates of the on-bench reloading scenarios to the expected loading rates of the current fixed plant reloading setup.

As the graph in Figure 3 indicates, Scenario 3 achieves a higher production rate compared to the current arrangement (S1). Scenario 4 achieves a similar result except for the WC Pit. This is because the distance between this pit and the fixed plant is only 2 km, which is much shorter than the other pits, and in effect is an on-bench reloading arrangement with two MMUs. Scenario 1 provides at best a production rate similar to the current arrangement for the CC and PL pits only, whereas for the other two pits the production rate is less.

Further analysis was conducted on the potential loading rates for on-bench reloading and the expected loading rates for the current fixed plant reloading setup. Figure 4 shows the estimated number of days it would take to load the 2013 budgeted explosives for each scenario.

Two of the on-bench reloading scenarios require fewer days to load the 2013 budgeted explosives. There are many advantages in having faster loading times and fewer number of loading days. The main advantage being it provides the potential to increase annual mining rates.

**Savings**

A cost analysis was undertaken that considered the difference between the different scenarios. The costs included in the analysis were based on the Orica contracting costs, operator costs and the costs for the on-bench reloading equipment. Fixed costs, such as explosives and potential demurrage costs, were not included as they remain the same for each scenario.

Comparing the costs for the different scenarios is an indicator of the potential financial savings that may accrue from the introduction of on-bench reloading. Table 4 provides a summary of the equipment and operators required for each of the scenarios.

A comparison of the on-bench reloading costs and the current fixed plant setup costs is shown in Figure 5. As the graph shows, Scenarios 2 and 4 have lower costs compared to the current arrangement (S1). Substantial savings in the order of 40 per cent could be achieved with Scenario 2 while the savings with Scenario 4 are slightly less at 32 per cent. These translate to savings of $990,000 and $775,000 respectively per year. There were no significant savings found with Scenario 3.

---

**TABLE 3**

Daily operating hours per mobile manufacturing unit (MMU).

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Number of operators</th>
<th>Operating hrs/d/MMU</th>
</tr>
</thead>
<tbody>
<tr>
<td>S1: Current fixed plant setup</td>
<td>2</td>
<td>5.5</td>
</tr>
<tr>
<td>S2: On-bench reloading 1 MMU</td>
<td>2</td>
<td>6.5</td>
</tr>
<tr>
<td>S3: On-bench reloading 2 MMUs</td>
<td>4</td>
<td>5</td>
</tr>
<tr>
<td>S4: On-bench reloading 1 MMU</td>
<td>3</td>
<td>8</td>
</tr>
</tbody>
</table>

**FIG 3** – Comparison of fixed plant and on-bench reloading daily productivity rates.

**FIG 4** – Comparison of different scenarios to load the 2013 budgeted amount of explosives.
Combination of productivity gains and cost savings

A comparison of the combination of the effects of potential changes in loading rates and cost savings of each scenario is shown in Figure 6. The graph shows that the biggest cost saving results from one MMU on-bench reloading while the highest loading rate results from two MMUs on-bench reloading.

It is difficult to weigh up the best option for on-bench reloading based on the results in this graph alone. Therefore a comparison was made based on the potential cost per unit of explosives. This value accounts for the cost per bank cubic metre for an average day of loading using Equation 4. The results of this unit cost basis comparison are shown in Table 5.

\[
\text{total cost for } 2013 \times \frac{\text{kg explosives }}{\text{kg loaded/day}}
\]

All on-bench reloading options provide a significant improvement in terms of loading rates and cost savings. This is highlighted by the significant reduction in the cost per bank cubic metre for an average day of loading. This analysis suggests the best option is Scenario 4 with one MMU on-bench reloading and three operators. This contrasts with Scenario 3 that achieves the highest daily loading rates and Scenario 2 and the option that achieves the largest cost savings.

TABLE 4
<table>
<thead>
<tr>
<th>Item</th>
<th>S1</th>
<th>S2</th>
<th>S3</th>
<th>S4</th>
</tr>
</thead>
<tbody>
<tr>
<td>MMU</td>
<td>2</td>
<td>1</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>Fixed plant</td>
<td>1</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>MMU operator</td>
<td>4</td>
<td>2</td>
<td>4</td>
<td>3</td>
</tr>
<tr>
<td>AN transit bin</td>
<td>-</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Auger</td>
<td>-</td>
<td>2</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td>Prime mover</td>
<td>-</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Fuel storage</td>
<td>-</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
</tbody>
</table>

Table 4: Equipment and personnel requirements for each scenario.

FIG 5 – Comparison of costs between the four scenarios based on 2013 costs.

FIG 6 – Comparison of current fixed plant and on-bench reloading productivity and savings for 2013.

Risks

Considering the different scenarios and the risks associated with each of these it was found that they could be divided into three different areas. These different areas are the prill storage, the number of MMUs on-site and the number of operators on-site. These risks all have the potential to result in an interruption of loading on-site, and considering that blasting is an integral component of the mining process, this would be a significant risk for the operation.

The risks associated with the prill storage are mainly associated with on-bench reloading. This is due to this setup not having the large storage facility on-site that the fixed plant reloading process does. This large fixed plant storage facility generally provides a buffer for any mistakes in ordering prill or late prill deliveries, although in the past loading on occasion was prevented due to depletion of prill on-site. The on-bench reloading option has a greater risk of causing a stoppage in loading due to the depletion of the prill on-site as the storage facility is smaller. Therefore, mistakes in ordering prill and late prill deliveries pose a significant risk to on-bench reloading.

The risks associated with the number of MMUs on-site relate to potential MMU breakdowns, unexpected maintenance of MMUs and damage to the MMUs. If any of these arise then an MMU cannot load and if there was only one MMU on-site then no loading could take place. With two MMUs on-site it is less likely there would be delays caused by both units being down for maintenance.

The risks associated with the number of personnel on-site are related to the potential for an operator to be absent from site. This is particularly an issue if there is only one MMU on-site and one operator for that MMU, since if an operator is absent then no loading will occur.

CONCLUSIONS

A study of on-bench reloading at the Foxleigh Coal Mine has shown it would be beneficial in terms of an improvement in daily explosives loading rates and cost savings compared to the current fixed plant reloading arrangement. Each of the on-
bench reloading scenarios considered in the study resulted in a saving for the mine with two out of the three scenarios resulting in better daily loading rates. On a unit basis comparison, taking into account the daily loading rates and savings, all the on-bench reloading scenarios would provide a better outcome than the current fixed plant scenario. Overall, the best option is Scenario 4, which provides on-bench reloading with one MMU and three operators.

This scenario provides the following benefits in comparison to the current fixed plant reloading arrangement:

- 15 per cent increase in daily explosives loading rate that is equivalent to an additional 10.4 t of explosives
- 32 per cent or $775 000 saving/year
- 39 per cent or $7/bank cubic metre (bcm) decrease for an average day of loading.

Overall, this on-bench reloading scenario would result in higher productivity at a lower cost.

The analysis of on-bench reloading identified several potential risks associated with this reloading arrangement. These risks are the depletion of prill on-site and MMUs being out of operation, both of which could result in an interruption to explosives loading. These risks could be mitigated by ensuring that a constant supply of ammonium nitrate is maintained to the mine site and increasing the level of preventative maintenance to the MMUs.

REFERENCES

INTRODUCTION

Overbreak in underground stopes is an issue prevalent to all underground mines that employ stope techniques. Overbreak is seen when more rock is broken in a stope compared to the original design shape (Germain and Hadjigeorgiou, 1997). As illustrated in Figure 1, overbreak usually occurs in waste. The overall grade of the stope is diluted thus hindering potential profits that may have been seen by that stope (Maerz, Ibarra and Franklin, 1996). In addition overbreak results in a number of other undesired consequences such as:

• ground control problems as a reduction in the module and strength of rocks are observed
• poor fragmentation
• production scheduling errors (Singh and Xavier, 2005).

Numerous researchers have highlighted the detrimental hazards that overbreak has on an underground operation. This has subsequently driven a number of studies into isolating parameters that affect overbreak and determining methods of control. While these studies acknowledge the importance of the overbreak phenomenon, only a handful has focused on predicting overbreak. According to Jang and Topal (2013), predicting overbreak would be the first step in developing an effective overbreak management and blasting reconciliation system. Overbreak prediction begins in the design stage of underground excavations. To date no set methodology exists for predicting stope overbreak with practices for predicting overbreak varying dramatically from site to site. A set method/system that could accurately predict stope overbreak has advantages including:

• increased realisation of costs and profits
• improvements in scheduling and ground control
• ability to use this method as a learning/design to effectively implement methods of control.

Due to the high number of variables that affect overbreak and the fact that no linear correlation exists between these variables, empirical methods would prove to be ineffective in predicting overbreak.
Overbreak

Overbreak, as illustrated in Figure 1, is a result of drilling and blasting to extract ore in an underground excavation. While to date drilling and blasting remains one of the most cost-effective methods for extracting ore, overbreak and the associated effects can have a detrimental effect on any underground operation. In terms of costs, overbreak that usually occurs in waste, acts to dilute the grade of a stope. Material that is below the cut-off grade cannot be mined at a profit (Maerz, Ibarra and Franklin, 1996). While Maerz, Ibarra and Franklin (1996) focused on the financial effect that overbreak has on an operation, Singh and Xavier (2005) focused on the operational consequences that overbreak causes. These include the following:

- breakdown of the inherent interlocking of the weakness planes
- ground control problems as a reduction in the moduli and strength of rocks is observed
- poor fragmentation
- production scheduling errors
- restricted access due to damaged ground for drilling and charging operations
- increased costs due to installation and maintenance of supports

These operational hazards pose a significant risk to personnel working in the vicinity. Coupled with the economic effects of overbreak, it is evident that improper management of overbreak can result in a number of consequences to an underground excavation.

Factors influencing overbreak

In a study conducted by Singh and Xavier (2005) into the causes of overbreak in underground excavations, it was discovered that there were numerous parameters that affect the extent of overbreak. In general they can be classed as:

- rock mass features
- explosive characteristics and distribution
- blast design and execution

Rock mass features also referred to as geological parameters include strength of rock mass, discontinuity characteristics and water conditions. These factors are fixed and cannot be changed. Numerous studies have identified some relationships that exist between these factors and overbreak. For instance Cunningham and Goetzsch (1990) suggested that the presence of joints affects the attenuation of the induced stress wave. Attenuation was shown to be minimal, when the angle of joint orientation is perpendicular or parallel to the face and increases to a maximum when the angle is between 15 and 45° as shown in Figure 2.

Unlike rock mass features, explosive characteristics/distribution and blast design/execution are not fixed factors and can be altered. These blasting and drilling parameters include velocity of detonation (VOD), powder factors, drilling geometry (burden and spacing), delay sequencing and hydraulic radius, etc. It was observed that during trials conducted by Singh and Xavier (2005) that explosives that have a higher VOD yield less damage around the perimeter of an excavation. This damage around the perimeter was found to directly increase the amount of overbreak. Higher VOD explosives are decoupled and yield higher shock energy and less gas energy. This is illustrated in Figure 3.

Predicting stope overbreak

Predicting overbreak is an extremely important part of the underground mining process and begins when designing the stope. Mining operations have suffered terrible financial and technical consequences from incorrect predictions of overbreak. While some of the studies discussed previously identified the relationship that exists between some variables and the influence they have on overbreak, they were unsuccessful in determining a method that could predict stope overbreak. Predicting stope overbreak has in the past been avoided as there exists a lack of understanding of the extent that certain parameters have on overbreak and the relationship that exists between these parameters is unclear (Monjezi and Denghai, 2008). To date there is no set methodology or generalised practice for predicting stope overbreak.

Stope overbreak (also referred to as the dilution) is determined during the design stage of an underground excavation. Predicting overbreak in stopes has been achieved by a collaboration of geotechnical and production engineers analysing historical data on a site-by-site basis. Therefore the methodology for predicting overbreak at one site may be significantly different to that of another site (Vecchio, May 2013, personal communication).

METHODOLOGY

The proposed approach in the analysis of the project followed a structure outlined by a number of objectives. These objectives all aimed to be able to derive a system that could accurately predict stope overbreak at Plutonic underground gold mine. After analysing different methods it was evident...
that ANNs would be one of the most suitable methods for predicting stope overbreak.

**Artificial neural networks**

ANN is a branch of artificial intelligence (AI) and is essentially an information processing system that simulates structures and functions of the human brain. ANN utilises mathematical algorithms to predict an output pattern when it recognises a certain input pattern (Monjezi and Dehghai, 2008). Original inspiration of ANN came from the early models of sensory processing by the brain (Krogh, 2008). ANN has been used to solve a range of different problems in different fields including engineering, medicine and more recently in the mining industry.

An ANN can be created simply by simulating a network of model neurons. By applying mathematical weighted algorithms that mimic the process of real neurons, the network is forced to learn (to be trained) to solve a variety of problems (Krogh, 2008). A model neuron is referred to as a threshold unit whose function is illustrated in Figure 4.

![Function of a threshold unit](image)

The neuron receives input from a number of external sources (input data), weighs each input and then finds the sum of these weighted inputs. If the weighted sum is above a certain threshold, the output of the unit is assigned a numerical value of one. Reversely if the weighted sum is below this threshold it is assigned a numerical value of zero. Therefore the output changes from zero to one when the weighted sum of inputs is equal to the threshold. Those points in the input space which satisfy this condition define a hyper-plane. Points on the lower side of the hyper-plane are classified as zero and those on the upper side are classified as one. Essentially this means that a classification problem may be able to be solved by this threshold unit if the two classes can be separated by a calculated hyper-plane (Krogh, 2008). During training, the hyper-plane moves around according to weighted data sets until it finds its correct/optimal position in the learning space. Once the correct position is found, the hyper-plane will move minimally. If the data is linearly separable, the threshold unit will be able to correctly classify input data and determine the correct output associated with the given input data. Figure 5 demonstrates a graphical example of a classification problem with a hyper-plane.

![Hyper-plane in a classification problem](image)

However many classification problems are not linearly separable and consist of complex interconnected data sets. In these situations more complex neural network with a different training mechanism must be employed. The first step when presented with a non-linear problem is to separate classes by introducing more hyper-planes. This is done by introducing multiple threshold units. This is usually done by adding in an extra (hidden) layer of threshold units each of which does a partial classification of the input and sends its output to a final layer. This sums up the partial weights to determine the final output. Such a network is called a multi-layer perception network (MLP) and the general configuration is illustrated in Figure 6. MLP networks are used predominately for non-linear separable problems. By replacing the step function with a continuous function, the neural networks produce an output which is a real number. For this reason MLP networks are the most common and widely used neural network (Roy and Singh, 2004).

An ANN can be considered as an intelligent hub of mathematical algorithms that are able to predict an output pattern when it recognises a given input pattern (Monjezi and Dehghai, 2008). As explained previously the neural network is first trained by processing input data. A large number of data sets which have been normalised is ideal as it allows for the hyper plane to be positioned to the most optimal position. After the completion of training with reliable data sets, neural networks are able to identify similarities when presented with a new data set. Subsequently the ANN can predict an output variable (Krogh, 2008). This property of ANN gives excellent interpolation capability to the technique, especially when input data isn’t exact. This is one of the main reasons why neural networks have been adopted over other empirical methods to solve complex problems. ANNs have been used as direct substitute of the following empirical techniques (Monjezi and Dehghai, 2008):

- auto correlation
- multivariate regression
- linear regression
- trigonometric
- other statistical analysis techniques.

Selecting the appropriate network and training algorithm when developing an ANN is one of the most important tasks. Users of an ANN often know what problem they wish to solve.
and from previous research and experiences, they can easily determine the input and output parameters they wish to employ. This can be done in an iterative manner when using programs such as MATLAB®, version R2013a (by MathWorks) to construct an ANN. The number of hidden layers, neurons, training algorithms and network types can all be altered in an attempt to create an ANN that can predict an output based on certain inputs accurately.

There are a number of different mining related problems that have been solved by ANN in recent times especially with the fast development of computer based software programs (Tawdrous, 2006). This is supported by the studies conducted by Roy and Singh (2004) who suggested that ANN had become a favoured method for solving mining related problems due to the complexity of these problems. Traditional empirical methods have proven to be ineffective in areas where the number of variables is high and the relationship between them is not clear (Monjezi and Rezaei, 2011). A case study involving the use of ANNs to solve a mining related problem was conducted at the Gol-E-Gohar mine in Iran. The mine was experiencing large amounts of backbreak and struggling to predict backbreak accurately. After failing to predict backbreak whilst utilising standard empirical methods, the researchers adopted a MLP neural network. The team were able to predict backbreak to at 97.94 per cent accuracy and were subsequently able to reduce backbreak from 20.0 m down to 4.0 m (Monjezi and Denghai, 2008).

**PLUTONIC GOLD MINE**

Plutonic Gold Mine is located approximately 180 km north of Meekatharra in Western Australia (Figure 7). Plutonic was first obtained by Barrick Gold Corporation at the end of 2001 following the acquisition of Homestake Mining Company. Barrick Gold is an international gold and copper mining company with operations in six continents. In recent history Plutonic was the sixth largest gold camp in Western Australia with an estimated total endowment of 12.2 Moz of gold. The Plutonic orebody is currently well into the underground stage of mining. The Plutonic underground operation utilises longhole open stoping for extraction of stopes ranging from 100 to 45 000 t in size. In 2012, Plutonic produced 112 000 oz of gold at a total cash cost of $1220/oz. Proven and probable reserves at the end of 2012 were 206 000 oz (Barrick Gold, 2013).

Stopes in the Timor region 20
Stopes in the Timor region with an ultramafic hanging wall or foot wall 30
Stopes that utilise a “dice five” hole pattern 15–20
Large bulk stopes in competent ground 10
Stopes in the Baltic region 10
Minimum dilution factor regardless of stope size, location, host rock, etc 10

It is estimated that approximately 90 per cent of dilution figures are determined using the information outlined here (Vecchio, May 2013, pers comm). Plutonic underground gold mine currently intends to investigate methods in which stope overbreak can be predicted more accurately. This particular area (stope overbreak prediction) is an area that has been ignored for a period of time as it was considered deemed ‘too hard to determine accurately’. This suggestion aligns with the knowledge gained in previous sections of this paper.

**Data collection**

The performance of ANN largely depends on the quality of data that is put into the ANN (Krogh, 2008). Accurate data would give a clear indication of whether an ANN is suitable for overbreak prediction while inaccurate data would neither confirm nor deny the suitability of an ANN. It wasn’t until the data was collected at Plutonic that an analysis on the accuracy of the data could be determined. Information was gathered from a variety of different sources including stope reconciliation forms, stope design packages and charge plans. The collected data is tabulated in order to gain a better understanding of which parameters could be used for ANN input. With nearly eight years and a total of 1100 data sets, it was evident that there were many data sets that could be inputted into the ANN. However, inconsistent record keeping practices meant that a number of parameters known to affect stope overbreak could not be used in the ANN. These include:

- any geological parameters
- burden and spacing ratios
- explosive types and relative distribution.

**Artificial neural network development**

The first step in ANN development was to choose input parameters for ANN construction. As mentioned previously some of the parameters that have known to affect stope overbreak were not available due to the inconsistent record
Mining Education Australia – Research Projects Review

Using Artificial Neural Networks to Predict Stope Overbreak at Plutonic Underground Gold Mine

Keeping practices. Input parameters were therefore selected based on their reliability and the potential effects they may have on overbreak. The input parameters along with the general ANN architecture are illustrated in Figure 8.

Following the determination of input parameters the minimum acceptable number of data sets required for network training and testing needed to be determined. A total of 300 data sets for training and testing were collected as the number of data sets for ANN construction in this research project. Three classes of ANN were constructed in order to determine the optimum ANN model and they are:

- Class 1: an ANN with all 300 data sets
- Class 2: an ANN with outliers removed, 226 remaining
- Class 3: four separate ANN based on stope sizes with outliers removed.

During the process of training each of these ANN, the network type, number of neurons, type of algorithm, number of hidden layers and error regression analysis were altered in an iterative manner to determine the optimum ANN model. This was done by utilising the ANN function in MATLAB.

RESULTS AND ANALYSIS

After training ANN models, untrained data set was used to validate the accuracy of the models. Table 2 demonstrates the number of neurons and correlation coefficients (R) values for each model. As shown in Table 2, the majority of ANN predictions showed strong R values between measured and predicted overbreak, with Class 3 (10 000 t+) producing the most accurate results. In addition, training of ANN models are designed to stop when the mean square error (MSE) is reached to 1.0E-2. MSE is the difference between predicted overbreak and measured overbreak.

The Class 2 ANN was selected as a general overbreak prediction model because it showed stronger R value than Class 1. Class 2 ANN model contains two hidden layers, 25 neurons and utilises the gradient descent method training algorithm. Of the 226 data sets used to train and test this ANN model, 70 per cent of the data was used for training while the remaining 30 per cent was used for testing and validation.

Utilising the simulating function in MATLAB, the input parameters were inputted into the ANN and a prediction for stope overbreak for each stope was derived. Once these overbreak predictions were derived, a comparison between the ANN and Plutonic predicted overbreak versus actual observed overbreak could be conducted. The two methods used to analyse these predictions were:

1. correlation coefficient between predicted and observed overbreak
2. tonnage variances between predicted and observed overbreak.

The results are summarised in Table 3.

The first analysis technique which examines R values between the predicted and observed overbreak tonnages indicates that current practices at Plutonic are marginally more accurate than the ANN (0.873 versus 0.862). On the other hand, the second technique indicates that the ANN is substantially more accurate (-273.81 t versus +14788.64 t). The R calculated gave a false representation of the ANN accuracy. This was due to the phenomenon of over-fitting. Over-fitting occurs when there are limited data sets in certain regions of the ANN, which cause the ANN to greatly over or under estimated stope overbreak tonnages. Even though the ANN predicted stope overbreak more accurately, the large errors in ANN prediction in regions where there were limited data sets caused R to be lower. This phenomenon is highlighted in Figure 9.

As can be seen in Figure 9, for stopes in the 2000 t region, the ANN is substantially more accurate. However for the 169 t stopes, the ANN is vastly inaccurate due to limited data sets. This was one of the limitations of an ANN that was highlighted by Krogh (2008) and one that was experienced in this study. However for the majority of the data sets, the ANN is much more accurate than current practices. This was demonstrated by the Class 3 ANN (1000–5000 t) where the majority of data sets are found. The ANN in this case was much more accurate than current Plutonic Practices (0.88412 versus 0.79986). This highlighted the importance on the number and quality of data sets required for ANN development as well as the effects that over-fitting can have on an ANN. Second analysis

### Table 3

<table>
<thead>
<tr>
<th>Method</th>
<th>Correlated against</th>
<th>R</th>
<th>Tonnage variance</th>
</tr>
</thead>
<tbody>
<tr>
<td>Plutonic method*</td>
<td>Observed overbreak</td>
<td>0.873</td>
<td>+14 788.64 t</td>
</tr>
<tr>
<td>Artificial neural network method</td>
<td>Observed overbreak</td>
<td>0.862</td>
<td>-273.81 t</td>
</tr>
</tbody>
</table>

*Plutonic method: conventional overbreak predicting practice in Plutonic Mine.

![FIG 8 – Artificial neural network architecture for Plutonic Gold Mine.](image)
clearly indicates that the ANN is much more accurate than Plutonic methods. As was seen in the results section, the ANN prediction over the 36-month period was only 0.27 per cent under the predicted tonnage while that for Plutonic methods was 14.4 per cent.

With the optimum ANN model producing accurate results, an assessment into the tasks and costs associated with implementation of an ANN based prediction system at Plutonic was conducted. Minimal tasks are involved with setting up the system. For a small initial investment of $10,000, the projected annual savings between $190,000 and $230,000 (from the increase in stope overbreak prediction accuracy) a year could be expected.

CONCLUSIONS

Overbreak is an unavoidable consequence of longhole stoping and can result in large financial losses to any underground operation. While many researchers in the past have focused on identifying parameters affecting stope overbreak and implementing methods of control, few have focused on predicting overbreak accurately. Predicting stope overbreak accurately would be the first step in developing an effective overbreak management and blasting reconciliation system. Empirical methods would prove to be ineffective in predicting stope overbreak due to the high number of variables that exist and the relationship between these variables is unclear. ANNs would therefore prove to be a more effective method for predicting stope overbreak as they are able to identify relationships between input variables by utilising mathematically weighted algorithms and then use them to solve for an output variable.

After developing an ANN that could predict stope overbreak a comparison between the ANN predictions and current methods employed at Plutonic was conducted. Correlation coefficient between observed and predicted overbreak were 0.862 (ANN) and 0.873 (Plutonic Overbreak Prediction Practices) while tonnage variances between predicted and observed overbreak were -273 t (ANN) versus +14 800 t (Plutonic). While correlation coefficient would indicate that the ANN was less accurate than current methods employed, further analysis indicated that the problem of over fitting had occurred and the ANN was in fact more accurate.

The results in this paper have proven that ANNs can predict stope overbreak at Plutonic Gold Mine. The ANN stope overbreak prediction system can save time for a low initial investment and produce a significant cost saving. The results indicate that the method here could be adopted at other mine sites to predict stope overbreak. Furthermore the results from this project highlight how ANN can be used to solve complex problems in the mining industry.

Recommendations

The following recommendations were derived from this project:

- Plutonic should implement an ANN-based overbreak prediction system as a tool for comparison so engineers can compare the ANN and their predictions and make an more informative estimation of stope overbreak.
- Each new stope data set should be imported back into the ANN to improve accuracy.
- Plutonic should start recollecting geological and geotechnical data and construct different ANN to improve the productivity of the mine.
- Barrick should conduct similar trials at operations, which could also benefit from predicting stope overbreak more accurately.

ACKNOWLEDGEMENTS

The authors would like to acknowledge Jeff Brown and Chris Vecchio for their assistance in providing data.

REFERENCES


A Comparison of Methods to Create the Initial Void for Longhole Open Stoping Blasting

K Lee¹ and A Halim²

ABSTRACT
This study is based on confidential case studies collected from a Western Australian gold mine, known from here on in as ‘Mine X’. The primary mining method used at Mine X is longhole retreat open stoping with paste backfill. The main objective of this research project is to investigate which method/s is the most appropriate to develop the initial void for stope blasting at Mine X.

Two methods were investigated during this research; conventional longhole rising and slashing against paste. Rising is a very common method, used frequently at Mine X as well as in the broader mining industry as determined through a literature review. Because of the common use of rising, it was expected to be a more reliable option resulting in a higher success rate. Slashing against paste, however, is a new method to the industry and not well documented, but was trialled as a possible cheaper option. These two methods were compared in terms of: success rate; drill and blast costs; limitations for their use; and benefits derived from combining the two methods.

The analysis of the case studies revealed several key points. Firstly, the success rate estimated by comparing stope design geometry and final surveyed shape showed that both methods achieved the same success rate of 78 per cent. The cost analysis determined that the average rise required more than double the costs per tonne of rock than slashing against paste. Slashing against paste was found to be a much more restrictive option, requiring new stopes to be situated next to paste filled stopes of specific dimensions. Rising does not have these restrictions; however, more often required slot drives to be developed prior to stoping than slashing against paste. Combining the two methods proved to be a beneficial option having the advantages of both methods. Overall this study recommends, for Mine X, the use of slashing against paste over rising in situations where slashing against paste is permitted. All other situations should consider the use of rising or a combination of rising and slashing against paste.

INTRODUCTION
Mine X is a gold mine located in the Goldfields region of Western Australia. The mine utilises longhole open stoping as its main mining method with the use of paste as backfill. The annual material mined at Mine X is just less than one million tonnes. Due to confidentiality reasons, the mine cannot be identified, hence, it is written as Mine X in this paper.

The orebody at Mine X changes throughout the mine; however, the focus for this research is limited to the stopes in the ‘T area’ of the mine. These single level stopes range in size from narrow vein stopes (4 m width) to bulk stopes (up to 20 m width), and up to approximately 22 m in height.

This project looked into the following problem: which method/s are the most appropriate to create the initial void for stope blasting at Mine X?

Rising
Traditionally, Mine X produced the initial void for stoping using a conventional longhole rise, followed by production rings into the rise void. A rise is a vertical or steeply inclined opening, traditionally driven in the upwards direction (Tatiya, 2005). Rising uses closely spaced drill holes and uncharged reamer holes to create void for the blastholes of the rise to fire into. This is designed using burncut techniques (Orica Technical Services, 2008) as shown in Figure 1. Based on the authors’ industrial experience, rises are the traditional way in which void space is created for stoping and this is a well-established practice, which has developed significantly over time.

Slashing against paste
Recently, the new method of slashing against paste, has been trialled to open voids instead of rises, in an attempt to reduce costs associated with drilling and blasting. Slashing against paste requires firing forward dumping production rings towards a paste filled stope. The firing of the dumped production rings propels the blasted rocks forward, compressing the paste, before falling into the drive below and creating a void (Anonymous Engineer at Mine X, November 2005).

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². MAusIMM, Lecturer, Western Australian School of Mines (WASM), Curtin University, Locked Bag 30, Kalgoorlie WA 6433. Email: a.halim@curtin.edu.au
2012, pers comm). Figure 2 shows how a slashing against paste design used at Mine X looks.

Whilst slashing against paste has been practiced in several mines around the Western Australian Goldfields region, there are no literatures that could be found about this method, other than personal communication with engineers who have used the method.

**Objectives**

To compare the aforementioned two methods, the following objectives were investigated:

- How rises are used as the current method?
- How has slashing against paste been used?
- How do both methods perform compared to their design?
- How do the drill and blast costs associate with each method compare?
- In what specific situation each method can be used?
- What benefits can be achieved from combining the two methods?

**Methodology**

To complete this project, several steps were taken as follows:

- data collection from Mine X
- comparison between blast designs and results
- determine which factors affected success
- calculate costs
- compare limitations
- analyse combining methods in one stope.

**Data collected**

The collected data was for all stopes completed in the ‘T area’ of the mine during January 2012 and July 2013. This included
A COMPARISON OF METHODS TO CREATE THE INITIAL VOID FOR LONGHOLE OPEN STOPING BLASTING

27 rise designs and nine slashing against paste designs. The data required were stope notes, drill plans and charge plans, stope reconciliations, stopes design files, paste data and drilling and explosives costs.

**Compare design to results**

To determine whether both methods have worked well at Mine X and which method has worked better, the success rate for each method was calculated. This was determined by comparing the stope shape designed in Surpac™ and the cavity monitoring system (CMS) survey taken after stope extraction. As all stopes will have some amount of over- or underbreak, stopes were classified as ‘successful’ if the level of over- and underbreak experienced did not significantly contribute to underperformance for the rest of the stope. If there was a significant amount of over- or underbreak, the stope was classified as ‘unsuccessful’.

**Factors affecting success**

A variety of factors can contribute to the success of a rise or slashing against paste design. To determine which factors most impact the results a series of analyses were carried out comparing the successful and unsuccessful designs. These considered both engineering design variances as well as operational inaccuracies where possible.

The following factors were considered:
- drill hole deviation
- incorrect charging
- length of drill hole design
- angle of inclination of drill holes
- width and height of stope
- drill hole and reamer diameters
- drill rig used
- paste properties for slashing against paste.

**Cost analysis**

The drill and blast costs for the all the stopes can be calculated to determine which method is the cheapest option. The costs associated for drilling and blasting activities were obtained from Mine X and the total cost for each rise and slashing against paste design was calculated.

To allow easier comparison between different sized stopes, the tonnes of rock designed to break for each design were obtained from Surpac™ and the cost per tonne of rock calculated for both drilling and blasting.

**Limitations**

This section aimed to determine in which situations each of the methods can be used based on the case studies analysed from Mine X. It also considered whether slot drives were used and in which future stopes they are likely to be necessary for each method.

**Combination of methods**

From a previous trial, it was recognised that both slashing against paste and a downhole rise were used to open a particularly tall stope. This analysis aimed to determine what benefits were derived by combining the two methods in the one stope. The possible advantages and disadvantages were explored by following this particular stope through the previous analyses; overall success of the stope, factors affecting this success, cost analysis and limitations to the design.

**ANALYSIS AND RESULTS**

**Comparing results to design**

Whilst the rise method has been used three times more than the slashing against paste method during the trial period, the success rate for both methods is the same at 78 per cent. This was calculated from the percentage of ‘successful’ stopes during the trial period. These results suggest that both methods can be used in the right situation. It is especially interesting to note, that the slashing against method has been very successful given it has only been used for such a short time, whilst the rise method is a practiced method, used for many years in the ‘T area’, yet still has the same success rate in the time frame sampled.

**Factors affecting success**

Analysis was performed into which engineering design factors most affected the success of both rises and slashing against paste in the case studies. A number of other operational factors were briefly considered, such as drilling accuracy and charging accuracy, however, these were difficult to detect and were not considered in the analysis.

The main reasons for unsuccessful rises in this study appear to be flatter inclination of the rise, long lengths drilled with a smaller drill rig, smaller diameter reamers and the geology in particular areas. These may possibly be improved by drilling more vertical rises, using the larger drill rig with larger diameter reamers for longer rises and trying new ways to open some of the ‘trickier’ areas.

For slashing against paste, there was no real correlation between the factors analysed and the success of the stopes. This is most likely due to the large variety of drill designs and differences between designs, as the method is still being trialled and improved. Also less data was available to see any clear trends. With more trials it may be possible to gain a better understanding of which factors influence the success of slashing against paste the most.

**Cost analysis**

The drill and blast costs for the all the stopes were calculated and compared between the different methods.

For drilling costs in cost per metre of drill hole ($/m) was obtained from Mine X. It was assumed for the calculations that the cost of drilling holes was the same for both drill rigs used on-site. It was also assumed that the cost of drilling reamer holes was the same as drilling a standard blasthole two or three times, depending on the number of passes taken to ream the pilot hole. The drilling cost for the rise or slashing against paste design was determined based on the number of drill metres in the drill plans issued to the longhole drillers and the cost per tonne was then calculated for comparison. The following steps were followed to achieve this:
- total drill metres for each design obtained from engineering drill plans
- cost per design calculated (cost per design = drill metres × cost per metre)
- total tonnes per design determined from Surpac™
- cost per tonne calculated (cost per tonne = cost per design / tonnes).

From the drill and blast cost results per tonne of rock for each method shown in Figure 3 it can be seen that the cost per tonne for drilling rises is much higher than the cost per tonne for drilling slashing against paste designs. This is mostly due
to rise designs containing more drill holes and the additional lengths required to drill reamer holes.

For blasting costs, only the cost of primers and detonators were considered in this analysis. These are the most substantial cost for explosives, and the other costs for operators, ANFO, detonator cord, etc., would be very similar for both methods. Two main types of detonators are used at Mine X; programmable electronic delay detonators (iKon) and non-electric delay detonators (millisecond/long period (MS/LP)). The cost of these different detonators is considerably different; with iKon detonators costing almost double that of MS/LP detonators.

To calculate the blasting cost, the number of detonators required for each design was obtained on the charge plan issued to the charge-up crew. From here, the cost per design was calculated. To allow comparison between different sized stopes, the cost per tonne of rock was calculated for each design. Similar steps were taken as for calculating drilling costs:

- total number of primers and detonators for each design obtained from engineering charge plans
- type of detonators obtained from charge plans
- cost per design calculated (cost per design = number of detonators × cost per detonator)
- total tonnes per design determined from Surpac™
- cost per tonne calculated (cost per tonne = cost per design / tonnes).

The average cost per rise design and slashing against paste design per tonne of rock is shown in Figure 3. Again, rising is a much more expensive option. This is predominantly due to the preference for the more expensive iKon detonators used in most rise designs. Rises require very accurate and slow timings to fire correctly as enough time must be allowed for the blasted rock from one blasthole to fall out of the rise before the next hole fires. To accommodate for this, the iKon detonators are usually used to allow for both time and flexibility in the timings. Slashing against paste however, is much faster and does not require as much accuracy and therefore the cheaper MS/LP detonators are sufficient.

**Limitations**

Two main limitations were considered for this analysis, the position of surrounding stopes and access drives compared to the new stope; and the requirement for slot drives.

Rising used at Mine X is very flexible in its use as most stopes in the ‘T area’ of the mine are paste filled. This means stoping next to a previous stope can be very similar to stoping next to a pillar. Also a variety of rises are used; uphole, downhole and blind rises depending on the position of access drives around the stope.

Slashing against paste, however, is limited in its use by its requirement to slash against a previously paste filled stope. The paste filled stope also determines the size of the new stope, as you cannot slash against a smaller or narrower stope. This would not provide void space from compressed paste next to some part of the stope, causing potential underbreak and freezing of the stope. Thus, it is necessary to stope using slashing against paste when the right sized paste filled stope is available.

Slot drives are used with rising to create a slot that spans the width of the stope in wide stopes. It can also be used in flat stopes where a rise positioned along the footwall of the stope would be placed on too shallow of an angle. An example stope where a slot drive was required to increase the angle of the rise is shown in Figure 4. Slot drives incorporate additional costs to the stope development, through the cost of developing an additional drive of the main development as well, as increased development time before the stope can be extracted. In certain situations, by moving the position of the rise, the rise is now developed half in waste and half in ore. This adds dilution to the stope once it is fired which can reduce the economic value of the stope.

For slashing against paste, the main constraint for width is the width of the adjacent paste filled stope. Therefore, if the paste filled stope is wide enough, a slot drive should not be required for the new stope. This reduces the costs associated with slot dives described previously.

**Combination of methods**

From the previous analysis carried out, it can be seen that whilst rising is the more flexible method, slashing against paste is a much cheaper option. In an effort to gain the benefits of both methods, a stope where both slashing against paste and rising were used to create the initial void was examined. The design for this stope is shown in Figure 5.

If only one method was used for this stope, slashing against paste would not be possible due to the small size of the adjacent paste filled stope, so the only other method which could be used would a rise of at least 24 m in length. This would require very accurate drilling and charging practices to successfully blast such a long rise and, based on the previous rises, has a lower chance of success. By incorporating both methods, only a 12 m downhole rise was required, with a much better chance of success. Also, the cost was greatly reduced by combining the two methods. For this design it was found that the cost per tonne for drilling and blasting was much less than the average rise cost and even less than the average slashing against paste cost in this case.

Whilst this analysis has provided some very good results and benefits, the situations where it would be best considered to combine the two methods was outside the scope of this study and should be considered in further studies.

**CONCLUSIONS AND RECOMMENDATIONS**

**Conclusions**

From the analyses and results found during this study, the following conclusions were made:

- both the rising and slashing against paste methods have worked equally well in the ‘T area’ of Mine X
- several factors such as length, angle of inclination, drill rig and reamer size selected greatly affects the success rate of rising, however, cannot always be easily chosen
the analysis into possible factors influencing the success of slashing against paste was found to be inconclusive due to lack of data and any clear correlation in results

- the cost for drilling and blasting per tonne of fired rock for slashing against paste is less than half the cost of rising
- from the limitations, rising is very flexible however, can require additional development, whereas slashing against paste is restricted by the adjacent paste filled stope, but less often should require slot drives
- combining the two methods can be a beneficial exercise, which can gain the advantages of both methods.

Recommendations

For Mine X, the better method depends on the circumstances of the stope position. If slashing against paste can be used, it is recommended that it should be used to gain the benefits of cost and possibly less development. In all other situations, rising or a combination of both methods should be considered.

It is recommended that further study into which stopes would benefit most from using the combination of the two methods rather than a single method.

It is also recommended that with further usage of slashing against paste, the data collected from these stopes are analysed in an aim to determine the factors which most influence success.

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Mine X, 2011. Stope Note T870 Nth C.
INTRODUCTION
In recent times, sweeping changes have occurred with regards to the employment of Indigenous Australians. External forces driving corporate social responsibility initiatives have seen changes in public, governmental and private sector opinions on the role of Indigenous Australians. As such, many mining companies now offer specific indigenous training and employment programs that have been successful in placing Aboriginal people in a variety of mining roles. There have also been leaps and bounds made in the area of indigenous owned and operated companies, particularly contractors, who have secured work in the minerals industry.

This study analyses the indigenous employment policies and trends of three large mining companies and two mining contractors. On the basis of the reported quantitative and qualitative data, a maturity scale is advanced for indigenous employment practices. The study is of relevance to industry as the indigenous population of Australia is growing steadily, particularly in remote and regional areas where many mines are located.

The major benefits for industry are threefold: gaining information and data on their programs for internal usage in developing indigenous employment programs; increased social licence to operate as a result of corporate social action; and economic benefits of employing local staff in areas of the mine rather than fly-in fly-out (FIFO) / drive-in drive-out (DIDO) workforces. A maturity scale ranking companies who have taken part in this research has been compiled based upon data collected.

CURRENT STATE
Indigenous Australians
The indigenous peoples of Australia comprising Aboriginal and Torres Strait Islander (ATSI) people are the oldest continuing culture on earth and have lived in Australia for over 40 000 years (Albrechtsen, 2011). Prior to European colonisation they were the sole inhabitants of Australia and as such have a rich history. Mining has had a tumultuous relationship with Indigenous Australia throughout its history as described by prominent indigenous leader Professor Marcia Langton, as over the past 50 years we have seen ‘the transformation of the mining industry from the pillagers into the major investors in the indigenous world’ (Langton, 2013).

In 2006 the Australian Bureau of Statistics (ABS) census found that approximately 455,028 people, or 2.3 per cent of the population, identified as being of ATSI descent. This figure is, however, revised upwards to 2.5 per cent as a result of undercount and unknown indigenous status (ABS, 2006). The
most recent 2011 census found that this figure had increased to 669,736 or three per cent indicating a steady rise (albeit not as fast as non-indigenous population growth) over the five-year period (ABS, 2011). Figure 1 shows the indigenous population count from 1986–2006. The final statistics for the 2001–2013 period were expected to be released in mid-2013 by the ABS.

Employment data
Current figures for indigenous employment in the mining industry do not exist as an aggregated total. Figures for specific companies are generally published in company sustainability reports. In 2000–2001 an Australian Bureau of Agriculture and Resources Economics (ABARE) study was commissioned specifically to conduct a survey on indigenous people in mining. Unfortunately it has not since been updated.

A research paper published by the Centre for Aboriginal Economic Policy Research stated that in 2011 Non-CDEP (community development employment projects) indigenous employment in remote areas was 27 per cent. However, this is across all industries. Mining employment itself changed from employing 3300 indigenous people in 2006 to 7500 in 2011, a rise which has been largely associated with the mining boom (Gray, Hunter and Howlett, 2013).

Community and mine locations
As of 2008 90 per cent of indigenous peoples lived regionally or remotely in Australia as shown in Figure 2 (ABS, 2008). Analysis of the location of mineral tenements in Australia show that substantial overlap exists with the locations of indigenous populations. Thus excellent opportunities exist for indigenous employment in mining.

There has been little study into the reasons for disparity in employment levels given geographical proximity, however, socio-economic disadvantage as well as a lack of corporate policy are believed to be the primary drivers.

Rio Tinto corporate policy
Rio Tinto has managed to raise its indigenous workforce percentage from less than half a per cent in 1995 to just over eight per cent in 2008 (Rio Tinto, n.d.) This success is primarily attributed (by Rio Tinto) to:

- family and community support programs
- prevocational training and development
- mentoring of indigenous employees
- cross-cultural education programs.

A specific case demonstrating Rio Tinto policy is highlighted by a study of the Argyle Diamond Mine located in the East Kimberley. The 2005 Indigenous Land Use Agreement (ILUA) stipulated that not only would Argyle be required to invest financially in the local area but that the indigenous community would also have a greater say in employment (Foley, 2011).

In 2011 there were over 50 indigenous apprentices and trainees employed at Argyle. Indigenous employees comprised 25 per cent of the total workforce of which 70 per cent live in the local area. Ten per cent of the frontline leadership positions at the mine were indigenous with a long-term goal for Rio Tinto Argyle to transition more indigenous people into higher level management (Foley, 2011).

BHP Billiton corporate policy
BHP Billiton (BHPB) released their Reconciliation Action Plan in 2008. It concentrated on three main objectives:

1. promoting cultural awareness
2. education, training and employment
3. promoting indigenous business opportunities.

BHPB set both medium- and long-term targets for the employment of indigenous people, basing their criteria on the locality of prospective employees, unemployment levels and operational requirements. Indigenous business opportunity is focused around the increased capacity of indigenous people to engage in the economy (BHP Billiton, 2008) and does not necessarily include employment directly within the minerals industry.

Indigenous Land Use Agreements
As seen at Argyle Diamond Mine ILUAs make a significant contribution to the employment and land use arrangements of indigenous peoples. The input of ILUAs between various State Governments and industry aims to have an additional 3000 people employed by 2020 (BHP Billiton, 2009). As of March 2012, 588 ILUAs have been registered with the National Native Title Tribunal (NNTT) (Boyer Lectures, 2012). ILUAs are very broad in scope, however, they typically involve employment and training and protocols for future development as key aspects (NNTT, 2008).

METHODOLOGY

Interviews
Industry interviews were the primary method of data collection for this research, allowing the authors to obtain one-on-one access to industry. Interviews allow experts to get their knowledge across in a way that is not possible through other methods, particularly because interviews allow a deeper insight into how people think and reflect (Folkestad, 2008).
The data collected from the interviews has been split into two categories: quantitative and qualitative data.

Quantitative data involves information such as indigenous employee numbers, dollar contract values, etc. This data was very important as it formed the basis of many of the key performance indicators (KPIs) used in developing the maturity scale.

Perspective and relevance are highly important when dealing with statistics. Quantitative data does not give a full understanding of each company’s indigenous employment maturity. Qualitative data gathered throughout the interview process enables the quantitative data to be put into perspective for comparison against other companies. Qualitative data also gives a chance for company representatives to discuss their feelings, experiences and opinions with regards to this topic. This is highly useful as indigenous issues can sometimes be polarising with more emotive responses from interviewees eliciting information that would not be able to be gathered from the quantitative data.

Maturity scale criterion

Using examples of maturity scales developed for other strategic partnerships (for example Lendrum, 2000), a scale and criterion was formulated for indigenous employment. Maturity levels are assessed considering four categories:

1. regressive (no strategy at all)
2. reactive (adapts to fit government legislation)
3. proactive (actively engages and is ahead of legislative change, positive outcomes achieved)
4. strategic (companies are regarded as best practice in industry, partnerships and alliances are created at an individual level as well as strategically with other companies, contractors and government).

Each of these levels is measured around a set of KPIs as well as analysis of the qualitative data presented by each company. The KPIs are:

- indigenous employment rate
- indigenous frameworks for recruitment, training and human resources
- indigenous contractor procurement
- planning for increased indigenous involvement
- specific ATSI policies in place.

DATA

Rio Tinto Iron Ore

Rio Tinto Iron Ore (RTIO) is a global iron ore producer, the second largest iron ore trader and part of one of the world’s largest mining companies. Australian operations are located primarily in the Pilbara region of Western Australia with 14 mines in operation. RTIO has multiple ATSI employment and training programs and has steadily increased its indigenous employee percentages from 2007. Table 1 shows the quantitative data gathered from RTIO.

RTIO also have taken part in a number of indigenous scholarships, cadetships, apprentices and trainees:

- 12 RTIO scholarships
- 18 RTIO cadets
- 49 RTIO apprentices
- 196 RTIO trainees (including indigenous participation program).

<table>
<thead>
<tr>
<th>Total employees</th>
<th>Total ATSI employees</th>
<th>ATSI percentage of workforce</th>
<th>Monthly retention rate (%)</th>
<th>Contract value ($ M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>13 309</td>
<td>1070</td>
<td>8.04</td>
<td>Not reported</td>
<td>950 (2012)</td>
</tr>
<tr>
<td></td>
<td>+ 519 (contractors)</td>
<td></td>
<td>615 (2013 YTD)</td>
<td></td>
</tr>
</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.

In addition, in 2012 RTIO spent $4.85 M with 14 community partners and a number of charitable and community organisations.

Downer EDI Mining

Downer EDI Mining (DEDIM) is one of Australia’s largest mining contractors employing thousands of people in a variety of commodities and mining types across the country at both site and corporate level. DEDIM has a tailored Aboriginal and Torres Strait Islander Policy that sets out ‘high-level business commitments to achieving improved economic participation for Indigenous Australians’ (Downer EDI Mining, 2013). DEDIM was also the Indigenous Employment Award winner of the 2012 Australian Human Resources Institute Diversity Award. The award recognises the specialist recruitment and training methodology undertaken by DEDIM to recruit indigenous employees.

DEDIM also has a number of indigenous metrics that are measured against company progress each year and against set targets. These include: female indigenous employees showing a 42 per cent increase in the 2012 financial year; and total indigenous employment exceeding the target by 18 per cent. Table 2 shows the quantitative data gathered from DEDIM.

MMG

Headquartered in Melbourne, Australia, MMG is a mid-tier global mining company with operations spanning multiple continents and commodities. MMG boasts significant indigenous employment in its Australian operations including Century and Dugald River mines. Century Mine in particular is located adjacent to the community of Doomadgee and MMG has been able to employ a significant amount of the local populace. Table 3 shows the quantitative data gathered from MMG.

Glencore

Glencore is one of the world’s largest global diversified natural resource companies. Its Australian operations include coal, copper, cotton, grain and oilseeds, nickel, zinc and technology, with mines regionally located, some very near local indigenous communities. For the purposes of this
These open days, which was attributed partly to social norms. Only about one third of eligible students attended traineeships and cadetships) for local students in the Pilbara region. RTIO noted that they had run specifically encourage young indigenous people to take up roles in the mining industry. RTIO noted that they had run open-days and mine site tours (separate to the structured traineeships and cadetships) for local students in the Pilbara region. Only about one third of eligible students attended these open days, which was attributed partly to social norms.

### Results

#### Industry Trends

#### Education and Training

All companies interviewed (with the exception of MSCU) identified education and training as key to increasing indigenous employment in the Australian minerals industry. Strong programs and policies were identified in companies that ranked high on the maturity scale indicating the importance of these programs. In particular, highly ranked companies invest in education programs that begin whilst prospective employees are still in high school. Glencore in particular have had success with high school programs designed to specifically encourage young indigenous people to take up roles in the mining industry. RTIO noted that they had run open-days and mine site tours (separate to the structured traineeships and cadetships) for local students in the Pilbara region. Only about one third of eligible students attended these open days, which was attributed partly to social norms.

### Table 3

<table>
<thead>
<tr>
<th>Total employees</th>
<th>Total ATSI employees</th>
<th>ATSI percentage of workforce</th>
<th>Monthly retention rate (%)</th>
<th>Contract value ($ M)</th>
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<td>2789</td>
<td>266</td>
<td>9.55</td>
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<tr>
<td>1789</td>
<td>196</td>
<td>10.96</td>
<td>Not reported</td>
<td>Not reported</td>
</tr>
</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.

Glencore supplied data and statistics representative of its McArthur River Mine (MRM) in the Northern Territory. Table 4 shows the quantitative data gathered from Glencore.

### Mining Services Company (Unnamed)

Under the condition of anonymity Mining Services Company Unnamed (MSCU) was interviewed for this report. They are a Brisbane based small-scale mining services contractor that perform work across Australia and overseas. As per the scope of this report, all data recorded only pertains to MSCU’s Australian operations. Table 5 shows the quantitative data gathered from MSCU.

### Table 4

<table>
<thead>
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<th>Total employees</th>
<th>Total ATSI employees</th>
<th>ATSI percentage of workforce</th>
<th>Yearly retention rate (%)</th>
<th>Contract value ($ M)</th>
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<td>351</td>
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<td>18</td>
<td>72</td>
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ATSI – Aboriginal and Torres Strait Islander.

### Table 5

<table>
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<th>Total employees (‘000s)</th>
<th>Total ATSI employees (‘000s)</th>
<th>ATSI percentage of workforce</th>
<th>Monthly retention rate (%)</th>
<th>Contract value ($ M)</th>
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<td>0</td>
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</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.

### Traditional Ownership

Indigenous owned mines and indigenous owner-operator enterprises are beginning to emerge. This can be seen as a maturation of the industry as a whole, moving a step beyond indigenous employment and indigenous contractors. This is not to downplay the developments undertaken by existing mining companies in the hiring of ATSI peoples.

### Housing and Accommodation

Housing and accommodation requirements for indigenous employees in remote areas was mentioned by both RTIO and Glencore during interviews as a current challenge facing the industry. Typically the local indigenous communities are based in townships that have a high level of government supported housing. When the local population gain mining employment, their wage rates make them ineligible for housing assistance and they can be forced out of their homes. This presents obvious problems with family arrangements as indigenous people often live with extended family. It also causes financial problems as they are forced to rent or buy properties that are often exorbitantly expensive or not situated on traditional land.

### Changes Flow Downstream

This trend was identified by MSCU and corroborated by other companies interviewed. Social change is often the product of industry leaders such as RTIO, BHPB, Glencore, MMG or DEDIM beginning changes that later flow into smaller companies. RTIO identified this as happening in the Pilbara where smaller subcontractors are beginning to seek and employ ATSI employees of their own volition.

### Company Rankings

Analysis of all companies current practices along with a comparison to the literature (Lendrum, 2000) returned the following ranking:

- RTIO – strategic
- DEDIM – strategic
- Glencore – proactive
- MMG – strategic
- MSCU – reactive.

### Indigenous Employment Maturity Scale

Figure 3 shows the indigenous employment maturity scale developed as the culmination of the work undertaken for this research.

### Conclusions

This paper on the research topic of indigenous employment in the Australian minerals industry has been completed with quantitative data from two primary sources: the ABS and records of various companies whose progress were examined. With the data gathered through these methods it was possible for an indigenous employment maturity scale to be developed showing the ranking of the companies who participated relative to each other and relative to industry progress.

This report would not have been possible without the generous support provided by the companies who took place. However, for future work the following recommendations are advanced:

- the scope be expanded to include interviews and data from a greater number of companies to give a greater picture of the current situation.
• the companies that participated in the study be approached to include company wide data (only three of five company participants provided the data for all sites and operations)
• given further time the companies that participated in this research be re-examined in the future to gauge ongoing progress and track results over time.

It was found that two of five companies examined were sufficiently mature to qualify for the ‘strategic’ maturity category with two more qualifying for the ‘proactive’ level. This is a positive sign for the mining industry and can be seen as an indicator of where the industry is heading as a whole in the future.

ACKNOWLEDGEMENTS

We would like to thank the following people for their time and support in assisting with data collection: from Rio Tinto Iron Ore – Alison Smith and Lynette Upton; from MMG – Meg Frisby and Rodney Francisco; from Glencore – Joanne Pafumi; and from Downer EDI Mining – Bevan Whitby.

REFERENCES


Comparison of Two Excavations Systems for the Mining of Lunar Regolith

M T Lucas¹ and P C Hagan²

ABSTRACT

A study was undertaken to compare the feasibility of conventional pick cutting systems and pneumatic excavation systems in an off-world environment. This involved a technical and economic analysis of these methods under certain lunar conditions including a determination of the cuttability of simulate lunar regolith in order to assess the excavation effectiveness of conventional mining machines.

The technical and economic feasibility was determined by analysing different lunar conditions and the possible effects that these conditions would have on the excavation methods. These theoretical analyses were supported by fundamental mining knowledge and current literature on the subject matter.

The estimation of cuttability was based on a newly designed, modified version of the Newcastle-upon-Tyne cuttability test that has greater sensitivity at low force levels. The sample tested was the Australian Lunar Regolith Simulant-1 (ALRS-1). The cuttability forces and abrasivity was assessed across a range of packing densities with the results used to perform a sensitivity analysis on the efficiency of pick cutting methods in excavating soil.

While more research and development is necessary to be undertaken on the prototype pneumatic excavation system, it was found that both excavation systems have the potential to remove lunar regolith across a range of densities. It was also concluded that while neither system was economically feasible for lunar excavation based on current technology, the pneumatic excavation system was shown to be more advantageous under the conditions analysed.

INTRODUCTION

While mankind has long had an interest in space exploration, it has only been in recent times that it has had the capacity to achieve this. Ever since the Apollo missions in the 1960s and 1970s there has been an interest in the creation of a lunar settlement on the moon. With this comes a need for resource extraction to support the settlement that, with today’s advances in technology, is steadily becoming a feasible option. This situation gives weight to the objectives of this study that was to analyse and evaluate the feasibility of conventional and pneumatic mining methods particularly with respect to the excavation of the lunar surface material or regolith.

Project objectives

The project entailed both a theoretical analysis and experimentation. Testing the efficiency of conventional mining systems on lunar regolith requires a determination of the cuttability of the lunar material. The tests made use of a fine-grained simulant lunar regolith, ALRS-1, and a newly designed cuttability machine. It also involved evaluation of the effectiveness of each rock breakage technique against this simulant.

In addition, a theoretical analysis of each system in a lunar environment was undertaken, focussing on seven specific conditions, which is likely to influence the system efficiency, these being: gravity, vacuum environment, soil abrasivity, electrostatic soil, soil compaction, dust suppression and transport costs.

The study evaluated the two excavation systems, and recommends the most feasible technique. It also outlines some of the issues with the use of each system and potential concerns that need to be addressed.

COMPARISON OF EXCAVATION SYSTEMS

Whilst the excavation efficiency of each method is important in analysing system effectiveness, there are also a number of lunar conditions that influence the feasibility of each system.

Gravity

Gravity is a significant condition that changes between a terrestrial and lunar environment, with one-sixth of Earth’s gravity on the lunar surface.

Conventional pick mining methods rely on the weight of the machine to provide the sumping force for material cutting. As the equation shows, the reduction in gravity reduces the effectiveness of the machine as its sumping force is reduced.

\[
\text{sumping force} = \text{mass} \times \text{gravity} \times \text{coeff friction}
\]

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However, pneumatic excavation systems are not influenced by gravity and therefore there is no known effect of this on the machines efficiency in a reduced gravitational environment.

Vacuum environment
The main issue that the vacuum environment poses regarding use of conventional mining machines is the likely increase in surface to surface abrasive friction between the picks and regolith increasing the rate of pick wear. In an environment lacking an atmosphere, the internal angle of friction and strength of soils is increased. It has been claimed that an increase in friction would increase the cutting forces as well as increasing pick wear (Prado, 2013; Podnieks and Roepke, 1985). The US Bureau of Mines used experimental data to determine that the vacuum environment increased friction by between 1.5 and 60 fold compared to the friction experienced with terrestrial pick cutting (Prado, 2013; Podnieks and Roepke, 1985).

Additionally, the lack of an atmosphere in a lunar environment would adversely affect the rate of heat dissipation. As heat cannot be dissipated at the same rate as on earth, the cutting picks would quickly build up heat and hereby reducing the relative difference in hardness between the cutting tool and lunar material.

This increase in friction would not only reduce the efficiency of the pick cutting, but also costs of cutting with much higher pick consumption.

Pneumatic excavation systems would require the use of a to excavate the regolith. In a vacuum environment, these systems could only operate in a self-contained atmosphere. The system would require creation of this atmosphere sealing off the excavation hole through the use of an expanding ‘packer’ as well as grout (Bernold, 2013). If an efficient seal cannot be made, this self-contained atmosphere could not be created, as gas could constantly escape, which would make this excavation system partially redundant.

Regolith abrasivity
The main issue with abrasion for conventional mining methods is pick wear. A significant issue in relation to abrasivity of the regolith is lubrication. In a lunar environment, due to the vacuum nature, standard wet lubrication cannot be used (Honeybee Robotics, 2013). Instead, another means such as use of a dry or recyclable lubricant would be required.

There are also several possible issues that the abrasivity of lunar regolith may have on the pneumatic excavation system. The first being it may cause wear on the inside of the suction piping. While the quantitative effect of this is not known, it will lead to faster wear that if not planned for will eventually create holes diminishing the effectiveness of the system, as the pneumatic gas would just escape from the closed system. More regular replacement of the piping would be required then similar systems used on earth. Another potential concern is damage to the inflatable packers that are used to seal the hole. The soil abrasion will wear or cut the rubber packer, which will also require more regular replacement (Geopro, 2013).

Electrostatic regolith
The lunar regolith has electrostatic properties that would affect conventional mining machines, as it would tend to coat the picks and exposed moving parts, increasing component wear. It would also increase thermal adsorption, due to the dark material, which was observed on previous lunar equipment, which continually overheated (Bell and Phillips, 2005).

The pneumatic excavation system could also be influenced by the electrostatic properties, as the piping could be coated reducing the suction and hence efficiency of the system. The heat issues described previously would also apply to this equipment.

Regolith compaction
Whilst compaction of the regolith is likely to have a greater effect on excavation efficiency, it also influences the feasibility of these operations.

One issue that would affect both systems is a loss of traction over loose soil, which was observed during the Apollo 15 mission (Beale, 2009). Additionally, this would affect the coefficient of friction for conventional machines, further reducing the sumping force and excavation efficiency.

Also, soil compaction would influence the efficiency of pneumatic excavation systems, as blasting may be required in denser regolith to allow the system to operate. This would further increase the transportation costs and reduce its economic feasibility.

Dust suppression
Dust suppression is a significant concern, as the lower gravity and vacuum environment cause much slower dust settlement.

Conventional mining methods often create a large amount of dust during excavation. As water is not readily available on the moon, and continual supply from the earth would not be practical, alternate means of dust suppression would be required.

As the pneumatic excavation system operates within an enclosed atmosphere, no regolith is expelled from the excavation and no dust suppression would be needed, making this system more beneficial in regards to dust suppression.

Transportation costs
Transport costs are a significant factor in the feasibility of both operations. Mackey, Gaskins and Lally (1996) claimed that the transportation of material to the moon is likely to be between $54 000 and $121 000/kg of payload. Hence both operations would have a significant initial capital outlay. As pneumatic machines do not rely on mass for efficient operation, the mass of these machines could be minimised through the use of lightweight materials or other design methods to reduce transportation costs. Conventional machines are required to make a compromise between cutting efficiency and transport cost, as the reduced mass reduces the sumping force of the machine, but would also reduce the transport cost.

Both machines would require replacement parts for maintenance, which would be influenced by the operational reliability of each system. Additionally, conventional mining systems would require replacement picks, which would be a significant cost due to the higher pick wear, as well as a possible continual supply of material for dust suppression, if no other option is determined.

Pneumatic excavation systems may require replacement compressed gas, depending on the effectiveness of gas recycling in the closed-circuit system, as well as grout and replacement funnel heads.
EXPERIMENTAL APPARATUS

Requirement for new test facility
Due to the nature of ALRS-1 lunar regolith being an unconsolidated material, it is likely to have low cutting forces relative to rocks. An established test procedure developed to assess the cuttability of rocks is the Newcastle-upon-Tyne test (Roxborough, 1987). Its use in this case was limited, as it did not have sufficient sensitivity to measure the expected low forces in cutting the lunar regolith hence an alternate approach was required.

Apparatus
The experimental apparatus designed and constructed for the tests is shown in Figure 1. Essentially it is a scaled-down version of the Newcastle facility. The apparatus comprises a pneumatically driven arm to provide the motive force to the cutting tool mounted on a steel support frame, and associated electrical components. Attached to the front of the piston is a detachable cutting head with force transducers.

Lunar regolith sample preparation
The Australian Lunar Regolith Simulant, also known as ALRS-1 or ALS-1, was developed by Professor Leonhard Bernold at the UNSW Department of Civil and Environmental Engineering comprised mainly of basaltic soils obtained from the northern region of Sydney, Australia (Garnock and Bernold, 2012). The basalt material was sieved so that the resultant particle size distribution was in accordance with actual lunar soil samples.

Sample preparation involved water being added to the regolith before being compacted into a cylindrical shape. Zeng et al (2010) performed the Proctor compaction test on a similar lunar regolith simulant, JSC-1A, and determined the optimum moisture content for compaction was around 13.5 per cent. This was used as a baseline for estimating the amount of water to add to each sample for compaction.

Compaction was achieved using a ram and cylindrical mould with sufficient material added to the cylinder equivalent to a third of the height each time. This process was repeated until the mould was filled to the required level. The sample was then be left to dry in a drying oven at 500°C for up to three hours. After drying, each sample was weighed and the dimensions measured to determine the sample density.

EXPERIMENTAL PROGRAM

Testing methodology

Sample density tests
The apparatus was operated in a similar method to the Newcastle-upon-Tyne cuttability facility, with the cutting head driven parallel to the axis of the sample at the required depth as shown in Figure 1. The cutting and normal forces were measured using force transducers. After each test, the sample was rotated 90° and the test repeated with up to four test replications undertaken with each sample. Each test was performed using a 10 mm wide cutting head at a cutting depth of 5 mm. Tests were conducted on regolith samples with densities varying between 1.6 and 2 t/m³.

Cutting depth tests
In this series of tests, the effect of cutting depth in cutting was examined. Initially tests at three depths of 3, 5 and 10 mm were to be undertaken. It was later found that the 3 mm tests had to be eliminated due to the load cell interfering with the sample during cutting.

Abrasivity
After each regolith sample was tested, the cutting head was removed and weighed. The difference in mass was used to determine the level of abrasivity of the material at different densities, in units of milligrams of wear per metre of cutting.

Results

Cutting forces
Figure 2 shows the variation in forces with regolith density. All tests were conducted a constant depth of 5 mm with three test replications at each level of density. Over the range of conditions considered, there is a linear increase in cutting and normal force with density.

Cutting head depth
The results for the cutting head depth tests are summarised in Table 1. These results showed that the forces of the soil increased as expected, and that the specific cutting forces, given as the amount of force per millimetre depth of cut, were as theoretically calculated.

Regolith abrasion
The results from the soil abrasion measurements are shown in Table 2. These results indicate that the soil has minimal wear on the pick which in this study was not tungsten carbide as used in the standard cutting test but made of aluminium, a much softer metal that would exaggerate the wear rate. However, this wear is likely to be higher under lunar conditions, as discussed previously.

Analysis
These results were used to determine the effectiveness of conventional mining machines in soil excavation. Of these machines, continuous miners are generally used to excavate
low strength material. These machines are either limited by torque or thrust. For the purpose of quantifying its effectiveness in this application a JOY 14CM9A Continuous Miner, with machine specifications as shown in Table 3, was considered in cutting a lunar regolith having a density of 1.8 t/m³.

A sensitivity analysis was undertaken based on the machines specifications for both the torque limited and thrust limited scenarios.

Analysis of the torque limited scenario considered the impact of changes in several machine parameters on cutting depth and machine mass the results of which are graphed in Figures 3 and 4 respectively. The horizontal dashed line in each of these two graphs indicates the ceiling above which the machine changes from being torque limited to thrust limited.

These two graphs indicate that a reduction in either motor power or efficiency, or conversely an increase in cutter speed would reduce cutting depth and effective machine mass. The reduction in mass is as a result of the reduction in depth, as a smaller cutting depth would require a lower sumping force.

The sensitivity analyses for the thrust limited scenario considered depth of cut and motor power. The effect of these changes can be seen in Figures 5 and 6.

In this thrust-limited scenario, changes in cutting speed and motor efficiency had no effect on cutting depth as shown...
in Figure 5. Consequently the machine would effectively remain above the ceiling and hence would always be torque limited. Only reductions in machine mass and the coefficient of friction of more than 20 per cent would cause the machine to become thrust limited, these two parameters being related to sumping force.

In terms of motor power, Figure 6 shows power varies inversely, as would be expected, with motor efficiency. However even at the limit of an increase in efficiency of 25 per cent, the machine remains torque limited. Reductions in the other machine parameters of machine mass and cutter speed of more than 20 per cent would cause the machine to become thrust limited.

As machine mass is a significant factor in the effectiveness of material excavation for conventional mining machines, an analysis was undertaken on the mass required to excavate the lunar regolith across a range of densities, from a minimum density of 1.5 to 2 t/m³. The results of this analysis can be seen in Figure 7.

As the graph shows, conventional mining systems can excavate lunar regolith without any significant challenge. A study by Bernold (2013) has found that pneumatic excavation systems can adequately excavate lunar regolith up to the maximum density. Consequently systems have the capability to excavate lunar regolith.

CONCLUSIONS

It has been found that both conventional and pneumatic excavation systems have the potential of excavating lunar regolith. In terms of technical and economic feasibility, the pneumatic excavation system is likely to be more advantageous for excavation in a lunar environment.

The feasibility of each system considered seven lunar conditions; gravitational force, vacuum, soil abrasion, electrostatic soil, soil compaction, dust suppression and transport costs. Of these conditions, the pneumatic excavation system was more adaptable in all conditions except for soil compaction. Transportation costs for both systems remains the main impediment in terms of overall economic sustainability.

The efficiency of conventional pick cutting on lunar regolith was determined using a newly design cuttability test facility and a regolith simulant material. The facility confirmed a low level of excavating forces over a range of compaction densities. The level of abrasion of the aluminium cutting tool indicated minimal amount of wear compared to conventional rock cutting.

ACKNOWLEDGEMENTS

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Using Clustering Methods for Open Pit Mine Production Planning

H Ren¹ and E Topal²

ABSTRACT

Typical mine planning process includes creating a mining block model, applying the ultimate pit limit analysis and creating mining cuts for production scheduling. Mixed integer linear programming (MILP) has been used extensively for optimal mine production scheduling of open pit mines. One main obstacle for large scale mine scheduling is the size of the problem. In large scale open pit mines, the number of blocks are too numerous to allow an optimal solution to be developed in a reasonable amount of time frame. However, the problem can be simplified by aggregation of mining blocks to create the mining cuts. The objective of this research is to analyse and validate a clustering algorithm for mining block aggregation. Popular cluster algorithms are reviewed and compared for the effectiveness of clustering for production scheduling purposes. Fuzzy C-Means (FCM) cluster method is used to partition mining benches and it is tested against the efficiency and practicality of mine production scheduling. The block aggregation algorithm is validated with a case study of a copper deposit. The net present value (NPV) of the production schedule which is created by using clustering algorithm is $2.25 M higher than the traditional pushback based production schedule.

INTRODUCTION

Mine planning is critical and directly influences the value of the project which is important to the shareholders and the community. A typical mine planning process involves finding the ultimate pit limit from a resource block model, developing a number of mining cuts and deriving a mine schedule which maximises the net present value (NPV). Mixed integer linear programming (MILP) has been used for optimising the production schedule of open pit mines since 1960s (Hustrulid and Kuchta, 2006). The objective function of the MILP formulation is to maximise the NPV of the mining operation subject to a set of constraints such as mining, milling and refining capacities, sequencing, blending requirement and minimum mining width (Askari-Nasab, Awuah-Offei and Eivazy, 2010). For a practical size mining block model, the time to find the optimal block-by-block mine life schedule on a powerful computer is too long or sometimes it is impossible to obtain a solution. Many researchers have studied the problem of mine production scheduling in open pit mines. Some of the approaches to the scheduling problem include heuristics, Lagrangian relaxation, parametric methods, dynamic programming techniques, mixed integer linear programming techniques and the application of artificial intelligence algorithm such as simulated annealing, genetic algorithms and neural networks. All of those approaches have limitations. Due to the complexity and size of the problem, most of those approaches suffer one or more of the following limitations: they cannot provide what is needed for most of the constraints that arise; they yield suboptimal solutions in most cases; or they can only handle small sized problems (Caccetta and Hill, 2003).

The constraints and variables can be reduced by aggregation of mining blocks. Block aggregation is a very important step in the preprocessing of data sets before optimisation. If it is done properly, despite the loss of detail, it enables difficult problems to be simplified without material loss of model integrity (Stone, 2013, pers comm). Therefore, one solution to reduce the computation time to find the optimum solution in scheduling is aggregation of mining blocks. Clustering algorithm technique has been used widely for the categorisation of large data set. There is a wide range of cluster algorithms that have been developed in psychology, biology, control and signal processing, information theory and data mining (Kogan, 2007). The clustering analysis seeks to group objects of a similar properties or value into different categories. It reveals conceptual, spatial and/or temporal and representational aggregation (Lee, Qu and Lee, 2012). Askari-Nasab et al (2010) have used the Fuzzy C-Means (FCM) clustering algorithm in creating mining cuts for production schedule. This algorithm is among the latest aggregation approaches available. The study focuses further on developing a deterministic MILP formulation, implementing and documenting the details of the MILP models in TOMLAB/CPLEX (by MATLAB®) environment, verifying and validating the MILP production scheduler by a case study against GEOVIA Whittle™ (by GEOVIA, Dassault Systèmes). The MILP scheduling result which was obtained using the data from FCM algorithm produced a NPV of $50.4 M or three per cent higher value than Whittle Millawa Balance algorithm from GEOVIA Whittle™ (Askari-Nasab et al, 2010). Further research were conducted on a two-stage clustering algorithm for block aggregation in scale open pit mines. One main obstacle for large scale mine scheduling is the size of the problem. In large scale open pit mines, the number of blocks are too numerous to allow an optimal solution to be developed in a reasonable amount of time frame. However, the problem can be simplified by aggregation of mining blocks to create the mining cuts. The objective of this research is to analyse and validate a clustering algorithm for mining block aggregation. Popular cluster algorithms are reviewed and compared for the effectiveness of clustering for production scheduling purposes. Fuzzy C-Means (FCM) cluster method is used to partition mining benches and it is tested against the efficiency and Practicality of mine production scheduling. The block aggregation algorithm is validated with a case study of a copper deposit. The net present value (NPV) of the production schedule which is created by using clustering algorithm is $2.25 M higher than the traditional pushback based production schedule.

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open pit mines. The two-stage clustering method is based on hierarchical clustering for block aggregation where the size of the maximum cluster size is controlled as an input. The second step involves improving the mining cuts that is created from the mining cuts with Tabu Search, which is a local search algorithm that balances between the quality of clusters and the number of binary constrains (Tabesh and Askari-Nasab, 2011).

The aim of this research is to evaluate the mine planning process in creating mining cuts for production scheduling and compare the NPV value of the schedules based on block aggregation technique versus that derived from classic pushback design.

METHODOLOGY
The approach in the analysis and the completion of the research required several steps to be taken. The major project tasks and activities include:

• familiarising with the most popular clustering algorithm for high dimensional data
• applying a suitable clustering algorithm for a block model data and revision of the block aggregation algorithm
• evaluating a case study by creating a long-term mine scheduling with clustering algorithm and compare it with classic pushback design production schedule.

Two main clustering algorithms, Hierarchical Cluster Algorithm and Fuzzy C-Means Cluster Algorithm were considered for the clustering of mining blocks. Brief descriptions and comparisons of these methods can be found in the following section.

Hierarchical cluster algorithm on generic block model
Hierarchical clustering is a method that groups data based on similarity by creating a cluster tree or dendrogram. The general procedure of these types of algorithms is reviewed as follows (Johnson, 1967):

The logic of the algorithm starts by considering each object as a cluster, in other words; if there are \( N \) objects, then there are \( N \) clusters. The similarities between clusters are the same as the similarities between the objects they contain. The major steps of the hierarchical cluster algorithm is summarised as follows:

• find the most similar pair of clusters and merge them into a single cluster
• compute similarities between the new cluster and the rest of the clusters
• repeat steps until all objects are merged together and a cluster with size of \( N \) is created.

Fuzzy C-Means cluster algorithm
Dunn (1973) developed the FCM algorithm. Bezdek (1981) introduced an improvement on traditional clustering methods. The technique assumes each data point belongs to a cluster to some degree, which is specified by a membership grade. The iterations minimise the objective function in Equation 1 that represents the distance from any given data point to a cluster centre.

\[
J_m(U, v_1, \ldots, v_K) = \sum_{i=1}^{n} \sum_{k=1}^{K} (u_{ik})^m d^2(x_i, v_k)
\]

where:

\( u_{ik} \) is the degree of membership of \( x_i \)

\( x_i \) is the \( i \)th object in the measured data

\( v_k \) is the centre of the cluster

\( d^2(x_i, v_k) \) is the square Euclidean Distance between \( x_i \) and \( v_k \).

The exponent \( m \) is chosen from \((1,\infty)\) in advance; it is used to determine the degree of fuzziness of the clustering, \( m \) between 1.1 and 5 is typically reported as the most useful range as recommended by Bezdek (1981).

Sato, Sato and Jain (1995) evaluated conventional clustering as classifying the given data into exclusive subsets. They described that the technique can clearly determine whether an object belongs to a cluster or not. However, this kind of partition is not sufficient to represent many real situations. The research team offered a fuzzy clustering method to construct clusters with uncertain boundaries. The FCM is one of the methods of fuzzy clustering. Figure 1 shows the scatter plot of the data. In this case, the FCM is trying to divide the data set into two sets based on their location. Figure 2 depicts the result of the FCM results. As can be seen from Figure 2, the data set marked x has centre X, whilst the data set o has centre O.
These two cluster methods are both available for clustering block model. Pros and cons of both methods are summarised in Table 1. Both have been used in mining application for clustering mining block models, for this research FCM is used as the method for block aggregation. This is because it has high flexibility to cluster the blocks and it can control cluster sizes. More importantly, the calculation steps to find the solution are shorter than the traditional hierarchical algorithm.

### TABLE 1
Comparison of hierarchical and Fuzzy C-Means.

<table>
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<tr>
<th></th>
<th>Hierarchical</th>
<th>Fuzzy C-Means</th>
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<tbody>
<tr>
<td><strong>Advantages</strong></td>
<td></td>
<td></td>
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<tr>
<td>Can handle categorical variables.</td>
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<td>Fewer steps to find solution.</td>
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<tr>
<td>Produces better clusters.</td>
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<td>Can control cluster sizes.</td>
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<td></td>
<td></td>
<td>Highly flexible.</td>
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<tr>
<td><strong>Disadvantages</strong></td>
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<td></td>
</tr>
<tr>
<td>Has more steps to find similarity.</td>
<td></td>
<td>It is not designed to partition categorical variables.</td>
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<tr>
<td>Slower when dealing with larger data sets.</td>
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### IMPLEMENTATION OF THE CLUSTERING ALGORITHM FOR BLOCK AGGREGATION

**Case study of a copper block model**

The FCM algorithm is implemented on a copper block model to demonstrate the practicability and the high project value to produce mining cuts. The algorithm of the FCM is described in Equation 1. The criterion for aggregating the block is based on the Euclidean distance between each block. For this copper block model, the ultimate pit limit has been calculated which includes 8196 blocks with a total of 19 benches. Blocks in this pit shell are exported to a CSV (comma-separated values) file for cluster analysis.

There are several steps to cluster the entire block model. The data has to be processed bench-by-bench. For block aggregation with clustering method, every bench is partitioned into a number of mining cuts starting from the bottom bench. The bottom bench has 17 blocks, forming the first mining cut. The next bench has 58 blocks which is clustered into two mining cuts. Clustering GUI Tool in MATLAB®, version R20131 (by MathWorks) has been used to perform FCM algorithm. The GUI can perform tasks as follows:

- load and plot the data
- clustering the data with FCM algorithm
- apply different control parameters such as cluster number (varies each bench), maximum iterations (100), exponent = 2 and minimum improvement 1e-005 for clustering
- save the cluster centre.

After performing the FCM clustering with a fixed number of centres, each block in the original CSV has a centre which indicates which mining cut number it belongs to. The centres are imported back to the CSV file and eventually to the original block model which can be viewed in 3D with mining software. The same process is carried out bench-by-bench by assigning the mining cut number to the block in the block model. Figure 3 presents the Bench 2240 in the copper block model before the clustering algorithm is applied. The same bench after the clustering algorithm is demonstrated in Figure 4. Overall 81 clusters have been created as the total clusters with this process.

Scheduling is done using Chronos software (Maptek Pty Ltd, 2012) which utilises a spreadsheet to store, manipulate and report scheduling data. Conventionally, the reserve is broken down by bench and user defined variables and scheduling constraints are given to the optimisation engine CPLEX (ILOG, 2005).

The overall NPV value of the scheduling from 81 mining cuts is $215.74 M. Figure 5 demonstrates the yearly production schedule for the life of the mine using FCM.

The same block model was used to derive the optimal production schedule based on conventional approach in the second step. For the pushback design three pit shells are selected from the pit-by-pit graph produced by the Pit Optimizer Tool in GEOVIA Whittle™ using the Lerch-Grossman algorithm. The reserve blocks in the block model are aggregated into mining cuts with the following method:

- Every reserve block is categorised into either Pushback 1, Pushback 2 or Pushback 3 based on their pit shell that is produced from Lerch-Grossman algorithm and criteria for minimum mining width problem.
- For each pushback, blocks that are at the same elevation belong to the same mining cuts. This process has produced 48 mining cuts which are less than the mining cuts produced from FCM algorithm. Figure 6 presents the corresponding annual ore and waste schedules. The schedule of the traditional pushback design yielded a
The production plan which was produced using the mining cuts from the FCM algorithm has a $2.2 M higher NPV value than the classical pushback method.

The MILP solution time is the same for both methods which was under three minutes under the same input parameters. However, the traditional pushback design involves the several steps to find a series of pushbacks and to eliminate the minimum mining width problem. On the other hand, the cluster design involves manually exporting and importing the block models into MATLAB® for analysis. Both methods can be time consuming for average users who wish to create a production schedule from a raw block model. At this stage, it is acceptable to conclude the FCM is a valid method to create a practical production schedule. More clustered mining cuts can be created to test for the potential of yielding higher NPV.

**CONCLUSIONS AND RECOMMENDATIONS**

The objective of this research is to analyse and validate a clustering algorithm for block aggregation. Popular cluster algorithms are reviewed and compared for the goodness of clustering for mining purpose. The FCM cluster method is used to partition benches of a case study and it is tested against the goodness of clustering and practicality of life of mine production scheduling. The result shows the FCM can partition blocks on every bench into different clusters without losing significant details. By using this cluster algorithm, the case study block model which has 8158 blocks has been clustered into 81 mining cuts. On the other hand, the conventional pushback design has created 48 mining cuts from the same number of blocks. The mining cuts formed from clusters resulted in an increase in NPV of $2.2 M as compared to pushback design.

More clustered mining cuts can be created to test for the potential of yielding higher NPV. However, the time and effort to apply the FCM process on a block model is too high. If this process can be automated within the mine design software, it would be more efficient for users to find the production schedule from the mining cuts that are created automatically. The coding or the automatic process could be future research areas.

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Optimising Cut-off Grade Strategy for Seabed Massive Sulfide Operations

S A Skorut\(^1\) and S Saydam\(^2\)

Polymetallic seabed massive sulfide (SMS) deposits represent a significant resource base which can help bridge the supply of critical minerals to society. Analysis is needed to apply strategic mine planning decisions in an SMS mining context. The selection of cut-off grade is a key strategic decision and consequently, can greatly influence a project’s net present value (NPV).

This project demonstrates the application of an optimal dynamic cut-off grade strategy to a hypothetical SMS deposit. A generalised cut-off grade optimisation algorithm was derived and applied to a case study, which produced a 16.5 per cent improvement in the present value when compared to a break-even cut-off grade. The influence of \textit{in situ} grade variance was found to be relatively small, with approximately a 1 per cent loss in present value expected on average when following an optimised plan.

Furthermore, methods and techniques pertaining to the application of the cut-off grade algorithm as a strategic tool are developed. Practical implications of optimal strategies in various scenarios are explored. These developments enable mine planners and decision makers to develop economically viable life of mine plans.

The algorithm presented can be used as a foundation for further developing strategic mine planning approaches for SMS operations.

INTRODUCTION

With the increasing use of metals in society, the average grade of mines is declining, despite improving mining, processing and refining technologies. The mining industry is seeking alternative resources to traditional land based deposits to fill the gap in supply. One of these alternatives is seabed massive sulfide (SMS) mineralisations, which are high-grade, polymetallic mineralisations containing copper, zinc, lead and tin as well as gold and silver. Analogous to the petroleum industry moving offshore to meet demand, the global occurrences of SMS mineralisations provide an opportunity for the mining industry to also move offshore.

Massive sulfide mineralisations are deposited from hydrothermal venting systems, which form at a diverse range of tectonic settings. The majority of hydrothermal systems occur along fast spreading mid ocean ridges, however the largest deposits occur at intermediate and slow spreading centres, at ridge axis volcanoes, in deep back arc basins and in sedimented rifts adjacent to continental margins (Hannington, De Ronde and Petersen, 2005).

The ore formation process is almost identical in each case, however the mineral composition varies due to the different composition of host volcanic rocks (Herzig and Hannington, 1995). More than 300 sites of hydrothermal activity exist, with over 100 sites forming massive sulfide deposits as indicated in Figure 1.

Herzig (1999) claims marine mining will be feasible under conditions of:
- high gold and base metal grades
- site location close to land and ideally within territorial waters of a coastal state
- relatively shallow water depth, not significantly exceeding 2000 m.

Hannington et al (2011) mentions that easily accessible SMS mineralisations have tonnages estimated in the order of 600 Mt, containing 30 Mt of copper and zinc. Consequently, the mining of SMS mineralisations can help secure the supply of critical minerals in the future as the mining industry continues to face declining average grades.

Mining of polymetallic deposits is soon to be a reality with two companies; Nautilus Minerals and Neptune Minerals; developing mining systems to extract SMS deposits at depths of 1000–2000 m below sea level, as shown in Figures 2 and 3, respectively.

Many potential environmental benefits of SMS over land based mining were cited by Fouquet and Scott (2009).

In depleting minerals from the earth’s crust it is paramount that these minerals are extracted in a manner that optimises the projects’ objectives, as they can only be mined once.

Maximising the economic value of a deposit, specifically, net present value (NPV) is the most common objective of mining operations. A key variable in maximising project NPV is cut-off grade. Cut-off grade is the criteria used to define ore and thus, differentiates between material scheduled for treatment and material considered to be waste (Lane, 1964).

Cut-off grade is a dynamic, strategic mine planning decision which has profound effects on the size of reserves and hence, all operational decisions influenced by project size, such as

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throughput considerations. Therefore, there is a need to optimise cut-off grade decisions in order to maximise the economic value derived from mineral properties.

Optimising cut-off grades for terrestrial open pit and underground operations is now standard practice in industry. Most cut-off grade strategies are derived from the pioneering work by Lane (1964), which identified the need to maximise the NPV of the project when optimising cut-off grades. Whilst Lane’s Algorithm presents a generalised mine model based upon both open pit and underground operations, further research has indicated that cut-off grade selection for underground mining needs to be more carefully determined due to mining selectivity (Yi and Sturgul, 1987). Similar to the way in which underground operations optimise cut-off grade strategies differently from open pit operations; SMS deposits require analysis and research in order to augment, modify and apply traditional cut-off grade theories.

**METHODOLOGY**

**Seabed massive sulfide strategic mine design**

Strategic mine design begins with a geological block model of the mineralisation, which is a representation of a deposit as an array of blocks. Each block has properties assigned, such as metal grades, density and geological domains. In its entirety the block model provides an indication of a deposit’s size, shape, grade distribution and other characteristics. Knowledge of these parameters enables engineers to evaluate the likely size and costs involved in mining the deposit. From this start point SMS mine planning, predominantly follows the traditional open pit planning process as illustrated in Figure 4.

Initial cost estimates combined with geotechnical data are used to design an ultimate pit. The selection of an ultimate pit is largely the same as with traditional open pit mining, using

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**FIG 1** – Global occurrence of sea floor hydrothermal vents and related mineral deposits (Hannington, De Ronde and Petersen, 2005).

**FIG 2** – Nautilus minerals proposed seafloor production system (Jankowski et al, 2010).
Lerchs-Grossman pit optimisation algorithm to determine a viable final pit.

In a standard open pit scenario pushbacks are defined. Pushbacks outline nested pits which are sequentially excavated in order to maximise cash flows by delaying waste movement. This process also applies to SMS mine design, however, SMS operations will generally mine bench by bench and thus, pushback design is not entirely necessary. In a traditional open pit scenario mining bench by bench generally produces the ‘worst case’ for cash flows. However, in an SMS context, deposits are flat lying with little overburden removal required. Therefore, comparisons between mining bench by bench versus utilising pushbacks may not lead to large differences.

At this point, the bench excavation cuts are designed with respect to the cut-off grade strategy selected. A detailed production schedule is then developed from the selected cut-off grade strategy. The production schedule sets forth the timing of bench excavations creating a schedule specifying the waste and ore tonnes mined as well as the ore grade per a period. From this point the process is completed again from start as production and economic inputs initially estimated have been refined and further developed.

**Generalised seabed massive sulfide operational model**

In order to frame the optimisation sequence, a generalised SMS operation model needed to be created in order to validly model the mining sequence. Lane’s (1988) three stage mine model is used to correctly classify each activity into the correct throughput category. Activities are separated into mining, treating and marketing activities as shown in Figure 5.

Utilising the throughput classifications, a generalised profit function can then be defined as in Equation 1 (Lane, 1988).

\[
P_t = (p - k)Q - hQ - mQ - f
\]

where:
- \( \tau \) = length of time period (quarters)
- \( P_t \) = profit per a period ($US millions)
- \( p \) = selling price of final product ($US/t)
- \( k \) = selling cost of final product ($US/t)
- \( h \) = treatment cost per a tonne of ore treated ($US/t)
- \( m \) = mining cost per a tonne of mineralised material mined ($US/t)
- \( f \) = fixed cost per a time period ($US)

![FIG 3](image-url) Neptune Minerals proposed subsea mining system (Parenteau et al, 2013).

![FIG 4](image-url) Traditional open pit circular planning process (Dagdalen, 2001).

![FIG 5](image-url) Classification of SMS operational activities.
$Q_m =$ quantity of material mined (Mt)

$Q_t =$ quantity of ore treated (Mt)

$Q_f =$ quantity of final product sold (kt)

**Objective function**

The objective of cut-off grade optimisation is to maximise the NPV of a project, as in Equation 2.

$$NPV = \sum_{t=0}^{T} \frac{P_t}{(1 + d)^t}$$

where:

- $t =$ life of mine (quarters)
- $d =$ discount rate (equivalent annual discount rate)

For a cut-off grade strategy to be optimal, it must be optimal in every time period (Lane, 1988). Thus the objective function is reduced to the maximisation of present value per a period. The difference in value between two time periods is given by Equation 3.

$$v = P_t - F$$

where:

- $v =$ difference in value between time periods ($US$)
- $F =$ opportunity cost of remaining future value ($US$)

Thus, the final objective function is defined as in Equation 4, which states that per a unit of resource depletion, the value derived from that unit is maximised.

$$\text{Max}(v) = \text{Max}[(p-k)Q_r - hQ_t - mQ_m - (f+F)v]$$

**Cut-off grade solutions**

When maximising the objective function, three closed form solutions occur. These are the three limiting economic cut-off grades which occur when either the mine, treatment or market limits throughput. Three more cases arise when more than one component limits throughput. The following discussion is drawn from Lane (1988).

**Economic limiting cut-off grades**

The cut-off grade resulting from the mine limiting case is shown in Equation 5.

$$g_m = \frac{h}{(p-k)y}$$

where:

- $g_m =$ mine limiting cut-off grade (%)
- $y =$ yield in the ore treatment process (%)

The mine limiting cut-off grade is solely dependent on economics, thus no reference to present values exist. When the treatment limits throughput Equation 6 shows the corresponding cut-off grade.

$$g_t = \frac{h + \left(\frac{f+F}{H}\right)}{(p-k)y}$$

where:

- $g_t =$ treatment limiting cut-off grade (%)
- $H =$ treatment capacity (Mt)

The treatment cut-off grade is dependent on opportunity cost and thus future present values. The market limiting cut-off grade shown in Equation 7, is also dependent on opportunity cost.

$$g_r = \frac{h}{\left(p-k - \left(\frac{f+F}{R}\right)\right)y}$$

where:

- $g_r =$ market limiting cut-off grade (%)
- $R =$ market capacity (kt)

**Balancing cut-off grades**

In mining operations where more than one component is limiting throughput the utilisation of capacities is determined by the grade distribution, as well as cut-off grade. By comparing the ratios between capacities and comparing this value to the cumulative grade distribution of the deposit, a balancing cut-off grade can also be determined. Thus three more cut-off grades can be calculated corresponding to the balancing of each pair of throughputs as shown in Table 1.

For balancing cut-off grades to be valid the grade distribution of mineralised material must be known with reasonable certainty. Preferably, the grade distribution for each pushback or phase has been developed. Therefore, balancing grades are applicable in both strategic and tactical decision (Lane, 1988).

**Effective optimum cut-off grades**

Lane (1988) has shown that the optimum cut-off grade is one of the six cut-off grades. By graphically plotting the value derived from each cut-off grade an optimal grade can be found algebraically as shown in Figure 6. An effective optimum cut-off grade can be found for each pair of cut-off grades. The optimum cut-off grade is then the median of these three effective cut-off grades.

**Optimisation algorithm**

With the optimum cut-off grade now defined, the dynamic method of determining a cut-off grade strategy which maximises the objective function can be developed. The algorithm for determining optimum cut-off grades in each period is shown in Figure 7. The algorithm was validated against that developed by Dagdalen (1993), producing a result within 0.1 per cent.

**CASE STUDY**

**TABLE 1**

<table>
<thead>
<tr>
<th>Cut-off grade</th>
<th>Limiting pair</th>
<th>Ratio</th>
<th>Graph</th>
</tr>
</thead>
<tbody>
<tr>
<td>$g_{mb}$</td>
<td>Mining and</td>
<td>H/M</td>
<td>Cumulative grade distribution (x)</td>
</tr>
<tr>
<td></td>
<td>treatment</td>
<td></td>
<td></td>
</tr>
<tr>
<td>$g_{mr}$</td>
<td>Mining and</td>
<td>R/M</td>
<td>Recoverable mineral per a unit of</td>
</tr>
<tr>
<td></td>
<td>market</td>
<td></td>
<td>mineralised material ($/t$)</td>
</tr>
<tr>
<td>$g_r$</td>
<td>Market and</td>
<td>R/H</td>
<td>Recoverable Mineral per a unit of</td>
</tr>
<tr>
<td></td>
<td>treatment</td>
<td></td>
<td>ore ($/t$)</td>
</tr>
</tbody>
</table>

The optimisation algorithm was applied to a theoretical case study. The case study was constructed from publicly available data pertaining to Nautilus Minerals Solwara 1 deposit; in order to closely resemble reality. The block model contains grade estimations for a total of 22,471 blocks containing copper, gold, silver and zinc. Blocks have dimensions of

54
10.0 m in the Easting and Northing directions and 0.5 m in the vertical dimension. The vertical extent was selected to match the corresponding minimum mining unit and to match Nautilus Minerals Solwara 1 block model dimensions. At an average dry bulk density of 3.0 t/m³ the total mineralised region contains 3.4 Mt.

The deposit will be mined using Nautilus Minerals’ proposed seafloor mining system as described by Jankowski et al (2010).

The SMS mineralisation has a total resource tonnage of 3.4 Mt, containing copper and gold. It is planned to be mined at 1.2 Mt/a, with a 0.8 Mt/a treatment rate, resulting in an estimated life of mine (LOM) of 2–5 years.

Grade distribution data
A block model was created from publicly available drill hole data on Nautilus Minerals’ Solwara 1 deposit (Lipton et al, 2012). The block model is shown in Figure 8. Gold grades were converted to an equivalent copper grade for grade estimation using the method set out by Ataei and Osanloo (2003a).

The resultant grade-tonnage distribution data is presented in Table 2.

Input data
Operational and economic input data are shown in Table 3. Values used in the optimisation were either obtained or back
calculated from Nautilus Minerals’ Cost Study (Jankowski et al., 2010).

Importantly, some parameters stray significantly to those usually encountered in the traditional mining systems. The discount rate is relatively high at 25 per cent, this value is valid given the unproven nature of SMS operations. Treatment recovery is 100 per cent as the equivalence grades already include recoveries.

Treatment costs are relatively high to reflect the categorisation of specific activities into the treatment category. Finally, the cost associated with running a production support vessel regardless if mining is occurring or not, supports the high fixed cost used. Note that the optimisation has been completed in quarters to highlight less obvious findings, due to the short LOM. An equivalent quarterly discount rate was applied to ensure validity.

**CASE STUDY FINDINGS**

The results of the algorithm were compared to the use of break-even cut-off grades and Heuristic methods of cut-off grade determination. The present values are used to indicate value, as all methods have the same capital expenditure.

**Optimisation algorithm results**

Output from the optimisation algorithm, predicated from the case study, is shown in Table 4.

The strategy yields a present value of $462.9 M over a LOM of 12 quarters.

The outlined strategy results in a balanced cut-off grade of 2.76 per cent from the 1st to the 8th quarter, resulting in the mine and treatment capacities both limiting overall system throughput. From the 9th to the 12th quarter the treatment capacity is the limiting throughput and thus a declining cut-off grade ensues during this period.

**Break-even cut-off grade comparison**

The algorithm’s output was compared with results obtained from using break-even cut-off grade formula. The break-even strategy achieves the schedule shown in Table 5, resulting in a present value of $397.3 M over a LOM of 16 quarters. The cut-off grade remains fixed throughout the LOM at 0.83 per cent resulting in 11.6 kt of equivalent copper metal being refined each period.
The optimisation algorithm produced a present value of $462.9 M, a 16.5 per cent improvement over the break-even strategy. The improvement in present value is a result of the opportunity cost of the break-even strategy; lower grade material is being mined when higher grade material is available. The algorithm explicitly values opportunity cost and thus implements a higher cut-off grade earlier in the mines LOM to realise cash flows earlier and thus improve present value. Figure 9 illustrates the comparison between the two strategies.

### TABLE 4
Optimisation algorithm cut-off grade schedule.

<table>
<thead>
<tr>
<th>Qtr</th>
<th>Cut-off grade (%)</th>
<th>Average grade (%)</th>
<th>Qm (Mt)</th>
<th>Qh (Mt)</th>
<th>Qr (Kt)</th>
<th>Profit ($M)</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>53.9</td>
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<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>51.0</td>
</tr>
<tr>
<td>3</td>
<td>2.76</td>
<td>7.48</td>
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<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>48.3</td>
</tr>
<tr>
<td>4</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>45.6</td>
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<td>7.48</td>
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<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
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</tr>
<tr>
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<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>40.8</td>
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<tr>
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<td>57.0</td>
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</tr>
<tr>
<td>8</td>
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<td>7.48</td>
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<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>36.5</td>
</tr>
</tbody>
</table>

### TABLE 5
Cut-off grade schedule using break-even formula.

<table>
<thead>
<tr>
<th>Qtr</th>
<th>Cut-off grade (%)</th>
<th>Average grade (%)</th>
<th>Qm (Mt)</th>
<th>Qh (Mt)</th>
<th>Qr (Kt)</th>
<th>Profit ($M)</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
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<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
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<td>38.9</td>
<td>34.8</td>
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<td>11.6</td>
<td>38.9</td>
<td>32.9</td>
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<td>38.9</td>
<td>31.1</td>
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<tr>
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<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>24.9</td>
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</table>

### TABLE 6
Cut-off grade schedule using Heuristics algorithm.

<table>
<thead>
<tr>
<th>Qtr</th>
<th>Cut-off grade (%)</th>
<th>Average grade (%)</th>
<th>Qm (Mt)</th>
<th>Qh (Mt)</th>
<th>Qr (Kt)</th>
<th>Profit ($M)</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
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<td>40.7</td>
</tr>
<tr>
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<td>6.64</td>
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<td>0.20</td>
<td>13.3</td>
<td>48.1</td>
<td>38.5</td>
</tr>
<tr>
<td>5</td>
<td>1.58</td>
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<td>30.1</td>
</tr>
<tr>
<td>9</td>
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<td>0.20</td>
<td>13.1</td>
<td>46.9</td>
<td>28.4</td>
</tr>
<tr>
<td>10</td>
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<td>6.53</td>
<td>0.24</td>
<td>0.20</td>
<td>13.0</td>
<td>46.8</td>
<td>26.8</td>
</tr>
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<td>5.74</td>
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<td>0.20</td>
<td>11.5</td>
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<td>20.5</td>
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<tr>
<td>12</td>
<td>0.83</td>
<td>5.73</td>
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<td>18.1</td>
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<tr>
<td>14</td>
<td>0.83</td>
<td>5.65</td>
<td>0.21</td>
<td>0.20</td>
<td>11.3</td>
<td>36.8</td>
<td>16.9</td>
</tr>
<tr>
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<td>0.13</td>
<td>0.12</td>
<td>7.0</td>
<td>23.0</td>
<td>9.1</td>
</tr>
</tbody>
</table>

Totals 6.32% 3.37 2.92 184.4 $648.4 $438.9

**Heuristic cut-off grade method comparison**

Heuristic methods involve using break-even formulae but may add additional provisions over periods of time. The Heuristics method resulted in a present value of $438.9 M over a LOM of 15 quarters as shown in Table 6.

The cut-off grade declines in steps over time in correspondence to the assumptions used as shown in Figure 10. The total present value of the optimisation algorithm achieves a 5.5 per cent increase in present value over the use of Heuristic methods.

When compared with break-even and Heuristic methods of cut-off strategies the algorithm achieves a greater present value by 16.5 per cent and 5.5 per cent, respectively. In both cases the algorithm created strategies which:

- produced greater present values
- mined a higher average grade over LOM
- classified and hence treated less total ore
- refined and sold less final metal
- mined for less time (lower LOM).
A summary of results for the different cut-off grade strategies is shown in Table 7.

**DISCUSSION**

**Optimal capacities**

Based on the current SMS mining systems proposed, the limiting throughput is likely to be either mine, treatment or both mine and treatment limiting in balance. This is a consequence of the unique logistics of SMS operations and also SMS mineralisations characteristically low strip ratios.

The determination of an optimal capacity can be solved quantitatively by considering cut-off grades as a production driver. The cut-off grade optimisation algorithm can then be used as a strategic mine planning tool. By providing a capital cost matrix and resultant variable costs, various mine plans can be evaluated within the algorithm. Each scenario was evaluated using the algorithm, with the resultant NPV contours shown in Figure 11.

There is a clear plateau illustrated by Figure 11 of economically feasible mine plans, with this knowledge mine planners can focus on systems corresponding to those capacities.

The algorithm can also be used to identify risk-reward situations via sensitivity analysis in designing mine plans. Mine plans can be evaluated on their ability to handle adverse changes in commodity prices. A robust mine plan can then be triangulated which achieves the highest possible expected outcome given a set of economic conditions. Using the augmented case study data, a down case as represented by a -10 per cent change in copper price and an up case as represented by a +10 per cent change in copper price were evaluated against the various mine plan combinations as shown in Figures 12 and 13 respectively.

The identification of relative mine plan combinations which outperform can empower mine planners to make robust strategic decisions. The output for each scenario can be compared to identifying changing characteristics in the mining system and also what course of action to take. This

### Table 7: Summary of cut-off grade strategies.

<table>
<thead>
<tr>
<th></th>
<th>LOM (quarters)</th>
<th>Average LOM grade (%)</th>
<th>Total $Q_m$ (Mt)</th>
<th>Total $Q_e$ (Mt)</th>
<th>Total $Q_r$ (Kt)</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Break-even formula</strong></td>
<td>16</td>
<td>5.82</td>
<td>3.37</td>
<td>3.17</td>
<td>186.2</td>
<td>397.3</td>
</tr>
<tr>
<td><strong>Heuristic algorithm</strong></td>
<td>15</td>
<td>6.32</td>
<td>3.37</td>
<td>2.92</td>
<td>184.4</td>
<td>438.9</td>
</tr>
<tr>
<td><strong>Optimised algorithm</strong></td>
<td>12</td>
<td>7.43</td>
<td>3.37</td>
<td>2.27</td>
<td>168.8</td>
<td>462.9</td>
</tr>
</tbody>
</table>
provides a relatively rapid and valid basis for making mine planning decisions in a changing economic environment.

**Grade control**

Grade control is a necessary activity for any mining operation and often adds significant cost. In an SMS setting grade control issues become more contentious due to poor core recoveries and high drilling costs. Consequently SMS operations are likely to have highly variable head grades when compared to expected or predicted grades.

An optimal strategy is based upon the assumption that the grade-tonnage curve is correct and grade descriptors will be correctly and validly applied in practice. Consequently strategic decisions will be presumed optimal from the data, but non-optimal when compared to what is actually in the ground.

Monte Carlo analysis was conducted to determine the potential loss in value of following a mine plan which has been based off a grade-tonnage distribution which is truly variable in nature. One thousand iterations were completed, with metal grades the only stochastic variable. The variability was achieved assigning a normal distribution for the interval tonnages, with an expected mean equal to the base case tonnages and a standard deviation of 20 per cent of the mean applied. Table 8 shows resultant descriptive statistics.

The value derived from following the perceived optimal strategy (that is the strategy derived from the original case study) was then compared with the optimised value for each iteration. The present value differentials were then plotted on a frequency histogram as shown in Figure 14.

The variance is surprisingly small, with a mean present value loss of $4.1 M and a maximum loss of $16.6 M. In terms of the initial present values a percentage spread is more meaningful. Descriptive statistics are shown in Table 9, the mean confers a 0.9 per cent loss and a maximum loss of 4.3 per cent of present value. Subsequently on average, following a perceived optimal plan will produce a relatively small loss of approximately 1 per cent of an operations present value. This result in itself makes the use of cut-off grade optimisation worthwhile, as it produces quite robust mine plans.

The actual variance in the optimal strategy versus the perceived optimal strategy was also compared. Figure 15 illustrates the range of optimal strategies produced from the stochastic grade-tonnage distribution. The dashed lines represent the distribution of optimal cut-off grades with one standard deviation (68 per cent) of values falling between the dashed lines.

The optimal cut-off grades differ by up to 1 per cent equivalent grade from the perceived optimal strategy. The spread is clearly larger in early periods and converges at the end of LOM; a reflection that over time, the ability to influence a projects present value is diminished.

It is tempting to conclude that extensive grade control is not required from the perspective of strategic decision making. However, grade control is entirely necessary operationally in order to identify on a tactical level what is being mined; a particularly difficult task in a deep sea mining environment.

**Practical considerations and limitations**

**Grade distribution**

Cut-off grade optimisation assumes that any ore block has the same grade distribution specified by the grade-tonnage curve. Consequently the schedules produced in the analysis are inadequate for detailed mine planning. Additionally it is assumed that there is sufficient mineralised material uncovered for mining.

To overcome the limitation the orebody can be divided in panels, each with its own grade-tonnage distribution as described by Dowd (1976). A sequential mining sequence can then be generated.

---

**TABLE 8**

Descriptive statistics of Monte Carlo analysis.

<table>
<thead>
<tr>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
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<tr>
<td>Minimum</td>
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<tr>
<td>Maximum</td>
</tr>
<tr>
<td>Range</td>
</tr>
<tr>
<td>Standard deviation</td>
</tr>
</tbody>
</table>

**TABLE 9**

Potential value loss descriptive statistics.

<table>
<thead>
<tr>
<th>Present value loss differential (% of optimal)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
</tr>
<tr>
<td>Minimum</td>
</tr>
<tr>
<td>Maximum</td>
</tr>
<tr>
<td>Range</td>
</tr>
<tr>
<td>Standard deviation</td>
</tr>
</tbody>
</table>

**FIG 14** – Frequency distribution of present value differentials of perceived optimal strategy.

**FIG 15** – Range of optimal cut-off grade strategies produced.
Stockpiling intermediate grades

Optimised strategies lead to overall lower mineral recovery. As a consequence of implementing a higher cut-off grade in early periods, material that would have been defined as waste may be economical later in the operations LOM when the cut-off grade is lowered. Intermediate material is usually stockpiled and fed to the mill during periods of low utilisation, for blending purposes and after mining has ceased. Seafloor stockpiles of fragmented intermediate material are improbable as they will require upkeep over the LOM. It is unlikely such methods will be economically feasible and consequently intermediate ore will most likely become sterilised.

Equivalent grades

The optimisation uses equivalence grades in order to convert minor economic mineral grades to a single, primary mineral. This method of handling polymetallic deposits becomes invalidating if: no definitive correlation exists between the mineral occurrences, one or more metals is limited at the market level, or input parameters significantly change in value from that used to calculate the equivalence grades.

The limitations regarding equivalence grades can be resolved by utilising ‘Grid Search’ or ‘Golden Section’ methodologies as describe by Lane (1988) and Ataei and Osanloo (2003b) respectively. The methods extend the operational model and hence the objective function to include two metals, instead of converting one metal to another.

Fixed treatment yields and prices

In practice treatment yields will vary with the average head grade entering the mill. Furthermore mill facilities are generally designed and optimised for a specific range of head grade entering the mill. Furthermore mill facilities are generally designed and optimised for a specific range of head grades. As a result, it is invalidating to assume a low-grade block will achieve the same recovery yield as a high-grade block.

The implementation of parametric equations into the grade estimation process can assist in relieving the assumption of fixed treatment yields. By doing so yields will be able to vary with the estimated block grade, thus providing further validity to the model and hence the optimisation.

CONCLUSIONS

This paper has presented models, algorithms and considerations to apply existing cut-off grade strategies to an SMS operation. The application of cut-off grades to SMS operations was achieved by augmenting existing cut-off grade theories.

When applied to the theoretical case study the strategy produced by the optimisation algorithm provided significant increases (16.5 per cent) in project present value over break-even cut-off grade strategies; and a modest increases (5.5 per cent) in present value over the use of Heuristic methods. Techniques for using the cut-off grade optimisation algorithm to evaluate mine plans were also presented.

Importantly, in a SMS context, grade control is a contentious issue. It was shown that, by following a perceived optimal plan, a relatively small loss of approximately 1 per cent of an operations present value on average can be expected. Additionally practical implementation issues and limitations such as grade distribution, equivalence grades, stockpiling and fixed yields were discussed.

By utilising optimisation algorithms, mine planners can readily identify operational characteristics, which can be exploited to ultimately add value to an operation. As a key component in strategic mine planning, cut-off grade optimisation can have a profound influence on the economic value derived from mineral properties.

RECOMMENDATIONS

This research and subsequent findings have facilitated the formation of several recommendations aimed at improving strategic mine planning in SMS operations. Importantly the algorithm and conclusions presented can be used as a foundation for further development and implementation of advanced techniques in an SMS mining setting. The findings include the following recommendations:

- implement grid search or golden section methodologies to improve optimisation validity
- incorporate a stochastic framework for grade estimation in order to minimise the influence of modelled geological variability on mine planning decisions.

REFERENCES


Optimising Cut-off Grade Strategy for Seabed Massive Sulfide Operations


A Critique of Selective Mining Unit Sizing at Century Mine to Optimise Productivity with Dilution
C B Smith¹, M Nehring² and M K Bosompem³

ABSTRACT
The mining industry of the modern era is at a fever pitch with innovation driven by the motivation to keep sales margins high amidst an ever-volatile commodities market. Autonomous mining, hybrid ore mining methods and increasingly larger and more mobile machinery continue to have a greater impetus than in the past. Amidst this flourishing innovation, the underlying principle of mining remains true; an operation is running at its most optimal point when it can attain the highest net present value possible from the in situ ore within its lease boundary. There are numerous drivers that stipulate the discounted cash flow of an operation. Getting swept up in, and fixating on, just a select few of these drivers, is a critical flaw in some aspects of modern mine planning because other important value drivers can be undermined and significantly compromised.

This paper acknowledges and focuses on the selective mining unit (SMU) sizing selection at Century Mine in far north-west Queensland to demonstrate that an intricate optimisation can be achieved between all key value adding drivers in a mining project. Presently, the mine as an operation is faced with numerous challenges; not only must the site remain profitable throughout a trough in the industry, it must also ensure that logistical challenges such as ore stockpiling are acted on before the situation turns dire. Appropriate SMU sizing is crucial as it stipulates the selectivity of ore excavation, and in turn impacts the extraction rate of this ore. Downstream effects of this process ultimately tie in the whole operation with the SMU size selected. This optimisation study aims to elucidate an SMU size that simultaneously maximises the discounted cash value and improves the ore stockpiling of the open pit base metals operation. It will be shown that net present value improvements up to 1 per cent over a 30 month period are achievable, and run-of-mine stockpiles can be improved by as much as 1400 t/day. Furthermore the project highlights that there is a potential to improve the discounted cashflow of other surface hardrock mines, by fixating on the selection of an appropriate SMU size for the operation in subject.

INTRODUCTION
At the cornerstone of hardrock mine planning is an ore block model; it dictates both the tonnage and grade of mineralisation, and it is adapted into computer aided drawing packages in order to develop and design the progression of all forms of mines. Within this block model are individual blocks, all with a designated width, length and height. Due to the multidirectional facet of mining, the length and the width of these blocks are generally equal to each other, and their height is based on numerous factors such as the equipment on-site and their vertical mining capabilities. The dimensions of these blocks stipulate how selectively material must be excavated, and therefore each individual block is coined as a selective mining unit, or SMU.

The task was set forth to assess the impact that an SMU has on six key value drivers within any mining operation. These drivers are as follows:

1. production rate
2. extracted grade
3. production cost.
4. milling:
5. throughput rate
6. milled recovery
7. milling cost.

Century Mine was used as a case study for the foundations of this analysis. Century Mine is an asset owned and operated by MMG Limited in north-west Queensland, and produces upwards of 470 kt/a of zinc in concentrate form. Commencing operations in 1998, the ore reserve is projected to be fully exhausted in 2016, leaving the mine with a remaining life of approximately 30 months. Even in the later stages of the mining phase there is an emphasis on optimising the operation. In assessing the various impacts that the SMU size...
has on a mining operation, the analysis was scoped in a way to delineate an SMU size that would:
- maximise the net present value (NPV) of the in situ orebody
- improve ore stockpiling logistics
- be more practical for operators to abide by.

This paper aims to elucidate the findings of the project by investigating the current state of knowledge, sharing the project methodology undertaken and discussing the findings and results of the project overall.

CURRENT STATE OF KNOWLEDGE

Selective mining

Noble (2011) defines selective mining as a process that specifically focuses on the accurate and precise extraction of economically optimal material in an effort to make the greatest return on a finite deposit. It is synonymous for its difference to bulk mining in the following ways (Swanepoel, 2003):
- mine plans are generally more intricate, and blast patterns are more precise
- exploration core is continually assayed for geological data, rather than just geotechnical data
- excavating equipment is generally smaller and more flexible
- excavation is focused more on precision than total movement
- haul trucks may be smaller to optimise cycle times with excavators, however this is not always the case
- mining unit costs are higher
- mining production rates are lower.

Kruger and van der Westhuizen (2005) conducted a study on the Thabazimbi iron ore mine in South Africa and found that significant economic improvements could be made on the operation by:
- adjusting the mining direction to follow the dip of the orebody and separate waste material more freely
- downsizing the equipment on-site to mitigate dilution and adhere to proper grade control measures
- reducing the bench height from 15–10 m to allow for an effective delineation between ore and waste.

Importantly the general consensus within the industry is that selective mining should ideally optimise the point between productivity and precision with regards to ore extraction.

Ore dilution

Bertinshaw and Lipton (2007) divides the roof causes of dilution into the following categories:
- Dilution due to geometry whereby the operation of excavation equipment cannot be physically matched with the shape and size of the orebody. Such an issue is commonly related to vein style ore deposits being extracted by surface methods.
- Dilution due to uncertain knowledge of the orebody whereby sampling and assaying precision is either limited or inadequate.
- Dilution due to blast movement whereby ore is thrown or heaved during the blasting process.
- Dilution due to mining errors and poor grade control where ore is incorrectly marked out on benches.

Importantly, dilution is measured as the percentage change between the in situ grade and the extracted ore grade, rather than the magnitude of the change between these two values. To assess the influence on the extracted ore grade that various block sizes has, a block model assembled with the geostatistical method of kriging is encouraged. From there, the blocks can be manipulated into various shapes and sizes by using a volume weighted average technique. The analysis conducted in this report utilised this method.

Selective mining unit sizing

Leuangthong, Nenfeld and Deutsch (2002) describes the conventional definition of the SMU as the smallest volume of material on which ore waste classification is determined. The author however stresses that such a convention is impractical, as the sheer size of mining equipment generally negates the practicality of freely separating ore from waste material. The paper then introduces the concept of optimising the SMU size whereby, instead of focusing on a perfect extraction of the ore as per its original definition, a point is determined that economically optimises the extraction of ore, even if such a practice is dilutive.

Leuangthong, Nenfeld and Deutsch’s (2002) refined definition of an SMU is therefore the block model size that would correctly predict the tonnes of ore, tonnes of waste, and diluted head grade that the mill will receive with anticipated grade control practice. It is important to note that while the author of the paper disagrees with the conventional definition of an SMU, he does agree that the selective capacity to mine ore depreciates with an increasing SMU size, a theory widely proven in multiple case studies (Szmiegiel, 2005). Ultimately the optimal selection of an SMU is an iterative process that encompasses and relies upon numerous economic factors (Leuangthong, Nenfeld and Deutsch, 2002; Hekmat, Osanloo and Asi, 2008), including:
- equipment size
- mining method
- mining costs
- ore dilution
- ore and waste tonnage variances
- mill recovery.

Net present value sensitivity to value drivers

Accounting for the time value of money and discounting cash flow accordingly is crucial in order to conduct a fair economic assessment. The discount relative to time that is influenced by numerous factors in Australia:
- consumer price inflation
- employee industrial action, especially in coal mining operations
- uncertainty surrounding future Australian taxation policies
- non-profitable commodity prices being realised
- foreign exchange rate fluctuations
- general project execution risks (Harrison, 2010).

Although the theory behind a discount rate is relatively straightforward, there is still great conjecture over what discount rate to use in certain scenarios. For example the present day average discount rate for Australian projects ranges between 8 per cent and 12 per cent, however there is no definitive, publicly accepted range that the industry turns to when in doubt of what rate to specifically use. It may pay to be conservative and use a higher discount rate, but therein lies its own issue (Quiggin, 1997) in that potentially profitable projects may be neglected due to having a negative NPV at one discount rate while still remaining positively valued at a slightly lower rate.
The following key strengths exist in the current state of knowledge. There is:

- ample evidence to suggest that an increased SMU size increasingly dilutes ore grade
- significant proof that selective mining has the ability to generate a more efficient ore mining operation over bulk mining
- an industry wide recognition of the risks of increasing milling throughput
- a general acceptance that Australian mining cashflow is discounted at 8 per cent to 12 per cent/annum in the current economic climate and at this point in time.

Having said this, there are also key weaknesses that exist in the current state of knowledge. There is:

- very limited research conducted on the manner which various specific SMU sizes simultaneously react with mined grade and mining productivity
- no current analysis conducted that simultaneously optimises SMU sizing with both mining and milling to maximise the value of *in situ* ore.

**METHODOLOGY AND ANALYSIS**

A combination of on-site and off-site work at Century and in Brisbane university offices respectively was required in order to undertake the analysis required. Contingency plans were put in place from the outset in order to ensure that the project’s progression was not compromised. At Century, an operational survey was conducted to assess the operational practicality of various SMU sizes relative to the excavation machinery on-site. A formal data collection process also allowed for the author to obtain crucial data related to historical production, future budgeted production and operational costs.

In Brisbane, significant progress was made on amalgamating, sorting and validating the obtained data. A formal analysis process then proceeded, allowing for relationships to be drawn between SMU sizes and the aforementioned key value drivers for Century Mine. A vast majority of the analysis was conducted in Microsoft Excel using an optimisation spreadsheet that drew data from various arranged sources within the workbook.

**RESULTS AND DISCUSSION**

**Century Mine**

**Practical optimisation**

The current SMU block size in place at Century has dimensions of 5 m × 5 m × 3 m, equivalent to a volume of 75 bank cubic metres (bcm). This size was originally chosen as it is considered somewhat of a convention within the industry. An operational survey was conducted on-site and it was determined the smallest practical SMU size was in fact 9 m × 9 m × 3 m. This is due to the following reasons.

Firstly, Komatsu (2009) specifications for the three excavators on-site, PC1800, PC2000 and PC3000, denote that the average dig reach is approximately 9 m in length, as seen in Figure 1.

Further analysis on the dig span revealed that the Komatsu excavators at Century require a lateral working area of at least 9 m as well. The excavators cannot selectively delineate material within this dimensional range, therefore limiting their selective capacity to a length of 9 m and a width of 9 m.

Finally, the bucket specifications for the excavators depict a bucket depth of 3 m on average, as seen in Figure 2. Once again, this limits the ability of the excavator to freely separate ore from waste material within this 3 m boundary.

Thus, it can be seen that any SMU size smaller than 9 m × 9 m × 3 m is impractical for the operation at Century.

**Economic optimisation**

Data has been amalgamated and assessed for the impact that it has on the six key value drivers for Century Mine’s operation. It was found that:

- mined grade increasingly decays further with an increasing SMU size and this is due to a decreasing selectivity
- mine production rates increase linearly up to the equipment threshold with an increasing SMU size
- ore mining costs decrease linearly with an increasing SMU size due to improved equipment utilisation
- milling rates must increase with an increasing SMU size in order to offset the decreased mined grade and produce the same quote of metal as the status quo
- in turn, milling recovery drops with an increased SMU size, as float resonance and grinding cycle times decrease
• milling and asset management costs increase with an increased SMU size as wear and tear on milling equipment increases with an increased throughput rate.

A NPV optimisation was conducted, incorporating these findings and the current foreign exchange and commodity prices for the Australian dollar and zinc spot price respectively. It was found that, for a 12 per cent/annum discount rate, over the remaining mine life of approximately 2–2.5 years at Century, the SMU size that will maximise the NPV of the in situ ore is the greatest is 250 bcm. At this SMU size, compared to an SMU size of 75 bcm:
- mined grade is diluted by 0.93 per cent
- mine production rates rise by 8.04 per cent
- mine operating costs drop by 5.78 per cent
- milling rates must increase by 0.96 per cent
- mill recovery will fall by 0.18 per cent
- milling and asset management costs will increase by 0.33 per cent.

Figure 3 denotes the relationship between the percentage change in NPV and the SMU size selected.

Most importantly, the NPV increase is within the realm of 1 per cent for the remaining mine life, which will boost the project’s discounted value by approximately $5 M to $7 M. The reason for this finding can be summarised by the optimisation between ore dilution and the pace of ore extraction. This is highlighted by the fact that an SMU size of 600 bcm implemented over a 30 month period will decrease the NPV of the in situ ore by almost 5 per cent. Ultimately it can be seen that an SMU size of 250 bcm is ideal for the proposed time period of implementation.

Stockpiling optimisation
Figure 4 denotes that the most ideal SMU size to optimise stockpiling, and improve it by the greatest extent is 525 bcm. However, for the remaining mine life this SMU size will have a detrimental impact on the project’s NPV by as much as 3 per cent. The economically optimised SMU size of 250 bcm will improve ore stockpiling by as much as 1400 t/day, significantly helping the operation mitigate stockpiling issues.

Hypothetical scenarios
The economic optimisation conducted earlier can be generalised for longer time periods, riding on the entirely hypothetical situation that the Century deposit had the ore to provide for this scenario to pan out. This is useful for the purposes of showing this study’s relevance to the wider mining industry because it reflects on other likely industry situations. It can be seen in Figure 5 that the most optimal SMU size increases with an increasing duration of implementation. This concept can be explained by the time value of money, whereby with an increased time period, a greater demand is placed on the procurement of a product now than in the future. To achieve this, an increased SMU size is required, boosting production rates significantly.

Even more interesting is the potential increase in NPV that could be achieved in this hypothetical situation with the SMU sizes proposed for various time lengths. Figure 6 illustrates that improvements of excess of 10 per cent are quite plausible.

**FINDINGS FROM ANALYSIS**

**Significance to Century Mine**
Most importantly, there is a potential to improve the NPV of the in situ ore by as much as 1 per cent which will equate to a gross pre-tax discounted cash flow improvement of approximately $5 M to $7 M. At a time when the price of zinc
Mining Education Australia – Research Projects Review

A Critique of Selective Mining Unit Sizing at Century Mine to Optimise Productivity with Dilution

Mining is stagnating and foreign exchange rates are volatile, every minute detail of improvement helps to aid the profitability of the Century operation. In hindsight it is easy to say that the Century project could have been even more profitable if this study was conducted sooner, however there is still 30 months of mining that this proposed SMU size can be implemented to help have a positive financial impact. A re-evaluation of the ultimate pit limit could also be investigated, as the mine’s economics do adjust with this altered SMU size. Due to the reasonably exhausted orebody it is expected that any pit limit changes would be minimal.

Significance to the mining industry

On a wider level, this study highlights that there is an equilibrium point between dilution and production. Too often there is the misconception held that ore mining must be as selective as possible and that dilution is an unfathomable outcome. Excessive focus on selectivity hampers mining rates significantly, and serves to delay ore extraction. To put an analogy to the situation, using a hand shovel to extract ore is extremely selective but also very slow. Using a 40 bcm hydraulic shovel to extract ore is particularly quick, but also limited in selectivity. This project uncovers that the current Century operation is being slightly too selective and not extracting ore as quick as it potentially could be. This concept can be universally applied to just about every operation, wherein it is likely that operations are not quite hitting the perfect equilibrium between dilution and extraction rates. Ultimately this project proves that the most pristine and selective of ore mining might not be the most economically optimal method.

Importantly, there are some key downstream effects out of the open pit if the SMU size is adjusted. This serves to make the study even more complicated and, therefore, extensive modelling unique to each site must be conducted if an assessment of this type was to be conducted at various other hardrock minds. One key take away message is that the faster delivery of ore can deliver significant double digit percentage improvements in the NPV of an orebody, and, such a study, if conducted in the feasibility stage of a project, could help to significantly boost a project’s economic credentials.

A further important message from this project is that the most optimal SMU size increases in line with the time period of implementation. This reinforces the time value of money wherein a tonne of zinc metal in the hand today is worth more than one tonne of zinc metal tomorrow, assuming prevailing conditions. As the effect of discounting compounds itself, the time value of money is stressed even more, up to the point in the long-term where it is more beneficial to mine the ore with methods more akin to bulk mining as opposed to a selective manner.

Ultimately, this project shows that often the most significant improvements in profitability can be achieved by revisiting the ground roots of mining and working on optimising the basics. Flamboyant innovation is definitely needed in the industry to maintain technological progression, but it is also important to have an appreciation and understanding of the basic principles of mining to ensure that value is continually added to an operation through day to day decisions.

CONCLUSIONS

The author set about to fill the void in the current state of knowledge by tackling and attempting to solve an issue at Century Mine related to maximising the NPV of the in situ ore that remains to be mined over the next 30 months. The task involved simultaneously optimising numerous critical conditions that impact on a mine site’s economic profitability including mining rates, mining costs, mined grades, mill throughput, mill recovery and milling costs. In the act of conducting this analysis, two key conclusions can be drawn. The mine engineering department at Century should:

1. utilise an SMU size of 9 m × 9 m × 3 m to maximise the NPV of ore at Century;
2. utilise an SMU size of 13 m × 13 m × 3 m to maximise ore stockpiling at Century.

If the first pathway is chosen, an NPV improvement of approximately 1 per cent will be drawn on the remaining orebody, as the ore will be brought forward in the mine life as fast as feasibly possible while also maintaining a tight control on costs. Ore stockpiling will improve by approximately 1400 t/day on average. If the second pathway is chosen, ore stockpiling will improve dramatically at Century, in the order of upwards of 2500 t of ore/day. Unfortunately this is not a cost effective method over the longer term, as the NPV of the orebody will drop by approximately 2 per cent compared to the best case economic outcome.

Ultimately the SMU size should be increased from 75 to 250 bcm in order to maximise and optimise the value of the orebody at Century Mine. It was shown that due to the time value of money, the improvements that can be made through this form of implementation compound and improve upon each other for longer time periods. Hypothetically this form of optimisation could aid in long-term mines being increasingly more profitable from a discounted cashflow perspective.

REFERENCES


Selection of Haulage Fleet at Daisy Milano Gold Mine

B Yiu¹ and A Halim²

ABSTRACT

In the underground mining industry, one of the long-term operational issues involves longer haulage routes as the mine gets deeper with the life of mine. This longer travel route has a great impact on the productivity and cost of the operation. Choosing the best haulage fleet will result in improved productivity and lower cost per tonne of material moved. For this reason, it has motivated Daisy Milano Gold Mine, owned by Silver Lake Resources, to analyse their current Toro 50 Plus trucks against the Atlas Copco MT5010 truck. During a three month trial of both haulage options, trucking parameters were collected from a specifically designed trucking plod. This research used a time motion study to perform productivity calculations and cost analysis of the two haulage options.

Based on the assumptions made in this research, the conclusion is that the Atlas Copco MT5010 should be utilised as the preferred truck option at Daisy Milano. The Atlas Copco MT5010 has a higher average productivity of 254 tkm/hr (tonnes kilometre per hour) compared to the Toro 50 Plus with an average productivity of 160 tkm/hr. In terms of cost ($ per tkm) the MT5010 is the cheaper option by over 35 per cent when compared to the Toro 50 Plus ($0.85 versus $1.31).

INTRODUCTION

Like many other mining operations, tonnage and grade reports are presented daily at the management meeting and daily production in ounces are required to meet the production targets. In terms of production cycle, whilst the grade of the deposit cannot be changed, the quantity of material hauled can be controlled.

The study is based on Daisy Milano, an underground gold mine owned by Silver Lake Resources. The mine is located 55 km south-east of Kalgoorlie in the Mount Monger Goldfields. The current 600 kt annual production is mainly from the Haoma orebody which is steeply dipping and mined using narrow vein stoping methods such as selective longhole avoca stoping and hand-held airleg stoping. Currently, the haulage equipment used are six Sandvik Toro 50 Plus trucks. However, a three month trial of an Atlas Copco MT5010 truck (MT5010) was introduced in 2012 to compare trucking performance with the existing fleet. All underground equipment are second hand and hired from on-site contractors.

The aim of this study was to determine the better haulage option, Toro 50 Plus or MT5010, in terms of truck productivity and cost at Daisy Milano Gold Mine. The objectives were to address the following questions:

- What are the economic costs associated with each haulage option?
- What are the factors that affects trucking productivity?
- What is the difference in trucking productivity of the MT5010 compared to Toro 50 Plus?

- Industry measurement of trucking productivity

The literature review helped identify the current best industry practice of measuring trucking productivity. Even though limited literature on this topic is available, tonnes kilometre per hour (tkm/hr) was confirmed as the industry accepted measurement of trucking productivity (Carrick, Guilfoyle and Robertson, 2002; Robertson, Ganza and Noack, 2005).

METHODOLOGY

The approach as outlined in this design is critical in obtaining conclusions for this project. This project was carried out with the following steps:

- literature review to identify industry measurement of trucking productivity
- data collection with a designed ‘truck operator plod’
- determining travel distance of trucks, using mine planning software Surpac™
- determining truck productivity, setting up and using Microsoft® Excel spreadsheets
- cost analysis with mine site truck cost data
- other considerations to examine such as ventilation requirements and truck maintenance.

Industry measurement of trucking productivity

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As this topic is based on a real operation, applications and case studies were explored to provide guidance to the approach method that others have used to achieve similar outcomes. Case studies involving Kanowna Belle (Kerr, 2002), Gwalia (Savage, 2008) and Hope Downs (Soegri, 2010) mines were used and provided methodology to select haulage fleets.

**Data collection**

During summer employment at Daisy Milano Gold Mine from December 2012, a ‘truck operator plod’ was generated and utilised by day and night shift truck operators to record parameters as part of the trucking cycle. The recorded parameters include the times and locations of when the truck is loaded and other comments. A copy of a trucking plod sheet is shown in Figure 1. The plods were utilised by about half of the truck operators, many simply just forgot about them or ignored them. The plod data collected was in the period of 4 December 2012 to 1 January 2013 and 1 February 2013 to 19 February 2013, totalling 44 days. The plod data was collected and collated in a database.

It is in this part that the greatest amount of error can be introduced, with the plods easily filled out incorrectly. Plods found to be incorrectly reported were excluded from the analysis in order to improve the accuracy of results.

**Determining travel distance of trucks**

At Daisy Milano Gold Mine, there are currently no design files to provide the centreline for the decline from the portal down to the bottom of the mine. There are survey pickups of each of the levels and separate sections of the decline in string files. All of the string files were merged, removing unnecessary access drives and generated one string only for the entire decline as shown in Figure 2.

**Determining truck productivity**

After all the data was entered and Surpac™ work created, the trucking productivity calculations could begin. The cycle time of each truck load was then calculated using an Excel spreadsheet.

Following this the truck productivity in tkm/hr was determined. From this information, comparison of trucking productivity of every hour was analysed.

**Cost analysis**

To perform discounted cash-flow analysis (DCF), the truck costs associated are required. The cost data required to undertake simple DCF analysis should include:
• truck rental cost
• truck operating cost
• truck fuel cost
• maintenance cost, including parts.

The DCF analysis will be based on the net present cost (NPC), this allows the factor of time to be incorporated into the analysis of the cost information. By completing a DCF analysis, the NPC would provide an estimate of the current cost and the cost to replace Toro 50 Plus with the MT5010. Many mining operations are reluctant to provide any cost data due to the sensitive nature of the information. However, Silver Lake Resources were willing to provide most of the cost information, regarding this trucking project only.

DATA INPUT

Operational assumptions
At Daisy Milano, the operation is divided into two 12-hour shifts day and night, every 24 hours. Day shift begins from 6.00 am until night shift commences at 6.00 pm until the start of the next day shift.

According to the manufacturer specification handbook of the haulage options, both trucks have a designed payload of 50 t. However to perform trucking reconciliation at Daisy Milano, tonnages of truck load are not weighed and the payload of each truck load is taken as a constant of 32 t. This value is agreed by the engineers and geologists on-site given past data and reconciliations. This value is critical in the calculation of the productivity of the truck as described later.

Due to the limited decline access and pathways, both truck options are assumed to have used the same travel routes to reach the same destinations.

From the existing Toro 50 Plus fleet at Daisy Milano, five trucks were utilised in this study. They are named as ‘UT001, UT002, UT006, UT007 and UT008’. During the trial period, the MT5010 was named as ‘UT010’.

Trucking plods
The trucking plod, as shown in Figure 3, when filled out properly, represents the time motion of the truck. It provides the time and location when the truck is filled with material to when this material is dumped onto either the run of mine (ROM) or into the backfill stope.

The schematic of the two haulage truck cycles, to ROM or to the backfill stope is shown in Figures 4 and 5, respectively. For this study, the path of the main truck cycle concerned was the travel of the truck after it has been loaded and prior to dumping of material. The difference in loading and dumping time for the two options considered is negligible based on site observation that both trucks performed these tasks in a similar time frame. Therefore it has not been included in the analysis. According to the truck specifications given by the respective manufacturer’s handbooks and based on site observations, both trucks achieve very similar performance whilst travelling empty.

Travel time
Using the information entered into Excel, the travel time of each load was calculated from the time at which the truck was loaded and when the truck dumped the material. Travel time information was used as a direct indicator of production as a constant payload of 32 t was utilised. It can be summarised that travel time is inversely proportional to production, when comparing the same travel routes with constant payload. For example, Truck ‘X’ travel time to complete Route ‘A’ is less

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FIG 3 – Filled out trucking plod.
than that of Truck ‘Y’ on the same route. Therefore, Truck ‘X’ is more productive.

RESULTS AND DISCUSSION

Productivity

From the trucking data entered into Excel, calculations to obtain productivity in tkm/hr were performed. As discussed previously, tkm/hr is an approach used in this study to analyse productivity and uses the following formula:

\[
tkm/hr = \frac{t \times km}{hr}
\]

where:
- \( t \) = payload in tonnes
- \( km \) = distance travelled in kilometres
- \( hr \) = time taken in hours

The assumption of the 32 t payload constant was utilised in this study for all of the truck loads. Travel time and distance are utilised as discussed previously.

Toro 50 Plus

The tkm/hr for all Toro 50 Plus truck loads during the period was calculated and categorised into the 12 working hours/shift. The average load tkm/hr for each hour was graphed for both day and night shift as shown in Figure 6. In Figure 6, it is evident that the number of loads was low during three main hours of the shift, first hour, sixth hour and the last hour. This low number of loads would be a reflection of the start of shift, crib time and end of shift, respectively. It is also worth mentioning that the number of loads obtained for the first hour and the last hour is very low, 15 and one respectively. This would provide a less accurate indication of the productivity at these hours compared with other hours with many loads. The trend for both day and night shift productivity is very similar and both provide the result of 160 tkm/hr as the average productivity of Toro 50 plus.
The same analysis was performed for the MT5010, calculating the productivity per hour, as shown in Figure 7. The same comment can be made in regards to the accuracy of the calculated productivity of the first and last hour about the MT5010, given that only data from two loads were recorded during each of these hours. The trend for both day and night shift, like Toro 50 Plus, are very similar, peaking between the first hour until sixth hour for crib time and between the seventh hour after crib time and the end of shift (last hour). It can be seen from the ninth hour to the eleventh hour that the number of loads recorded from day shift is significantly greater than that of night shift. However the productivity at these hours were lower for day shift than night shift. This is a good example of how this design to analyse productivity of the truck is not biased by a greater number of loads recorded. The average productivity of the MT5010 calculated was 254 tkm/hr.

**Comparison**

To compare the productivity of each of the truck options, the average productivity from all the loads was analysed. In Figure 8, the productivity in tkm/hr for both Toro 50 Plus and MT5010 truck for each hour of the day is plotted, with an average line representing the productivity of each of the options.

MT5010 has a higher average productivity of 254 tkm/hr compared to Toro 50 Plus with an average productivity of 160 tkm/hr. This shows that the MT5010 is more productive than the existing Toro 50 Plus. Based on productivity analysis alone, the MT5010 is the preferred haulage option.

**Other considerations**

**Ventilation requirement**

According to the Western Australian Mines Safety and Inspection Regulations 1995 (WA Government, 1995), Regulation 10.52 (7):

A diesel unit referred to in subregulation (5)(b) must have a ventilation volume rate of not less than 0.06 cubic metres per second per kilowatt of the maximum rated engine output specified by the manufacturer for the fuelling and timing configuration at which the engine has been set.

This is based on the limited data available and that for both truck options the subregulation (5)(b) applies to all second hand equipment.

From the individual manufacturer’s specification for each of the trucks, it is to note that the MT5010 has an additional 93 kW of engine power than that of Toro 50 Plus. Whilst this additional power is expressed as faster truck speed, additional ventilation requirements as a result will need to be considered.

The additional ventilation requirement was calculated as follows.

\[
\text{Total ventilation requirement (m}^3/\text{s}) = \text{Airflow per kW (m}^3/\text{s/kW) \times total engine power (kW)} = 0.06 \times 93 = 5.58 \approx 6 \text{ m}^3
\]

From the above, an additional 6 m³ of ventilation will be required per truck if a MT5010 is to replace a Toro 50 Plus truck. This additional ventilation should be included in the ventilation design and monitored as per normal ventilation.

**Maintenance**

Maintenance data is obtained for two different reasons, performance evaluation and measurement of reliability. By examining the maintenance information on the trucks, information that is incorporated as ‘downtime’ can be examined in detail. For example, downtime for a truck can be due to a faulty part not being repaired properly. This would show up as many maintenance repairs and problems arise due to that certain part. The reliability of the truck, whether it is Toro 50 Plus or MT5010, can be evaluated given the...
maintenance downtime that it incurs. It can be expressed that the less downtime due to mechanical failure means that the more reliable the equipment is.

For this study, no maintenance data could be obtained from the mine site and therefore was not considered.

**Financial analysis**

In this part of the study, the costs associated with the truck options were analysed and compared. At Daisy Milano, all of the underground equipment including trucks are hired, therefore capital cost is not considered. To operate the truck for one month, the cost for the truck operators are the same to both options of $55,000/month.

**Rental cost**

The cost associated with Toro 50 Plus truck is based on the flat rental rate of $70,000/month. This flat rental rate includes the base rental of the truck and operating cost minus fuel and maintenance cost, regardless of the number of hours that the truck is operated for.

The cost associated with the MT5010 is split into fixed and operating cost. The fixed cost of $34,000/month is considered as the ‘base rate’ or the cost to rent the truck, regardless of the number of hours that the truck completes. The operating cost is charged at $86/hour and is based on the number of ‘operating hours’ that the truck accumulates over the monthly periods. For budgeting reasons and during the use on-site during the trial period, 420 operating hours/month is utilised. The total cost associated with the MT5010 can be summarised as $70,000/month (as budgeted).

**Fuel cost**

Diesel fuel cost analysis was performed in this study for both of the haulage options. Due to the limited fuel data obtained, fuel data in litres was only available for certain times of the study period. Operating hours of this period was assumed to be a ratio of the monthly operating hours including utilisation and availability. Fuel consumption per hour for this period was calculated from the fuel consumed divided by the operating hours. This can be simplified as follows:

\[
\text{Fuel consumption/hour (L/hr)} = \frac{\text{Fuel usage (L)}}{\text{Operating hours (hr)}}
\]

For calculating fuel cost, the cost of diesel was obtained from the mine. The full price of diesel is $1.38/L, however the mine only pays $1.03. This is due to the deduction of 35 cents from off-road use government rebate. Fuel cost per month was calculated as follows.

\[
\text{Fuel cost per month ($) = Cost of fuel per litre ($/L) \times fuel consumption (L/hr) \times operating engine hours (hr)}
\]

Fuel is a large component of the haulage cost and was analysed in this study for the two options. The recorded number of operating engine hours for Toro 50 Plus and MT5010 obtained was different during the study period, due to the biased nature of the haulage fleet trial. To ensure that the normal cost of fuel per month was more accurately estimated, a constant 420 operating engine/month was utilised. Using $1.03 as the cost of diesel, a summary of the monthly fuel cost for each truck option is summarised in Table 1.

**Discounted cash-flow**

By using NPC analysis, the time value of money can be included. The net present values can then be compared as discounted cash-flow analysis. In this study, no capital costs are associated and therefore NPC analysis could not be included.

### TABLE 1

<table>
<thead>
<tr>
<th>Truck</th>
<th>Fuel consumption (L/hr)</th>
<th>Fuel usage per month (L)</th>
<th>Cost per month</th>
</tr>
</thead>
<tbody>
<tr>
<td>MT5010</td>
<td>47</td>
<td>19,740</td>
<td>$20,332</td>
</tr>
<tr>
<td>Toro 50 Plus</td>
<td>42</td>
<td>17,640</td>
<td>$18,169</td>
</tr>
</tbody>
</table>
Cost comparison

By comparing the truck costs of two options, $70 000 for Toro 50 Plus and $70 120 for the MT5010, the cost difference of $120/month is almost negligible in the context of haulage cost. The break-even cost between the two options occurs when the MT5010 costs the same flat rate as Toro 50 Plus truck of $70 000/month. With analysis as shown in Figure 9, the break-even cost would occur when the MT5010 accumulates about 418 operating hours/month.

Figure 9 also shows that the cost of the MT 5010 will be less than that of Toro 50 Plus if the operating hours are less than 418 hours/month. Operating hours per month far above the 418 break-even hours could see that Toro 50 Plus would be preferred, given its lower cost.

The total monthly cost of the two options is the sum of the rental cost and fuel cost for each month as summarised in Table 2. This is calculated by the formula:

\[ \text{Total monthly cost} (\$) = \text{rental cost} (\$) + \text{fuel cost} (\$) \]

To compare the two options with different productivity, the cost per tkm was calculated. The cost per tkm is calculated from the total monthly cost divided by the total tkm per month. Using the assigned operating hours of 420 engine hours/month, the total tkm per month was calculated. A summary of the cost per tkm is shown in Table 3.

Comparison

Whilst the sum of the rental cost and fuel cost provides a monthly cost of the truck, it does not provide any information of how productive the truck is in the month. In terms of rental and fuel cost per month, Toro 50 Plus truck is $2273 cheaper than that of MT5010 ($88 169 versus $90 442). However, using productivity data as obtained, the cost per tkm was determined using the assigned 420 engine hours/month. The result concludes that MT5010 is the cheaper option by $0.46/tkm when compared to Toro 50 Plus ($0.85 versus $1.31). This is a 35 per cent cost savings/tkm when MT5010 trucks are used.

It is advised that further study and analysis using field maintenance cost data should be undertaken. This will provide a more comprehensive and reliable cost analysis for the two options.

CONCLUSIONS

This study has found an indication that MT5010 is the better haulage option, over the existing Toro 50 Plus, in terms of truck productivity and cost at Daisy Milano Gold Mine.

MT5010 has a higher average productivity of 254 tkm/hr compared to Toro 50 Plus with an average productivity of 160 tkm/hr. This represents MT5010 as the more productive haulage fleet over the existing Toro 50 Plus. Based on the productivity analysis alone, the MT5010 is the preferred haulage option.

The financial analysis based on 420 monthly operating hours concludes Toro 50 Plus has a $2273 cheaper monthly rental and fuel cost than the MT5010 ($88 169 versus $90 442). Based on rental and fuel cost alone, Toro 50 Plus is the preferred haulage option.

However, analysis of productivity and economics of the truck separately does not provide an indication of how productive the truck is and how much that productivity would cost. Therefore an economic value based on productivity was determined, as cost per tkm. The result concludes that MT5010 is the cheaper option by $0.46/tkm when compared to Toro 50 Plus ($0.85 versus $1.31).

Based on the assumptions of this study, as stated previously, the project for Daisy Milano Gold Mine concludes that:

![FIG 9 – Cost per month comparison between Toro 50 Plus and MT5010 truck at different number of operating hours.](image-url)
• MT5010 truck compared to Toro 50 Plus is more productive and more cost effective per tkm
• MT5010 should be utilised as the preferred haulage option, replacing the existing Toro 50 Plus.

ACKNOWLEDGEMENTS
This research study would not have been possible without the support and assistance of several people and parties. The authors would like to thank Daisy Milano Gold Mine for providing the data necessary for this study, in particular acknowledges the assistance provided by Mr Nicholas Chernoff for his time and assistance. The authors would also like to thank Ms Kirstan Lee for her support throughout the study.

REFERENCES


Soegri, R S, 2010. Analyse the availability and utilisation of the production fleet at Hope Downs, BEng thesis (unpublished), Western Australian School of Mines, Curtin University, Kalgoorlie.

## APPENDIX 1 – PROGRAM 2013 MEA STUDENT CONFERENCE

**Date:** 2013 MEA Student Conference, 21 October 2013  
**Venue:** School of Mining Engineering, University of New South Wales  
Rm G51, Old Main Building, Kensington campus

<table>
<thead>
<tr>
<th>Time</th>
<th>Session</th>
<th>Event</th>
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<tbody>
<tr>
<td>08:00</td>
<td>Registration</td>
<td>Arrival &amp; Conference Registration</td>
</tr>
<tr>
<td>08:30</td>
<td>Opening</td>
<td>Welcome &amp; Opening of the Symposium by P. Hagan &amp; P. Dowd</td>
</tr>
<tr>
<td>08:45</td>
<td>Session 1</td>
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<tr>
<td>08:45</td>
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<td>Eric Lye (UNSW): The effects of particle stabilisers on bubble generation, coalescence and breakup</td>
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<tr>
<td>09:00</td>
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<td>Charles Byrne Smith (UQ): A Critique of SMU sizing at Century Mine to optimise productivity with dilution</td>
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<tr>
<td>09:15</td>
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<td>Mark Kong (WASM): Aboriginal land rights agreement – A Western Australia case study</td>
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<tr>
<td>09:30</td>
<td></td>
<td>Graham Ball, Jesse Clark, Michele Gifford, Rajendra Rathod (UA): Probabilistic stability analysis of rock excavations</td>
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<tr>
<td>09:45</td>
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<td>He Ren (WASM): Using clustering method for block aggregation in open pit mine planning</td>
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<td>10:00</td>
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<td>Nicholas Anthony Butel (UQ): Estimation of in-situ rock strength using geophysics</td>
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<tr>
<td>10:15</td>
<td></td>
<td>Prudence Grace Fischer (UNSW): Identify and optimise parameters influencing blast performance within basal till material with groundwater inflow</td>
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<td>10:30</td>
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<td>Morning Tea in Room G38</td>
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<td>Tyler Eastan, Brendan Connell, Cassandra Lazo Olivares, Natalie Yfantulis (UA): The study of the strength and deformability of rocks under random and</td>
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<tr>
<td>11:05</td>
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<td>Jamie Paul Jonsgebield (UNSW): Coal Project Valuation – A Hedonic Pricing Approach</td>
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<tr>
<td>11:20</td>
<td></td>
<td>Gareth Thomas Swal (UQ): Analysis of heat reduction opportunities for the George Fisher north expansion project decline</td>
</tr>
<tr>
<td>11:35</td>
<td></td>
<td>Stefan Andrzej Skorut (UNSW): Optimising cut-off grade strategy for seabed massive sulphide operations</td>
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<tr>
<td>11:50</td>
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<td>Yu Yang (WASM): Surface fauna as a new mining exploration method – fact or fiction?</td>
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<td>Paul Bryson, Andrew Mount-Smith, Xihong Wang, Daowen Zhang (UA): Deep drilling performance estimation using rock mass characterisation</td>
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<tr>
<td>12:20</td>
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<td>Jake Lee Small (UNSW): Time benefit analysis of through-seam blasting</td>
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<td>12:35</td>
<td>Lunch</td>
<td>Lunch in the Chancellors Garden – Fountain</td>
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<td>13:45</td>
<td>Session 3</td>
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<tr>
<td>13:45</td>
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<td>Yanchao Qu (UQ): Workforce hazard mapping and its optimum support design for North Goonyella Coal</td>
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<tr>
<td>14:00</td>
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<td>Leigh Bowen, Matthew Harding, Ashlee Kiss, Samuel Locken (UA): Deep open-pit mining – Rock haulage Optimisation</td>
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<td>14:15</td>
<td></td>
<td>Robert James Lecce (UQ): Optimisation of waste-dump lift heights for pre-strip operations at Meandu Mine, Queensland</td>
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<td>14:30</td>
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<td>Kirsten Lee (WASM): Comparison of methods for opening initial voids in longhole open stoping blasting</td>
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<tr>
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<td>Daniel Broxwell (WASM): Using Artificial Neural Network (ANN) to predict stope overbreak at Platinum Gold Mine</td>
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<td>Jingyu Chen, Menghua Gu, Jiachang Liu, Ke Zhang (UA): Simulation and Animation of a Surface Gold Mine</td>
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<tr>
<td>15:15</td>
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<td>Afternoon Tea &amp; Meeting of the Judging Panel</td>
</tr>
<tr>
<td>15:30</td>
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<td>Announcement of Prizes &amp; Closing of the Conference</td>
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## APPENDIX 2 – PRESENTERS AT MEA STUDENT RESEARCH CONFERENCES, 2009–2013

<table>
<thead>
<tr>
<th>Year</th>
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<th>Student presenter(s)</th>
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<td>Paul Bryson</td>
<td>Deep drilling performance estimation using rock mass characterisation</td>
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<td>Jingyu Chen</td>
<td>Simulation and animation of a surface gold mine</td>
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<tr>
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<td>Jesse Clark</td>
<td>Probabilistic stability analysis of rock excavations</td>
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<tr>
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<td></td>
<td>Mathew Harding</td>
<td>Deep open-pit mining: rock haulage optimisation</td>
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<tr>
<td></td>
<td></td>
<td>Cassandra Lazo Olivares</td>
<td>The study of the strength and deformability of rocks under random and systematic cyclic loading</td>
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<tr>
<td></td>
<td>UNSW</td>
<td>Prudence Fischer</td>
<td>Identify and optimize parameters influencing blast performance within basal till material with ground water inflow</td>
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<td>Jamie Jongebloed</td>
<td>Coal project valuation – a hedonic pricing approach</td>
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<td>Eric Law</td>
<td>The effects of particle stabilisers on bubble generation, coalescence an breakup</td>
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<tr>
<td></td>
<td></td>
<td>Stefan Skorut (equal third prize)</td>
<td>Optimising cut-off grade strategy for seabed massive sulphide operations</td>
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<td>Jake Small (second prize)</td>
<td>Time benefit analysis of through-seam blasting</td>
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<td>UQ</td>
<td>Nicholas Butel (fourth prize)</td>
<td>Estimation of in situ rock strength using geophysics</td>
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<td>Yanchao Hui</td>
<td>Workface hazard mapping and its optimization support design at North Goonyella</td>
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<td>Robert Lucas</td>
<td>Optimisation of waste-dump lift heights for pre-strip operations at Meandu Mine, Queensland</td>
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<td>Charles Smith</td>
<td>A critique of SMU sizing at Century Mine to optimize productivity with dilution</td>
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<td>Gareth Steel</td>
<td>Analysis of heat reduction opportunities for the George Fisher north extension project decline</td>
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<td></td>
<td>WASM (Curtin)</td>
<td>Daniel Boxwell</td>
<td>Using artificial neural network (ANN) to predict overbreak at Plutonic Gold Mine</td>
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<td></td>
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<td>Mark Kong</td>
<td>Aboriginal land rights – a Western Australia case-study</td>
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<tr>
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<td></td>
<td>He Ren (equal third prize)</td>
<td>Using clustering method for block aggregation in open pit mining</td>
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<td></td>
<td>Kirstan Lee (first prize)</td>
<td>Comparison of methods for opening initial voids in longhole open-stop blasting</td>
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<td>Yu Yang</td>
<td>Surface fauna as a new mining exploration method- fact or fiction?</td>
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<td>Multi-objective optimisation of mining-metallurgical systems</td>
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<td>Paul Carmichael</td>
<td>An investigation into semi-intact rock mass representation for physical modelling of block caving mechanics zones</td>
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<td>McLeod McKenzie</td>
<td>Prediction and modelling of blast vibration and its effects at Glendell Colliery</td>
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<td>Tim Graham (third prize)</td>
<td>Strategic project risk management for an emerging miner</td>
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<td>Vanessa Collins</td>
<td>Maximising production rates at Brockman 4 by minimising truck delays at the crusher</td>
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<td>Casey Costello (first prize)</td>
<td>Grizzly modifications at Ridgeway Deeps block cave gold mine</td>
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<td>Brenton Goves (second prize)</td>
<td>Continuous surface miner operations at Fortescue</td>
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<td>Comparing neural networks with JRSimblast prediction model in purpose of optimal ground vibration induced by blasting</td>
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<td>Vadim Strukov</td>
<td>A comparative study of truck cycle time prediction methods</td>
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<td>Weilin Wang</td>
<td>ANSYS for stress analysis of underground structures</td>
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<td>James Boffo</td>
<td>Acoustic emission monitoring of impregnated diamond drilling for deep exploration</td>
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<td>Developing a truck allocation model for Bengalla Mine</td>
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<td>Brendan Murphy</td>
<td>Development of an underground coal scheduling and simulation program</td>
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<td>Colin Thomson</td>
<td>Stoping analysis of Golden Grove under high stress conditions</td>
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<td>2010</td>
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<td>Adam Schwartzkopf and Daniel Hardeya</td>
<td>Comparisons between three-dimensional yield criteria for fractured and intact rock</td>
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<td>James Tibbett</td>
<td>Failure criteria of three major rock types affecting Ridgeway Deeps</td>
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<td>An investigation into final landform criteria required for a safe, stable, sustainable and non-polluting landform in the Bowen Basin</td>
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<td>WASM (Curtin)</td>
<td>Nathan Colli</td>
<td>Analysis of the principles for sound waste dump design and placement</td>
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<td>Kali Dempster</td>
<td>Quantifying time-dependent rock behaviour at Ridgeway Deeps block cave operation</td>
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<td>Michael Baque</td>
<td>Optimisation of coal scheduling at Dawson Mine</td>
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<td>Kieran Rich (first prize)</td>
<td>Paste fill for stope stability at the Challenger Gold Mine</td>
</tr>
</tbody>
</table>
# Author Index

<table>
<thead>
<tr>
<th>Author</th>
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</tr>
</thead>
<tbody>
<tr>
<td>Ball, G H</td>
<td>1</td>
</tr>
<tr>
<td>Bastian, T J</td>
<td>7</td>
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INTRODUCTION

The aim of this research is to assess in situ block size distribution using the discrete fracture network (DFN) approach for the proposed OZ Minerals’ Carrapateena block caving project. The financial viability of any block caving operation is determined by the ability of the orebody to cave naturally once the fracture propagation has been initiated. It is important to identify at the mine design stage the in situ block size distribution as this will determine the need for preconditioning and the suitability of drawpoint and layout design.

The literature review conducted revealed that DFN has major applications in the field of fluid flow through fractured reservoirs (eg Dershowitz, La Pointe and Doe, 2004; Xu, Dowd and Wyborn, 2013). Some researchers claimed their focus on the use of DFN to conduct block size analysis for mining applications, but there were in fact very limited related publications available. This represents a unique opportunity to apply the basic principles of DFN and to develop our own method for the analysis of block size distribution that can then be used for our case study.

A comprehensive review of previous research in this field has facilitated the development of a framework in this project to assist the block size distribution analysis within a jointed rock mass, with particular application to OZ Minerals’ proposed Carrapateena Mine. The developed tool provides an analytical approach that can be used in the feasibility study stage to enable the comparison between the proposed design parameters and the expected block sizes. The need and suitability of preconditioning on the rock mass can also be evaluated with the help of this tool.

IN SITU FRACTURES AND BLOCK CAVING

The success of block caving mining depends heavily on the ability of the rock mass to cave naturally under the action of gravity. Ideally the caved blocks should be small enough to pass through the ore drawing structures easily. The size distribution of the caved blocks is mainly determined by the existing fractures in the rock mass when it caves. To assess
the performance of block caving these fractures have to be modelled first. The common approach to model the spatial distribution of rock fractures is to use DFNs. A brief overview of previous research in DFN is presented further to provide a background for this research, including a review of current techniques and strategies being used for the assessment of block size distribution of a fractured rock mass.

Discrete fracture networks

DFN is a stochastic modelling technique that is used to represent the geological and various other properties of a fracture system within a certain body of a rock mass using statistical methods (Moffitt et al., 2007). This technique involves constructing a number of statistically equivalent DFNs based on models derived from field observations. These observations include fracture size (fracture trace length), fracture orientation as well as fracture intensity and they are conducted either by borehole fracture mapping or scanline or window survey of exposed rock faces. The main objective of constructing a DFN model is to represent the natural fractures in a statistical framework that can then be used in numerical simulations (Jin, 2007).

In the past, DFN models have been applied in various fields to simulate the transport of fluids through fractured rock mass (eg Xu, Dowd and Mohais, 2012). In recent developments with regard to the ability to model rock mass behaviour, DFN models have been implemented in the mining industry to analyse fragmentation and wedge or key block formation for stability assessment of slopes and other rock excavations (Rogers and Moffitt, 2006; Hadijgeorgiou and Grenon, 2005). DFN is generally regarded as a preferred modelling technique as it is capable of representing the rock mass more realistically so that an analysis closer to reality can be achieved (Rogers and Moffitt, 2006).

However, very limited research has been found in using DFN modelling directly to help the design process in fractured rock masses, either for a rock excavation structure or for a mining operation. This research will explore the opportunity to expand DFN modelling as an analytical tool to help the design of block caving operations at the feasibility study stage of the mining operation.

Mining by block caving

Block caving is becoming an increasingly popular mining method because of its features of low operation cost and high production rate. The method can be used for low-grade massive orebodies which are becoming more common as high-grade deposits are getting scarce and more difficult to find (Ford, Pine and Flynn, 2007). Block caving mining technique takes advantage of the natural fractures as well as the fractures created by the preconditioning treatment in the rock mass. The rock caves gradually due to gravity once an undercut, or void, has been driven underneath the orebody (Laubscher, 2000).

The critical aspects of block caving include caving initiation, propagation and fragmentation. The fragmentation of the rock initially occurs due to the extension of fractures that form blocks once the undercut is made. Drawpoints are constructed underneath the orebody and act as collection points for the fragmented ore. Therefore it is important to understand the fracture network of the orebody and assess the potential block size distribution during the caving operation.

Pretreatment methods

The success of the caving process is highly dependent on the existing fractures as mentioned above and if the rock mass does not contain sufficient in situ fragmentation, a pretreatment technique can be used. Various pretreatment techniques have been studied and implemented in the industry (Catalan et al., 2012; Jeffery, 2000; Laubscher, 2000).

Pretreatment of the rock mass typically is carried out before caving is initiated and is done in order to change the characteristics of the rock mass and to induce fractures that will enhance the caving process (Catalan et al., 2012). The technique commonly includes hydraulic fracturing, blasting and/or a combination of both. Hydraulic fracturing changes the rock mass characteristics by injecting fluid into packed boreholes to create additional fractures in the rock mass. Blasting also changes the rock mass characteristics by localised damages due to the explosion. The combination of these two pretreatment techniques is known as ‘intensive preconditioning’. The success of a pretreatment program depends on the existing fractures and the borehole design (including pattern and spacing) used (Catalan et al., 2012).

Software packages

Multiple computer programs can be used to analyse removable blocks and wedge stability in rock excavations and those suggested in the literature include Dips (by Rocsience Inc), FracSim3D (Xu and Dowd, 2010), FracMan® (by Golder Associates Inc) and Resoblok (Baroudi et al, 1990). For this study, the programs that were chosen to create DFNs and to conduct the analysis were FracSim3D and FracMan because of their accessibility. A comparison will be made between the results obtained from these two programs.

FracSim3D is a software package that can be used for stochastic simulation of fractures within rock mass in either two-dimensional plane or three-dimensional volume (Xu and Dowd, 2010). DFNs are created by the program through the use of Monte Carlo sampling of the probability density functions of fracture properties. The program also allows the statistical analysis of the DFN through the use of sampling planes, windows, and scanlines (Xu and Dowd, 2010).

FracMan is another software package that can be used for the generation and storage of stochastic simulations of fracture networks. The program also allows the incorporation of raw fracture data to derive parameters that are required to build DFNs. The main advantage of FracMan is its ability to analyse block volumes that are formed within the DFN using the BlockSize Analysis (by Golder Associates Inc) add-on and the ability to analyse rock wedges that form on excavation boundaries and their volumes using the RockBlock (by Golder Associates Inc) add-on. FracMan has been cited numerous times in various studies (Dershowitz, La Pointe and Doe, 2004; Tollenaar, 2008; Starzec and Andersson, 2002; Merrien-Soukatchoff, Korini and Thoraval, 2011).

CARRAPATEENA PROJECT

OZ Minerals’ Carrapateena Project, located in South Australia, is an iron oxide copper-gold (IOCG) deposit that has been extensively studied based on a 75 km drilling program undertaken at the site between 2007 and 2009. The deposit is hosted in a very strong hematite breccia complex that is overlain by 500 m of a sedimentary sequence. Copper is mostly in the form of chalcopyrite (OZ Minerals, 2013). The deposit can be split into three main sections: the main lens, the north-east lens and the north-west lens. The top of the
cylindrical deposit is located 470 m below the surface and the mineralisation extends over a vertical height of approximately 1000 m (OZ Minerals, 2013).

The considered options for mining the deposit included sublevel caving, sublevel open stoping and block caving. SRK Consulting was contracted by OZ Minerals to conduct a block caving scoping study for the Carrapateena Project. Within the scoping study, two methods were considered: one being a single-lift block caving option and one with a double-lift block caving option. It was concluded by SRK Consulting that the double-lift option was the optimal design for mining the Carrapateena deposit because of the vertical extent of the orebody and geotechnical considerations. The double-lift option involves splitting the deposit into two separate caving operations with the maximum height of 500 m (Figure 1).

The design for the extraction level and the drawpoints for each lift were also outlined in the study. To ensure that the blocks can be extracted from the drawpoints, the maximum size of each caved block needs to be 39 m following the primary fragmentation and 17 m after secondary fragmentation.

METHODOLOGY

To generate the stochastic fracture models a large amount of data needs to be collected and analysed in order to derive the parameters required. Two approaches are then used for block size distribution analysis of the rock mass. The first method involves the construction of fracture networks using FracSim3D and exporting the generated fracture network data for analysis with a visual basic for applications (VBA) macro developed in this research. The second method uses FracMan to generate DFNs from the same fracture data and then to conduct the assessment using the inbuilt block analysis tools. The process of data acquisition, analysis and the construction of the fracture models is described in details in the following sections.

Analysis of existing fracture data

Data is needed to derive statistical distribution parameters to be used as inputs for the DFN model. In general, fracture data can be collected using various methods such as scanline surveys, areal (window) surveys, oriented core fracture mapping and borehole imaging.

In our study, the first step taken for this project is a review of company technical reports to obtain the relevant geotechnical data for the Carrapateena site. This involved the acquisition of available geotechnical data from OZ Minerals which was primarily oriented core fracture mapping. This data set was processed with the relevant fracture set information being transferred into a spreadsheet for further analysis. The dip and dip direction of each of the identified fractures are calculated using the recorded alpha and beta angles (explained further) along with the trend and plunge recorded for the borehole.

Classification of fracture sets

Given that the deposit is at some depth below the surface without any of the lithology of interest exposed at the surface, data collection methods such as areal surveys cannot be used. Instead the data relating to fracture orientation and density were sourced from geotechnical surveys undertaken by OZ Minerals and other previous leaseholders. As mentioned, these surveys mainly involved oriented diamond drilling cores which can provide critical information about fracture location and orientation. Given that approximately 500 m of overburden consisting of sedimentary lithology covers the orebody of interest, boreholes within this section were omitted from the fracture assessment. Therefore the final fracture model was constructed from the data of 43 boreholes over a total length of 45 km. Note: the omission of the overburden in the analysis could potentially be an issue as the overburden could be the potential sources of unexpected cave propagation and ore sterilisation. Further assessment of its influence on the caving operation is recommended.

Fracture locations were found by calculating the corresponding depth of the borehole at the point where a fracture has been observed in the core sample. Similarly fracture dip direction and dip angle are calculated based on the recorded alpha and beta angles from the survey plus the trend and plunge of the borehole. The alpha angle recorded in the fracture mapping is the acute angle between the core axis and the long axis of the ellipse, between 0–90°. The angle which forms between the reference line along the core and the ellipse apical trace is the beta angle, measured in a clockwise sense (0–360°).

Unfortunately fracture persistence can not be measured by oriented cores and in the absence of a scanline or an areal survey, four lognormal distributions were used for the DFN models as lognormal distribution is regarded as the most common type of distribution for rock fracture sizes (Xu and Dowd, 2010). Following the advice from the geotechnical department of OZ Minerals, a mean fracture size of 4.0, 7.5, 10.0 and 15.0 m were tested. Each of these mean fracture sizes were modelled with an identical standard deviation of 1.82 m.

Fracture intensity calculation

Fracture intensity is an important input parameter needed to construct a DFN model. After the fracture set classification, the data was then used to calculate a linear fracture intensity (or spacing) for each individual fracture set. This process...
involved initially partitioning the data into their respective identified fracture sets. A correction was then made to the recorded downhole depth to convert it to the true vertical depth using the plunger angle of the borehole. Histogram analysis was then used to determine the number of fractures interpreted within 50 m intervals from 500–2500 m. The number of fractures was then divided by the bin width to generate the plot of linear fracture intensity (# fractures/m) against depth. A further correction is needed to the calculated figures that must be divided by the number of boreholes where the data was taken from for each interval. Failure to carry out this step would result in the linear fracture intensity being grossly overestimated. As was anticipated, the linear fracture intensity decreased with depth and the actual relationships vary depending on fracture sets.

In order to calibrate the fracture model generated in FracSim3D, the relationship between one-dimensional fracture intensity and three-dimensional fracture density needed has to be found for each fracture set. This relationship was found by generating each fracture set with varying volumetric density values of 0.0001, 0.001, 0.01 and 0.1 (# fractures/m$^3$) and taking scanline samples using FracSim3D’s plane sample function. This involved taking three vertical scanline samples within two planes both oriented parallel to the z-axis, one perpendicular to the x-axis and one perpendicular to the y-axis.

In order to minimise the sampling influence due to edge effects, these scanlines were 36 m in length and offset by 15.0 m. The average number of fractures intercepted by the scan lines was then divided by the length of the scan lines to give the linear intensity (# fractures/m) for each set of volumetric densities. This provides the relationship which in turn is used to interpolate the volumetric density for the measured linear intensity required as the input for each fracture set at a particular depth for the DFN model. Figure 2 shows a calibration example for Fracture Set 3.

**Construction of fracture models**

Once the fracture orientation properties and intensities had been sourced and corrected the construction of the fracture models using both FracSim3D and FracMan could be conducted.

**FracSim3D**

To assess the influence of models on the analysis results, a total of 12 models were generated. Each model replicates an individual scenario using the four different mean fracture sizes and fracture intensities for each individual fracture set calculated at the top, middle and bottom of the formation. An example of the generated fracture model is shown in Figure 3.

**FracMan**

The volumetric fracture intensity found in FracSim3D was then input into FracMan for all 12 models. This process generated statistically equivalent models to those generated in FracSim3D. An example of the generated FracMan fracture model is shown in Figure 4. The block size analysis was then performed using the Sybil-Frac algorithm available in the FracMan software. The Sybil-Frac algorithm overlays fine grid cells throughout the volume of investigation. The number of grid cells present within a closed polyhedron (ie a formed block) are counted and then multiplied by the grid cell volume to determine the volume of each polyhedron.

As part of the case study, the developed VBA code was used to analyse wedge volumes intersecting a selected plane within the model and the results were compared with those calculated from FracMan. The relationships between block...
volume and fracture intensity are found to be consistent between the two methods. The results presented here are those based on the FracMan output.

RESULTS
The following results were summarised using FracMan’s block analysis tool applied to the fracture models generated using the derived relationships between fracture intensity, size and block volumes. From FracMan’s block volume analysis it was found that when the mean fracture size was 10.0 m or less, a majority of the rock mass was intact. This indicated that only large blocks were formed and therefore the average block volume is extremely high. Table 1 shows the percentage of intact rock mass for each mean fracture size at different depths. This is calculated by dividing the volume of the largest identified block by the total volume of the model. As an example, for a mean fracture size of 15.0 m at the top and middle sections of the orebody, the intact rock proportion was 10 per cent and 23 per cent respectively.

Figure 5 shows the relationships between average block volumes and the mean fracture size used for the model. As expected, the average block volume decreased with increasing mean fracture size. This is consistent with other analysis as increasing fracture sizes will indirectly cause the increase in fracture intensity and therefore the average block volume will decrease. The average block volume also decreases from the bottom of the orebody upwards as fracture density increases (top part of the orebody is more fractured).

ANALYSIS AND DISCUSSION
As demonstrated, an increase in mean fracture size leads to an increase in fragmentation in the rock mass, hence reductions in the proportion of intact rock and the average block size. For example, an increase in mean fracture size from 10.0 to 15.0 m causes a significant decrease in intact rock mass from 74.6 per cent to 10.3 per cent at the top of the formation. Additionally the average block volume for the larger mean fracture size (15.0 m) model is 5 m$^3$ compared with 14 m$^3$ for the model with the smaller mean fracture size (10.0 m). This demonstrates the sensitive influences of the mean fracture size on the final block size distribution within the orebody. As the mean fracture size is the only parameter that can not be derived from oriented core fracture mapping and therefore its value can only be assumed in this case, the final mean fracture size for the DFN model to be used to help the mine design must be chosen with great care. It is recommended that further investigations are necessary to derive a more reliable estimate for the mean fracture size.

Block size distribution
As an example, Figure 6 shows the cumulative distribution of the block volumes formed at top section of the orebody when mean fracture size of 15 m is used. This curve accounts for each of the blocks generated in the model and is a better representation of the whole rock mass at that level in the formation in comparison to a single deterministic average block volume value. Note the maximum possible block volume is 1600 m$^3$, which suggests a minimum side length of approximately 12 m for possible large caved blocks. This assessment can help to determine the adequacy of drawpoint design or the necessity of orebody preconditioning. Note this curve is derived from only one simulation (realisation), ie one possibility. In practical applications, many simulations will be required in order to derive a more reliable estimation of the characteristics. Another point worth mentioning is that a comprehensive sensitivity study is necessary in practice to assess the influences of various fracture parameters, particularly the mean fracture size as it is a parameter with the greatest uncertainty in this case. This study is beyond the scope of the current research due to time constraint. However, the framework and the assessment procedure have been developed in this work.

From our analysis, the majority of the mean fracture sizes correspond to a rock mass which was primarily intact. For example, if the in situ rock mass contains discontinuities with a mean fracture size less than 10.0 m, block caving may not be suitable in this case without first conducting a preconditioning fracturing program. The best scenario modelled was for a mean fracture size of 15.0 m at the top of the formation. This resulted in the smallest intact rock mass of 10.3 per cent with 80 per cent of the block volumes being less than 1000 m$^3$.

Impact of results on mine design
Based on our initial and naïve assessment, the majority of the fracture models explored suggest that the orebody has high proportion of intact rock mass. The proportion of the intact rock increased with depth. The current design of a block caving with two lifts could remain feasible with the
introduction of a form of preconditioning fracturing plan to increase the intensity and size of fractures. The extent of the pretreatment program will be related to the amount of artificial fracturing required to reduce the average block volume to a satisfactory level. A balance will need to be found between the costs of pretreatment and the economic benefits of reducing the block volume. Details of the pretreatment design is beyond the scope of our research, however the technique developed in this work can be incorporated in the process to assess the suitability of a treatment program. If a hydraulic fracturing of packed borehole is used, it is possible to monitor the seismic events generated during the hydraulic fracturing process. These events can then be used to provide a more reliable estimate of the fracture size which is the critical parameter that needs to be determined in this case. The seismic events can also be used to construct more realistic conditional fracture models using technique such as the Markov chain Monte Carlo simulation (Xu, Dowd and Wyborn, 2013) so a more realistic design and assessment of the caving operation can be achieved.

CONCLUSIONS AND RECOMMENDATIONS

For this case study, fracture models were created using both FracMan and FracSim3D. Analyses of these generated models provided information on the distribution of block volumes and their dependence on fracture parameters such as the mean fracture size of the DFN model. The analyses were completed at different depths within the orebody and the results indicated that a smaller percentage of rock mass was intact at shallower depths. In addition to this, it was found that block volume distribution is sensitive to the mean fracture size used in the model and large fracture sizes resulted in significantly small average block volumes.

Based on the mean fracture sizes modelled, the block volume analysis indicated that a preconditioning treatment might be necessary in this case to ensure an efficient block caving operation. A more intense preconditioning will be needed in the lower part of the orebody due to the larger proportion of intact rock.

Further investigation could be undertaken to analyse variation in fracture size distribution with depth as it has been demonstrated to be a critical parameter in this case. It is possible that in conjunction with fracture intensity, fracture size will also decrease with depth. This could alter the fracture model and resultant distributions of block sizes significantly.

ACKNOWLEDGEMENTS

We acknowledge Golder Associates for giving us the permission to use their FracMan software. Without this software, our research would not have progressed as far as it did.

We would also like to extend our gratitude to Jonathan Veale, Louise Faint, Lucas Ljubicic and Sam Thomson for providing us with the structure for our VBA code which we were able to adapt to use with the FracSim3D output.

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INTRODUCTION

The underground excavation of rock results in the disturbance and redistribution of the in situ stress field. As a result, the surrounding undisturbed rock compensates for the excavated rock by supporting the redistributed stresses. With continuous mining activity, the burdened rock mass experiences further loading and unloading in the form of cyclic loading due to drilling and blasting, truck haulage vibrations and mine induced seismicity. This study concentrates on the changes in the strength and deformability of a brittle rock subject to systematic cyclic loading.

The notion of fracture damage and quantification of its effects on the mechanical properties of a material has developed into a study known as damage mechanics (Eberhardt, Stead and Stimpson, 1999). Stress-induced brittle fracture damage is caused by the initiation, propagation and coalescence of microfractures within a rock during compression. Brace (1964) and Bieniawski (1967) define the five stages of brittle fracture of rock during compression. These stages have since been discussed in a number of publications (Goodman, 1989; Lajtai, Carter and Scott Duncan, 1991; Martin and Chandler, 1994; Gatelier, Pellet and Loret, 2002).

This study primarily focuses on the two stages, crack damage threshold and ultimate rock strength. The crack damage threshold, $\sigma_{cd}$, is the critical value at which cracks begin to propagate in an unstable manner. Unstable crack growth continues until numerous microfractures have coalesced and the rock can no longer support any load, whether it be constant or increasing and results in failure of the material (Eberhardt et al, 1998; Eberhardt, Stead and Stimpson, 1999).

In this study the crack damage threshold $\sigma_{cd}$ was determined by monitoring the secant Young’s modulus $E_{sec}$; the crack damage threshold occurs when $E_{sec}$ reaches a maximum as shown in Figure 1.

Progressive Damage of Hawkesbury Sandstone Subjected to Systematic Cyclic Loading

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ABSTRACT

An experimental investigation was carried out on the Hawkesbury sandstone to identify and predict the change in mechanical properties of the rock during uniaxial and triaxial cyclic compressive testing. Cyclic tests were completed at different stress levels and unloading amplitudes. Damage increased with an increase in unloading stress level and unloading amplitude. Results indicate that leading up to unstable crack propagation, approximately 65 per cent of the cumulative axial strain (measured at the peak of each cycle) occurs prior to the critical damage point with a rapid increase after this point. The rate at which strain accumulated after the critical damage point is dependent on the stress level at unloading and unloading amplitude. A preliminary damage model was proposed to predict reduction in the peak strength and tangent Young’s modulus of a rock due to cyclic loading. For future development of this work, a methodology to conduct cyclic loading tests was proposed.

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develop a model for brittle rocks which considers the various cyclic loading parameters.

In this study, it is aimed to study the effect of various parameters including the magnitude of the stress level at the initiation of cyclic loading, \( q_{\text{un}} \), the unloading amplitude of the cyclic loading, \( q_b \), damage increment (the number of cyclic loads), \( i \) and the confining pressure/depth of the rock mass, \( \sigma_3 \) on mechanical properties of Hawkesbury sandstone. It is also intended to develop a preliminary damage model to predict reduction in the peak strength and tangent Young’s modulus due to cyclic loading.

**METHODOLOGY**

Uniaxial and triaxial compression testing was used to replicate systematic cyclic loading in the laboratory. To perform uniaxial and triaxial tests, a closed-loop servo-controlled testing machine with a loading capacity of 250 kN and a loading rate capability in the range of 0.001–10 mm/s was used. A high-pressure Hoek cell and hydraulic pressure system were used to apply and control the confining pressure for triaxial tests. The machine is equipped with a linear variable differential transformer (LVDT) to measure axial displacement and control axial loading. All samples were cored and prepared according to International Society for Rock Mechanics (ISRM) standards with approximate dimensions of 100 mm in height and 42 mm in diameter. A ramp waveform was selected for the machine to apply cyclic loading. Pairs of axial and lateral strain gauges were secured on opposite sides of the specimen to measure axial and lateral strains as shown in Figure 2.

The following procedure was followed to protect strain gauges from being damaged in the Hoek cell:

- all the voids on the specimen surface that would otherwise be in contact with the strain gauges were filled with epoxy precoating (PS XH107F) to eliminate rough surfaces, left to dry for 24 hours and sanded back to a smooth surface
- once the strain gauges were secured to the specimen surface, they were coated with polyurethane for protection
- immediately before conducting triaxial tests, microcrystalline wax was then applied over the strain gauge to allow movement between the strain gauge and the triaxial Hoek cell jacket.

**RESULTS**

A total of 24 tests were conducted for this study. This consisted of 12 uniaxial tests (three monotonic, nine cyclic)
and 12 triaxial tests (four monotonic, eight cyclic). The test results were compiled to plot graphs of the applied deviatoric stress \(q\) against the axial strain measured by the LVDT, the axial \(\varepsilon_{\text{ax}}\) and lateral strain \(\varepsilon_{\text{lat}}\) recorded by strain gauges and the volumetric strain \(\varepsilon_{\text{vol}}\), which was calculated using strain gauges results.

### Monotonic compressive loading

Figure 4 displays uniaxial monotonic testing results for samples HS1 and HS3. Monotonic testing was used to obtain the failure strength \(q_f\) of an unconfined specimen to determine the stress level at which cyclic loading should be initiated \(q_{\text{un}}\). The average peak monotonic strength for uniaxial loading was 47.1 MPa. The amount of axial strain measured by the axial external LVDT is larger than that recorded by strain gauges. This is because of the bedding errors and also deformations in loading piston, cap and axial loading system, which were included in the recorded displacements by the axial external LVDT. Figure 5 displays triaxial monotonic testing results for samples HS15 and HS18. These tests were used to obtain the failure strength \(q_f\) of a specimen confined at \(\sigma_3 = 4\) MPa to determine the stress level at which cyclic loading should be initiated \(q_{\text{un}}\). Due to sample variability a wide range of monotonic strengths were measured, ranging between 61 MPa and 82 MPa. The average peak monotonic strength for triaxial loading was 69.2 MPa.

### Cyclic compressive loading

Figure 6 summarises results for uniaxial cyclic loading tests HS4 and HS7. Both tests were conducted under the full unloading (100 per cent unloading amplitude) cyclic testing regime, requiring 70 and 106 cycles to failure respectively. Cyclic loading was initiated at 42.6 MPa and 40.3 MPa respectively causing a failure of the samples at 87 per cent and 82 per cent of the peak monotonic strength.

Figure 7 summarises results for triaxial cyclic loading tests HS12 and HS16. HS12 was conducted with unloading amplitude of 80 per cent and HS16 conducted under full unloading (100 per cent). Cyclic loading was initiated at 74.6 MPa and 68.8 MPa, requiring 42 and 129 cycles until failure respectively.

A summary of the successful systematic cyclic loading tests can be found in Table 1.

### ANALYSIS OF RESULTS

#### Predicting \(q_f\)

The highest stress experienced by samples that underwent cyclic loading was \(q_{\text{un}}\), ie the peak strength was never reached and therefore was not known for any of the samples which underwent cyclic loading. Due to the large inevitable variation among samples tested in this study, the peak monotonic strengths \(q_f\) of samples subject to cyclic loading were unknown. However, to quantify the degradation in strength due to cyclic loading, the peak monotonic strength \(q_f\) was required; therefore a method for predicting the peak strength of a specimen was devised.

By using monotonic test results, the ratio of \(q_{\text{cd}} / q_f\) was determined for \(\sigma_3 = 0\) MPa and 4 MPa as 0.988 and 0.973 respectively. Once this was established, the crack damage threshold \(q_{\text{cd}}\) could be determined (stress corresponding to the maximum secant Young’s modulus, Figure 8) for cyclic loading tests which had \(q_{\text{un}} \geq q_{\text{cd}}\). The predicted peak strength, \(q_{\text{fp}}\), could then be calculated from the following formula:

\[
q_{\text{fp}} = \frac{q_{\text{un}}}{q_{\text{cd}}}
\]
\[ q_{fp} = \frac{q_{cd}}{0.988} \]  

(1)

\[ q_{fp} = \frac{q_{cd}}{0.973} \]  

(2)

where:

- \( q_{cd} \) = crack damage threshold
- \( q_{fp} \) = predicted compressive strength of the specimen

For cyclic loading tests which had a \( q_{fp} < q_{cd} \), a curve of best fit was established for the secant Young’s modulus (\( E_{\text{sec}} \)) versus \( \varepsilon_{\text{ax}} \) curves in order to predict the maximum secant Young’s modulus, \( E_{\text{sec(max)}} \). An example of this is shown in Figure 8. Maximum \( E_{\text{sec}} \), which is measured during axial loading in monotonic test, has been used previously as a critical point in which extensive damages happen in a rock specimen after this point (Taheri and Tani, 2008; Taheri and Chanda, 2013). This concept is used in this study as well to determine critical...


Progressive Damage of Hawkesbury Sandstone Subjected to Systematic Cyclic Loading

It was observed that during cyclic loading the rock accumulated strain relatively uniformly followed by a rapid strain increase as it headed towards unstable crack propagation. This rapid accumulation in strain occurred on average at approximately 65% of the cumulative axial strain. The point at which the rapid strain accumulation occurred was termed the critical damage point \( \omega_c \). Figure 10 shows an example of results which in this test \( \omega_c \) has been measured at 58% per cent of the cumulative axial strain.

It was noted that at higher unloading stress levels the more rapid the accumulation of strain was after the critical damage point. It was also observed that the larger the amplitude of cyclic unloading the greater accumulation of damage experienced by the specimen and a slower accumulation of strain beyond the critical damage point. The same observations in the trend of strain accumulation were observed in the axial, lateral as well as the volumetric strain graphs. Therefore, it was concluded that there is an increase in damage with an increase in stress level at unloading and unloading amplitude.

**Deformability**

To quantify the effects of cyclic loading on the deformability of brittle rock, Young’s modulus and Poisson’s ratio were calculated. The tangent Young’s modulus and Poisson’s ratio were determined at 50% per cent of \( q_{un} \) for each damage increment.

In all tests there is a moderate decline in the tangent Young’s modulus of each sample until reaching its elastic limits, after which there is a rapid decrease in stiffness until unloading. Figure 11 show an example of results for a cyclic triaxial test. Poisson’s ratio initially increases rapidly, followed by a stage of slower accumulation. During this second stage, the rate of increase remains relatively constant as shown by the linearity of graphs in the middle cycles. As the sample begins

---

### TABLE 1

Summary of uniaxial and triaxial cyclic loading testing.

<table>
<thead>
<tr>
<th>Test type</th>
<th>Test #</th>
<th>Unloading amplitude (%)</th>
<th>Initiation stress, ( q_{un} ) (MPa)</th>
<th>Failure stress, ( q_f ) (MPa)</th>
<th>Strain at failure, ( \varepsilon_{ax,f} ) (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Uniaxial</td>
<td>HS2</td>
<td>100</td>
<td>47</td>
<td>44.2</td>
<td>0.48</td>
</tr>
<tr>
<td>Uniaxial</td>
<td>HS4</td>
<td>100</td>
<td>42.6</td>
<td>40.8</td>
<td>0.5</td>
</tr>
<tr>
<td>Uniaxial</td>
<td>HS5</td>
<td>50</td>
<td>42.7*</td>
<td>45.4</td>
<td>0.44</td>
</tr>
<tr>
<td>Uniaxial</td>
<td>HS6</td>
<td>100</td>
<td>44.1</td>
<td>42.7</td>
<td>0.38</td>
</tr>
<tr>
<td>Uniaxial</td>
<td>HS7</td>
<td>100</td>
<td>40.3</td>
<td>38.6</td>
<td>0.36</td>
</tr>
<tr>
<td>Uniaxial</td>
<td>HS8</td>
<td>75</td>
<td>42.6*</td>
<td>43.4</td>
<td>0.45</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS9</td>
<td>100</td>
<td>74.8</td>
<td>69.7</td>
<td>0.52</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS10</td>
<td>100</td>
<td>69.4*</td>
<td>73.1</td>
<td>0.504</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS11</td>
<td>80</td>
<td>74.6</td>
<td>67.9</td>
<td>0.51</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS12</td>
<td>80</td>
<td>74.6</td>
<td>69.7</td>
<td>0.49</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS13</td>
<td>80</td>
<td>74.6</td>
<td>68.9</td>
<td>0.45</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS16</td>
<td>100</td>
<td>68.8</td>
<td>68.6</td>
<td>0.71</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS17</td>
<td>100</td>
<td>61.5*</td>
<td>61.2</td>
<td>0.83</td>
</tr>
<tr>
<td>Triaxial</td>
<td>HS22</td>
<td>50*</td>
<td>61.8*</td>
<td>68.6</td>
<td>0.61</td>
</tr>
</tbody>
</table>

* This value was altered during the test to cause the specimen to fail.

---

**FIG 8** – Predicting the maximum secant Young’s modulus by using a curve of best fit (grey dashed line) established from monotonic tests.

**FIG 9** – Cumulative strain measurement during systematic cyclic loading.

**FIG 10** – Cumulative axial strain graph for the uniaxial test, HS7. The critical damage point occurs after damage increment 86.
to approach failure, Poisson’s ratio begins to rapidly increase until failure.

**Unloading with respect to the crack damage threshold**

The results indicated that when unloading commenced above the crack damage threshold $q_{cd}$, the specimen attained a rapid accumulation of damage as it approached failure (Figure 12, HS8). In contrast, when specimen was cyclic loading starts below $q_{cd}$, quite large number of cycles required to create damage into the rock and specimen showed to reach a standstill after a certain amount of cycles (Figure 12, HS22). It can be concluded that if a specimen is unloaded below $q_{cd}$ after a certain amount of cycles it will not incur further permanent damage with repeated exposure to cyclic loading.

**Confinement**

Results presented in Figure 13 show that for an increase in confining pressure a higher stress level at start of cyclic loading required. As well as this, it was found that for rock specimens under the same unloading amplitude the one at higher confinement required a greater amount of cycles for failure to occur. Note that in this study only cyclic triaxial tests at $\sigma_3 = 4$ MPa were performed. More tests at different confining pressures are required to completely validate this conclusion.

**DAMAGE MODEL**

A preliminary cyclic loading damage model is proposed for the Hawkesbury sandstone with reference to the change in the peak strength and the reduction in stiffness. The change in these properties was explored by considering controlled cyclic loading parameters such as $q_{un}$ and $q_{b}$ as well as the axial strain and damage increment at failure. The results of this study resulted in the conceptual relationships shown in Figure 14.

Figures 15 and 16 represent examples of relationships obtained from the experimental results. It can be concluded that for a high stress level at unloading, a specimen will experience a smaller axial strain at failure (Figure 15) as well as a smaller reduction in the tangent Young’s modulus (Figure 16).

From the developed conceptual model represented in Figure 14, it can be seen that for a higher stress level at unloading, a smaller number of damage increments are required to fail a specimen. Therefore, there will also be a larger reduction in the tangent Young’s modulus (Figure 14f).

**PROPOSE A CYCLIC LOADING TEST METHOD**

The difficulties experienced in this study primarily revolved around the high variance among the samples tested. In multiple cases the variability in strength and stiffness of a specimen required a change in one or more cyclic loading parameters in order to fail the specimen (loading rate, unloading amplitude). As a result, a systematic cyclic loading method is proposed to overcome these issues. The method is outlined as follows:
determine the extent of specimen heterogeneity from the relationship between $q_f$ versus $\varepsilon$
- ease of calculating $q_f$ without the need of a prediction curve for heterogeneous samples through knowledge of $q_{cd}$
- eliminate time issues associated with failing the rock as it is loaded at or beyond the unstable crack growth region ($\geq q_{cd}$)
- cyclic loading parameters remain constant; testing method is consistent, producing more reliable data.

CONCLUSIONS

The following conclusions were drawn from analysis of systematic cyclic loading:
- the higher the stress level of unloading the more rapid the accumulation of strain after the critical damage point
- the larger the amplitude of cyclic unloading the greater accumulation of damage experienced by the specimen and less rapid accumulation of strain beyond the critical damage point
- rock under confining pressure requires a greater number of cycles to failure and higher initiation stress of unloading in comparison to unconfined rock
- cyclic unloading below $q_{cd}$ will result in rock acquiring limited amounts of permanent damage
- cyclic unloading above $q_{cd}$ will result in permanent deformation of the rock through degradation of stiffness and accumulation of irreversible strain
- a preliminary damage model was developed to predict the degradation of the mechanical properties of Hawkesbury sandstone subjected to cyclic loading.

ACKNOWLEDGEMENTS

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REFERENCES


A Study of On-bench Explosives Reloading at the Foxleigh Coal Mine

E L Berriman\(^1\) and P C Hagan\(^2\)

**ABSTRACT**

Foxleigh Coal Mine currently has a fixed plant explosives reloading facility that services its four active mining pits. For the majority of pits, the distance between the plant and pit results in excessive cycle times due to lengthy reload times creating a bottleneck in the explosives loading capacities of the mobile manufacturing units (MMUs). An analysis was undertaken to assess the potential benefits of on-bench reloading in terms of explosives loading rates and financial savings.

A time and motion study was conducted to collect data on the cycle times for the two available Orica Bulkmaster MMUs operating on-site. The data was used to assess the potential loading rates that could be achieved and cost reductions associated with three different on-bench reloading scenarios. The results were compared to the current fixed plant reloading arrangement to demonstrate the potential increase in loading rate and financial savings that could be achieved from on-bench reloading.

A unit basis comparison showed that all on-bench reloading scenarios resulted in an improvement in terms of cost per bank cubic metre for an average loading day at Foxleigh. The best scenario resulted in an increase in daily explosives loading rates as well as a reduction in costs. A risk analysis identified a number of potential risks that on-bench reloading poses to the operation, but with the management of ammonium nitrate prill supply and constant preventative maintenance of the MMUs, these risks can be substantially mitigated.

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**INTRODUCTION**

Foxleigh Coal Mine, located in the Bowen Basin of Central Queensland, is a small open cut coal mining operation with a current annual production capacity of 3.3 Mt of product coal. It is managed by Anglo American Metallurgical Coal (AAMC) and is a joint venture between AAMC and two international organisations – POSCO and Itochu – with AAMC being the majority shareholder (Anglo American, 2012). As is common with most open cut coal operations, it is first necessary to blast and remove the overburden in order to expose the underlying coal seam. The blasting process generally uses bulk explosives vehicles to transport the explosives products from on-site storage facilities to the blast site. Foxleigh currently has two Orica Bulkmaster mobile manufacturing units (MMUs) on-site which transport, mix and deliver explosives products to the pits.

The current explosives reloading arrangement has created a bottleneck in the explosives loading capacities of the MMUs. The distance that the MMUs are required to travel to reload with explosives is quite long for some pits, resulting in long reload turnaround times that adversely impact on the loading capacities of the MMUs.

The current mine layout at Foxleigh is shown in Figure 1. Shown within each of the rectangles are the four current active pits within the mining lease, these being: WC, One Tree (OT), Pipeline (PL) and Carlo Creek (CC). Servicing these four pits is a single fixed plant ammonium nitrate (AN) reload facility located in the centre of Figure 1 designated as Orica Reload.

An analysis of the 2012 loading performance data showed that the WC Pit had the highest daily loading rate. This is...
due primarily to its close proximity to the reload facility. The aim of the study was to assess the benefits of reducing the distance of the reload facility to the active loading bench by considering on-bench reloading.

On-bench reloading involves purchasing or hiring mobile storage containers to temporarily store the explosives products. These mobile storage containers, with the aid of a prime mover/tipper, can be relocated to the bench that is currently being loaded with explosives therefore decreasing the distance from the loading bench to the reload facility. The decrease in travel distance should significantly reduce the reload turnaround times and therefore increase the daily loading rates of the MMUs.

TIME AND MOTION STUDY

Data collection
A time and motion study was conducted over two consecutive days at the PL and WC pits. Data collected included hole load times, travel times to and from the explosives reload and reload times. The four sequential steps in the process are shown in Figure 2.

Although data was collected on MMU performance for the two pits, they provide a snapshot of the loading rates for the entire operations as these pits are the furthest from and closest to the Orica reload facility respectively.

Data analysis
The data collected from the time and motion study was analysed to find the average reload time, production time, speed of the MMUs and on-bench delays in each cycle. Values for these parameters are provided in Table 1.

![Image](image.png)

**FIG 2** – Steps in the mobile manufacturing unit (MMU) loading process.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Mean value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average speed</td>
<td>35.8 km/h</td>
</tr>
<tr>
<td>Delays</td>
<td>9.6 min</td>
</tr>
<tr>
<td>Reload time</td>
<td>21.4 min</td>
</tr>
<tr>
<td>Production time</td>
<td>27.5 min</td>
</tr>
</tbody>
</table>

**TABLE 1**
Time and motion study analysed data.

ON-BENCH RELOADING
To introduce on-bench reloading at Foxleigh mine, the following extra equipment would be required:
- AN storage container
- diesel storage container
- auger (delivery mechanism for both AN and diesel)
- prime mover/tipper (to relocate the storage containers).

Scenarios
The four scenarios that were analysed included:
- S1 current fixed plant arrangement
- S2 one MMU with two operators
- S3 two MMUs with four operators
- S4 one MMU with three operators.

Scenarios 2 to 4 involve the acquisition of the same on-bench reloading equipment, only the use of current equipment and the number of operators are altered. Scenarios 2 and 3 involve alternating even time crews with no overlaps, therefore having one operator on-site per MMU. Scenario 4 involves three operators working overlapping rosters, therefore having two operators on-site the majority of the time with only one MMU. This scenario allows for fewer delays as the second operator can take care of such issues as prill deliveries and administration paperwork, which results in higher overall loading capacity for an MMU.

RESULTS AND ANALYSIS

Productivity
Productivity refers to the daily loading rates of the MMUs on-site as measured in kilograms of explosives loaded per shift. This calculation was based on the data analysed from the time and motion study that included:
- average production time
- average reload time
- average travel speed of the MMU
- average on-bench delays.

These factors remain unchanged in the analysis. However, two other factors were also considered in the productivity calculation, these being:
1. the distance of the loading bench to the reload
2. the operating hours of the MMU(s) per day.

The distance between each pit and the reload for the current fixed plant setup was measured using Minescape. For simplicity, the distance from the loading bench to the on-bench reload was held constant at 1 km. The distance to each pit is listed in Table 2. The number of operating hours per day was calculated for each scenario based on a 12 h shift taking into consideration the delays which would be experienced throughout the day. These delays include pre-start meetings, emulsion prill re-load times, prill delivery escort and wait times.

<table>
<thead>
<tr>
<th>Pit</th>
<th>Distance to fixed plant reload (km)</th>
<th>Distance to on-bench reload (km)</th>
</tr>
</thead>
<tbody>
<tr>
<td>WC Pit</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>One Tree (OT)</td>
<td>8</td>
<td>1</td>
</tr>
<tr>
<td>Pipeline (PL)</td>
<td>10</td>
<td>1</td>
</tr>
<tr>
<td>Carlo Creek (CC)</td>
<td>10</td>
<td>1</td>
</tr>
</tbody>
</table>
times, crib, wait on blast crew, wait on other MMU and administration. The operating hours per day can be seen in Table 3.

The method used to calculate the daily loading rates for each scenario is shown in Equations 1, 2 and 3.

\[ \text{cycle time} = l + t_r + r + t_b + d \]  
\[ \text{loads/day} = \frac{\text{op hours/day} \times 60}{\text{cycle time}} \times 2 \text{ for 2 MMLUs} \]  
\[ \text{t/day} = \text{loads/day} \times t/load \]

where:
- \( l \) = load time
- \( t_r \) = travel time to reload
- \( r \) = reload time
- \( t_b \) = travel time back to bench
- \( d \) = delays

The amount of explosives per load refers to the combined explosives products that the MMUs can discharge per load. This was calculated as an average from the data collected and was found to be 10.5 t.

Comparing the productivity of the fixed plant reloading setup and the on-bench reloading scenarios provides a good measure of the production benefits of moving to on-bench reloading. Figure 3 shows a comparison of the potential loading rates of the on-bench reloading scenarios to the expected loading rates of the current fixed plant reloading setup.

As the graph in Figure 3 indicates, Scenario 3 achieves a higher production rate compared to the current arrangement (S1). Scenario 4 achieves a similar result except for the WC Pit. This is because the distance between this pit and the fixed plant is only 2 km, which is much shorter than the other pits, and in effect is an on-bench reloading arrangement with two MMUs. Scenario 1 provides at best a production rate similar to the current arrangement for the CC and PL pits only, whereas for the other two pits the production rate is less.

Further analysis was conducted on the potential loading rates for on-bench reloading and the expected loading rates for the current fixed plant reloading setup. Figure 4 shows the estimated number of days it would take to load the 2013 budgeted explosives for each scenario.

Two of the on-bench reloading scenarios require fewer days to load the 2013 budgeted explosives. There are many advantages in having faster loading times and fewer number of loading days. The main advantage being it provides the potential to increase annual mining rates.

### Savings

A cost analysis was undertaken that considered the difference between the different scenarios. The costs included in the analysis were based on the Orica contracting costs, operator costs and the costs for the on-bench reloading equipment. Fixed costs, such as explosives and potential demurrage costs, were not included as they remain the same for each scenario.

Comparing the costs for the different scenarios is an indicator of the potential financial savings that may accrue from the introduction of on-bench reloading. Table 4 provides a summary of the equipment and operators required for each of the scenarios.

A comparison of the on-bench reloading costs and the current fixed plant setup costs is shown in Figure 5. As the graph shows, Scenarios 2 and 4 have lower costs compared to the current arrangement (S1). Substantial savings in the order of 40 per cent could be achieved with Scenario 2 while the savings with Scenario 4 are slightly less at 32 per cent. These translate to savings of $990 000 and $775 000 respectively per year. There were no significant savings found with Scenario 3.

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Number of operators</th>
<th>Operating hrs/d/MMU</th>
</tr>
</thead>
<tbody>
<tr>
<td>S1: Current fixed plant setup</td>
<td>2</td>
<td>5.5</td>
</tr>
<tr>
<td>S2: On-bench reloading 1 MMU</td>
<td>2</td>
<td>6.5</td>
</tr>
<tr>
<td>S3: On-bench reloading 2 MMUs</td>
<td>4</td>
<td>5</td>
</tr>
<tr>
<td>S4: On-bench reloading 1 MMU</td>
<td>3</td>
<td>8</td>
</tr>
</tbody>
</table>

**TABLE 3**

Daily operating hours per mobile manufacturing unit (MMU).

![Graph showing comparison of fixed plant and on-bench reloading daily productivity rates.](image1)

**FIG 3** – Comparison of fixed plant and on-bench reloading daily productivity rates.

![Graph showing comparison of different scenarios to load the 2013 budgeted amount of explosives.](image2)

**FIG 4** – Comparison of different scenarios to load the 2013 budgeted amount of explosives.
Combination of productivity gains and cost savings

A comparison of the combination of the effects of potential changes in loading rates and cost savings of each scenario is shown in Figure 6. The graph shows that the biggest costs saving results from one MMU on-bench reloading while the highest loading rate results from two MMUs on-bench reloading.

It is difficult to weigh up the best option for on-bench reloading based on the results in this graph alone. Therefore a comparison was made based on the potential cost per unit of explosives. This value accounts for the cost per bank cubic metre for an average day of loading using Equation 4. The results of this unit cost basis comparison are shown in Table 5.

\[
\text{total cost for 2013} = \frac{\text{kg explosives 2013}}{\text{total bcm for 2013}} \times \frac{\text{kg loaded/day}}{} \]

All on-bench reloading options provide a significant improvement in terms of loading rates and cost savings. This is highlighted by the significant reduction in the cost per bank cubic metre for an average day of loading. This analysis suggests the best option is Scenario 4 with one MMU on-bench reloading and three operators. This contrasts with Scenario 3 that achieves the highest daily loading rates and Scenario 2 and the option that achieves the largest cost savings.

Risks

Considering the different scenarios and the risks associated with each of these it was found that they could be divided into three different areas. These different areas are the prill storage, the number of MMUs on-site and the number of operators on-site. These risks all have the potential to result in an interruption of loading on-site, and considering that blasting is an integral component of the mining process, this would be a significant risk for the operation.

The risks associated with the prill storage are mainly associated with on-bench reloading. This is due to this setup not having the large storage facility on-site that the fixed plant reloading process does. This large fixed plant storage facility generally provides a buffer for any mistakes in ordering prill or late prill deliveries, although in the past loading on occasion was prevented due to depletion of prill on-site. The on-bench reloading option has a greater risk of causing a stoppage in loading due to the depletion of the prill on-site as the storage facility is smaller. Therefore, mistakes in ordering prill and late prill deliveries pose a significant risk to on-bench reloading.

The risks associated with the number of MMUs on-site relate to potential MMU breakdowns, unexpected maintenance of MMUs and damage to the MMUs. If any of these arise then an MMU cannot load and if there was only one MMU on-site then no loading could take place. With two MMUs on-site it is less likely there would be delays caused by both units being down for maintenance.

The risks associated with the number of personnel on-site are related to the potential for an operator to be absent from site. This is particularly an issue if there is only one MMU on-site and one operator for that MMU, since if an operator is absent then no loading will occur.

Conclusions

A study of on-bench reloading at the Foxleigh Coal Mine has shown it would be beneficial in terms of an improvement in daily explosives loading rates and cost savings compared to the current fixed plant reloading arrangement. Each of the on-
bench reloading scenarios considered in the study resulted in a saving for the mine with two out of the three scenarios resulting in better daily loading rates. On a unit basis comparison, taking into account the daily loading rates and savings, all the on-bench reloading scenarios would provide a better outcome than the current fixed plant scenario. Overall, the best option is Scenario 4, which provides on-bench reloading with one MMU and three operators.

This scenario provides the following benefits in comparison to the current fixed plant reloading arrangement:

- 15 per cent increase in daily explosives loading rate that is equivalent to an additional 10.4 t of explosives
- 32 per cent or $775,000 saving/year
- 39 per cent or $7/bank cubic metre (bcm) decrease for an average day of loading.

Overall, this on-bench reloading scenario would result in higher productivity at a lower cost.

The analysis of on-bench reloading identified several potential risks associated with this reloading arrangement. These risks are the depletion of prill on-site and MMUs being out of operation, both of which could result in an interruption to explosives loading. These risks could be mitigated by ensuring that a constant supply of ammonium nitrate is maintained to the mine site and increasing the level of preventative maintenance to the MMUs.

REFERENCES

INTRODUCTION

Overbreak in underground stopes is an issue prevalent to all underground mines that employ stoping techniques. Overbreak is seen when more rock is broken in a stope compared to the original design shape (Germain and Hadjigeorgiou, 1997). As illustrated in Figure 1, overbreak usually occurs in waste. The overall grade of the stope is diluted thus hindering potential profits that may have been seen by that stope (Maerz, Ibarra and Franklin, 1996). In addition overbreak results in a number of other undesired consequences such as:

• ground control problems as a reduction in the module and strength of rocks are observed
• poor fragmentation
• production scheduling errors (Singh and Xavier, 2005).

Numerous researchers have highlighted the detrimental hazards that overbreak has on an underground operation. This has subsequently driven a number of studies into isolating parameters that affect overbreak and determining methods of control. While these studies acknowledge the importance of the overbreak phenomenon, only a handful has focused on predicting overbreak. According to Jang and Topal (2013), predicting overbreak would be the first step in developing an effective overbreak management and blasting reconciliation system. Overbreak prediction begins in the design stage of underground excavations. To date no set methodology exists for predicting stope overbreak with practices for predicting overbreak varying dramatically from site to site. A set method/system that could accurately predict stope overbreak has advantages including:

• increased realisation of costs and profits
• improvements in scheduling and ground control
• ability to use this method as a learning/design to effectively implement methods of control.

Due to the high number of variables that affect overbreak and the fact that no linear correlation exists between these variables, empirical methods would prove to be ineffective in predicting...
Mining Education Australia – Research Projects Review

D Boxwell, H Jang and E Topal

Overbreak

Overbreak, as illustrated in Figure 1, is a result of drilling and blasting to extract ore in an underground excavation. While to date drilling and blasting remains one of the most cost-effective methods for extracting ore, overbreak and the associated effects can have a detrimental effect on any underground operation. In terms of costs, overbreak that usually occurs in waste, acts to dilute the grade of a stope. Material that is below the cut-off grade cannot be mined at a profit (Maerz, Ibarra and Franklin, 1996). While Maerz, Ibarra and Franklin (1996) focused on the financial effect that overbreak has on an operation, Singh and Xavier (2005) focused on the operational consequences that overbreak causes. These include the following:

- breakdown of the inherent interlocking of the weakness planes
- ground control problems as a reduction in the moduli and strength of rocks is observed
- poor fragmentation
- production scheduling errors
- restricted access due to damaged ground for drilling and charging operations
- increased costs due to installation and maintenance of supports.

These operational hazards pose a significant risk to personnel working in the vicinity. Coupled with the economic effects of overbreak, it is evident that improper management of overbreak can result in a number of consequences to an underground excavation.

Factors influencing overbreak

In a study conducted by Singh and Xavier (2005) into the causes of overbreak in underground excavations, it was discovered that there were numerous parameters that affect the extent of overbreak. In general they can be classed as:

- rock mass features
- explosive characteristics and distribution
- blast design and execution.

Rock mass features also referred to as geological parameters include strength of rock mass, discontinuity characteristics and water conditions. These factors are fixed and cannot be changed. Numerous studies have identified some relationships that exist between these factors and overbreak. For instance Cunningham and Goetzsche (1990) suggested that the presence of joints affects the attenuation of the induced stress wave. Attenuation was shown to be minimal, when the angle of joint orientation is perpendicular or parallel to the face and increases to a maximum when the angle is between 15 and 45° as shown in Figure 2.

Unlike rock mass features, explosive characteristics/distribution and blast design/execution are not fixed factors and can be altered. These blasting and drilling parameters include velocity of detonation (VOD), powder factors, drilling geometry (burden and spacing), delay sequencing and hydraulic radius, etc. It was observed that during trials conducted by Singh and Xavier (2005) that explosives that have a higher VOD yield less damage around the perimeter of an excavation. This damage around the perimeter was found to directly increase the amount of overbreak. Higher VOD explosives are decoupled and yield higher shock energy and less gas energy. This is illustrated in Figure 3.

Predicting stope overbreak

Predicting overbreak is an extremely important part of the underground mining process and begins when designing the stope. Mining operations have suffered terrible financial and technical consequences from incorrect predictions of overbreak. While some of the studies discussed previously identified the relationship that exists between some variables and the influence they have on overbreak, they were unsuccessful in determining a method that could predict stope overbreak. Predicting stope overbreak has in the past been avoided as there exists a lack of understanding of the extent that certain parameters have on overbreak and the relationship that exists between these parameters is unclear (Monjezi and Denghai, 2008). To date there is no set methodology or generalised practice for predicting stope overbreak.

Stope overbreak (also referred to as the dilution) is determined during the design stage of an underground excavation. Predicting overbreak in stopes has been achieved by a collaboration of geotechnical and production engineers analysing historical data on a site-by-site basis. Therefore the methodology for predicting overbreak at one site may be significantly different to that of another site (Vecchio, May 2013, personal communication).

METHODOLOGY

The proposed approach in the analysis of the project followed a structure outlined by a number of objectives. These objectives all aimed to be able to derive a system that could accurately predict stope overbreak at Plutonic underground gold mine. After analysing different methods it was evident...
that ANNs would be one of the most suitable methods for predicting stope overbreak.

**Artificial neural networks**

ANN is a branch of artificial intelligence (AI) and is essentially an information processing system that simulates structures and functions of the human brain. ANN utilises mathematical algorithms to predict an output pattern when it recognises a certain input pattern (Monjezi and Dehghai, 2008). Original inspiration of ANN came from the early models of sensory processing by the brain (Krogh, 2008). ANN has been used to solve a range of different problems in different fields including engineering, medicine and more recently in the mining industry.

An ANN can be created simply by simulating a network of model neurons. By applying mathematical weighted algorithms that mimic the process of real neurons, the network is forced to learn (to be trained) to solve a variety of problems (Krogh, 2008). A model neuron is referred to as a threshold unit whose function is illustrated in Figure 4.

The neuron receives input from a number of external sources (input data), weighs each input and then finds the sum of these weighted inputs. If the weighted sum is above a certain threshold, the output of the unit is assigned a numerical value of one. Reversely if the weighted sum is below this threshold it is assigned a numerical value of zero. Therefore the output changes from zero to one when the weighted sum of inputs is equal to the threshold. Those points in the input space which satisfy this condition define a hyper-plane. Points on the lower side of the hyper-plane are classified as zero and those on the upper side are classified as one. Essentially this means that a classification problem may be able to be solved by this threshold unit if the two classes can be separated by a calculated hyper-plane (Krogh, 2008). During training, the hyper-plane moves around according to weighted data sets until it finds its correct/optimal position in the learning space. Once the correct position is found, the hyper-plane will move minimally. If the data is linearly separable, the threshold unit will be able to correctly classify input data and determine the correct output associated with the given input data. Figure 5 demonstrates a graphical example of a classification problem with a hyper-plane.

However many classification problems are not linearly separable and consist of complex interconnected data sets. In these situations more complex neural network with a different training mechanism must be employed. The first step when presented with a non-linear problem is to separate classes by introducing more hyper-planes. This is done by introducing multiple threshold units. This is usually done by adding in an extra (hidden) layer of threshold units each of which does a partial classification of the input and sends its output to a final layer. This sums up the partial weights to determine the final output. Such a network is called a multi-layer perception network (MLP) and the general configuration is illustrated in Figure 6. MLP networks are used predominately for non-linear separable problems. By replacing the step function with a continuous function, the neural networks produce an output which is a real number. For this reason MLP networks are the most common and widely used neural network (Roy and Singh, 2004).

An ANN can be considered as an intelligent hub of mathematical algorithms that are able to predict an output pattern when it recognises a given input pattern (Monjezi and Dehghai, 2008). As explained previously the neural network is first trained by processing input data. A large number of data sets which have been normalised is ideal as it allows for the hyper plane to be positioned to the most optimal position. After the completion of training with reliable data sets, neural networks are able to identify similarities when presented with a new data set. Subsequently the ANN can predict an output variable (Krogh, 2008). This property of ANN gives excellent interpolation capability to the technique, especially when input data isn’t exact. This is one of the main reasons why neural networks have been adopted over other empirical methods to solve complex problems. ANNs have been used as direct substitute of the following empirical techniques (Monjezi and Dehghai, 2008):

- auto correlation
- multivariate regression
- linear regression
- trigonometric
- other statistical analysis techniques.

Selecting the appropriate network and training algorithm when developing an ANN is one of the most important tasks. Users of an ANN often know what problem they wish to solve...
and from previous research and experiences, they can easily determine the input and output parameters they wish to employ. This can be done in an iterative manner when using programs such as MATLAB®, version R2013a (by MathWorks) to construct an ANN. The number of hidden layers, neurons, training algorithms and network types can all be altered in an attempt to create an ANN that can predict an output based on certain inputs accurately.

There are a number of different mining related problems that have been solved by ANN in recent times especially with the fast development of computer based software programs (Tawdrous, 2006). This is supported by the studies conducted by Roy and Singh (2004) who suggested that ANN had become a favoured method for solving mining related problems due to the complexity of these problems. Traditional empirical methods have proven to be ineffective in areas where the number of variables is high and the relationship between them is not clear (Monjezi and Rezaei, 2011). A case study involving the use of ANNs to solve a mining related problem was conducted at the Gol-E-Gohar mine in Iran. The mining was experiencing large amounts of backbreak and struggling to predict backbreak accurately. After failing to predict backbreak whilst utilising standard empirical methods, the researchers adopted a MLP neural network. The team were able to predict backbreak to at 97.94 per cent accuracy and were subsequently able to reduce backbreak from 20.0 m down to 4.0 m (Monjezi and Denghai, 2008).

PLUTONIC GOLD MINE

Plutonic Gold Mine is located approximately 180 km north of Meekatharra in Western Australia (Figure 7). Plutonic was first obtained by Barrick Gold Corporation at the end of 2001 following the acquisition of Homestake Mining Company. Barrick Gold is an international gold and copper mining company with operations in six continents. In recent history Plutonic was the sixth largest gold camp in Western Australia with an estimated total endowment of 12.2 Moz of gold. The Plutonic orebody is currently well into the underground stage of mining. The Plutonic underground operation utilises longhole open stoping for extraction of stopes ranging from 100 to 45 000 t in size. In 2012, Plutonic produced 112 000 oz of gold at a total cash cost of $1220/oz. Proven and probable reserves at the end of 2012 were 206 000 oz (Barrick Gold, 2013).

Stope mining practices depend on a number of different factors and will vary drastically from site to site (Germain and Hadjigeorgiou, 1997). The more knowledge obtained after stoping practices, the more stoping practices can be optimised (Singh and Xavier, 2005). Since Plutonic first went underground in 1995, over 4000 stopes have been mined over a vast area under many different conditions (rock types, depths, etc). As would be expected, knowledge gained from different areas of the mine coupled with technological advances as well as ever changing legislation/company policy have resulted in changed in stope mining practices dramatically. Currently no accurate methodology exists for predicting overbreak prior to the stope being mined. In current practices the overbreak is calculated by the consultation of production and geotechnical engineers with the help of historical trends (Vecchio, May 2013, pers comm). Table 1 indicates common figures used for overbreak prediction.

<table>
<thead>
<tr>
<th>Condition</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stopes in the Timor region</td>
<td>20</td>
</tr>
<tr>
<td>Stopes in the Timor region with an ultramafic hanging wall or foot wall</td>
<td>30</td>
</tr>
<tr>
<td>Stopes that utilise a “dice five” hole pattern</td>
<td>15–20</td>
</tr>
<tr>
<td>Large bulk stopes in competent ground</td>
<td>10</td>
</tr>
<tr>
<td>Stopes in the Baltic region</td>
<td>10</td>
</tr>
<tr>
<td>Minimum dilution factor regardless of stope size, location, host rock, etc</td>
<td>10</td>
</tr>
</tbody>
</table>

It is estimated that approximately 90 per cent of dilution figures are determined using the information outlined here (Vecchio, May 2013, pers comm). Plutonic underground gold mine currently intends to investigate methods in which stope overbreak can be predicted more accurately. This particular area (stope overbreak prediction) is an area that has been ignored for a period of time as it was considered deemed ‘too hard to determine accurately’. This suggestion aligns with the knowledge gained in previous sections of this paper.

Data collection

The performance of ANN largely depends on the quality of data that is put into the ANN (Krogh, 2008). Accurate data would give a clear indication of whether an ANN is suitable for overbreak prediction while inaccurate data would neither confirm nor deny the suitability of an ANN. It wasn’t until the data was collected at Plutonic that an analysis on the accuracy of the data could be determined. Information was gathered from a variety of different sources including stope reconciliation forms, stope design packages and charge plans. The collected data is tabulated in order to gain a better understanding of which parameters could be used for ANN input. With nearly eight years and a total of 1100 data sets, it was evident that there were many data sets that could be inputted into the ANN. However, inconsistent record keeping practices meant that a number of parameters known to affect stope overbreak could not be used in the ANN. These include:
- any geological parameters
- burden and spacing ration
- explosive types and relative distribution.

Artificial neural network development

The first step in ANN development was to choose input parameters for ANN construction. As mentioned previously some of the parameters that have known to affect stope overbreak were not available due to the inconsistent record.
Keeping practices. Input parameters were therefore selected based on their reliability and the potential effects they may have on overbreak. The input parameters along with the general ANN architecture are illustrated in Figure 8.

Following the determination of input parameters the minimum acceptable number of data sets required for network training and testing needed to be determined. A total of 300 data sets for training and testing were collected as the number of data sets for ANN construction in this research project. Three classes of ANN were constructed in order to determine the optimum ANN model and they are:

- **Class 1**: an ANN with all 300 data sets
- **Class 2**: an ANN with outliers removed, 226 remaining
- **Class 3**: four separate ANN based on stope sizes with outliers removed.

During the process of training each of these ANN, the network type, number of neurons, type of algorithm, number of hidden layers and error regression analysis were altered in an iterative manner to determine the optimum ANN model. This was done by utilising the ANN function in MATLAB.

**RESULTS AND ANALYSIS**

After training ANN models, untrained data set was used to validate the accuracy of the models. Table 2 demonstrates the number of neurons and correlation coefficients (R) values for each model. As shown in Table 2, the majority of ANN predictions showed strong R values between measured and predicted overbreak, with Class 3 (10 000 t+) producing the most accurate results. In addition, training of ANN models are designed to stop when the mean square error (MSE) is reached to 1.0E-2. MSE is the difference between predicted and measured overbreak.

The Class 2 ANN was selected as a general overbreak prediction model because it showed stronger R value than Class 1. Class 2 ANN model contains two hidden layers, 25 neurons and utilises the gradient descent method training algorithm. Of the 226 data sets used to train and test this ANN model, 70 per cent of the data was used for training while the remaining 30 per cent was used for testing and validation.

Using the simulating function in MATLAB, the input parameters were inputted into the ANN and a prediction for stope overbreak for each stope was derived. Once these overbreak predictions were derived, a comparison between the ANN and Plutonic predicted overbreak versus actual observed overbreak could be conducted. The two methods used to analyse these predictions were:

1. correlation coefficient between predicted and observed overbreak
2. tonnage variances between predicted and observed overbreak.

The results are summarised in Table 3.

The first analysis technique which examines R values between the predicted and observed overbreak tonnages indicates that current practices at Plutonic are marginally more accurate than the ANN (0.873 versus 0.862). On the other hand, the second technique indicates that the ANN is substantially more accurate (-273.81 t versus +14 788.64 t). The R calculated gave a false representation of the ANN accuracy. This was due to the phenomenon of over-fitting. Over-fitting occurs when there are limited data sets in certain regions of the ANN, which cause the ANN to greatly over or under estimated stope overbreak tonnages. Even though the ANN predicted stope overbreak more accurately, the large errors in ANN prediction in regions where there were limited data sets caused R to be lower. This phenomenon is highlighted in Figure 9.

As can be seen in Figure 9, for stopes in the 2000 t region, the ANN is substantially more accurate. However for the 169 t stopes, the ANN is vastly inaccurate due to limited data sets. This was one of the limitations of an ANN that was highlighted by Krogh (2008) and one that was experienced in this study. However for the majority of the data sets, the ANN is much more accurate than current practices. This was demonstrated by the Class 3 ANN (1000–5000 t) where the majority of data sets are found. The ANN in this case was much more accurate than current Plutonic Practices (0.88412 versus 0.79986). This highlighted the importance on the number and quality of data sets required for ANN development as well as the effects that over-fitting can have on an ANN. Second analysis

<table>
<thead>
<tr>
<th>Class</th>
<th>Number of neurons</th>
<th>R</th>
</tr>
</thead>
<tbody>
<tr>
<td>Class 1</td>
<td>35</td>
<td>0.75471</td>
</tr>
<tr>
<td>Class 2</td>
<td>25</td>
<td>0.86163</td>
</tr>
<tr>
<td>Class 3 (0–1000 t)</td>
<td>25</td>
<td>0.69067</td>
</tr>
<tr>
<td>Class 3 (1000–5000 t)</td>
<td>35</td>
<td>0.88412</td>
</tr>
<tr>
<td>Class 3 (5000–10 000 t)</td>
<td>25</td>
<td>0.79219</td>
</tr>
<tr>
<td>Class 3 (10 000 t+)</td>
<td>25</td>
<td>0.93018</td>
</tr>
</tbody>
</table>

*TABLE 3*  
Prediction results comparison.

**FIG 8** – Artificial neural network architecture for Plutonic Gold Mine.
CONCLUSIONS

Overbreak is an unavoidable consequence of longhole stoping and can result in large financial losses to any underground operation. While many researchers in the past have focused on identifying parameters affecting stope overbreak and implementing methods of control, few have focused on predicting overbreak accurately. Predicting stope overbreak accurately would be the first step in developing an effective overbreak management and blasting reconciliation system. Empirical methods would prove to be ineffective in predicting stope overbreak due to the high number of variables that exist and the relationship between these variables is unclear. ANNs would therefore prove to be a more effective method for predicting stope overbreak as they are able to identify relationships between input variables by utilising mathematically weighted algorithms and then use them to solve for an output variable.

After developing an ANN that could predict stope overbreak a comparison between the ANN predictions and current methods employed at Plutonic was conducted. Correlation coefficient between observed and predicted overbreak were 0.862 (ANN) and 0.873 (Plutonic Overbreak Prediction Practices) while tonnage variances between predicted and observed overbreak were -273 t (ANN) versus +14 800 t (Plutonic). While correlation coefficient would indicate that the ANN was less accurate than current methods employed, further analysis indicated that the problem of over fitting had occurred and the ANN was in fact more accurate.

The results in this paper have proven that ANNs can predict stope overbreak at Plutonic Gold Mine. The ANN stope overbreak prediction system can save time for a low initial investment and produce a significant cost saving. The results indicate that the method here could be adopted at other mine sites to predict stope overbreak. Furthermore the results from this project highlight how ANNs can be used to solve complex problems in the mining industry.

Recommendations

The following recommendations were derived from this project:

- Plutonic should implement an ANN-based overbreak prediction system as a tool for comparison so engineers can compare the ANN and their predictions and make an more informative estimation of stope overbreak.
- Each new stope data set should be imported back into the ANN to improve accuracy.
- Plutonic should start recollecting geological and geotechnical data and construct different ANN to improve the productivity of the mine.
- Barrick should conduct similar trials at operations, which could also benefit from predicting stope overbreak more accurately.

ACKNOWLEDGEMENTS

The authors would like to acknowledge Jeff Brown and Chris Vecchio for their assistance in providing data.

REFERENCES


A Comparison of Methods to Create the Initial Void for Longhole Open Stoping Blasting

K Lee¹ and A Halim²

ABSTRACT
This study is based on confidential case studies collected from a Western Australian gold mine, known from here on in as ‘Mine X’. The primary mining method used at Mine X is longhole retreat open stoping with paste backfill. The main objective of this research project is to investigate which method/s is the most appropriate to develop the initial void for stope blasting at Mine X.

Two methods were investigated during this research; conventional longhole rising and slashing against paste. Rising is a very common method, used frequently at Mine X as well as in the broader mining industry as determined through a literature review. Because of the common use of rising, it was expected to be a more reliable option resulting in a higher success rate. Slashing against paste, however, is a new method to the industry and not well documented, but was trialled as a possible cheaper option. These two methods were compared in terms of: success rate; drill and blast costs; limitations for their use; and benefits derived from combining the two methods.

The analysis of the case studies revealed several key points. Firstly, the success rate estimated by comparing stope design geometry and final surveyed shape showed that both methods achieved the same success rate of 78 per cent. The cost analysis determined that the average rise required more than double the costs per tonne of rock than slashing against paste. Slashing against paste was found to be a much more restrictive option, requiring new stopes to be situated next to paste filled stopes of specific dimensions. Rising does not have these restrictions; however, more often required slot drives to be developed prior to stoping than slashing against paste. Combining the two methods proved to be a beneficial option having the advantages of both methods. Overall this study recommends, for Mine X, the use of slashing against paste over rising in situations where slashing against paste is permitted. All other situations should consider the use of rising or a combination of rising and slashing against paste.

INTRODUCTION
Mine X is a gold mine located in the Goldfields region of Western Australia. The mine utilises longhole open stoping as its main mining method with the use of paste as backfill. The annual material mined at Mine X is just less than one million tonnes. Due to confidentiality reasons, the mine cannot be identified, hence, it is written as Mine X in this paper.

The orebody at Mine X changes throughout the mine; however, the focus for this research is limited to the stopes in the ‘T area’ of the mine. These single level stopes range in size from narrow vein stopes (4 m width) to bulk stopes (up to 20 m width), and up to approximately 22 m in height.

This project looked into the following problem: which method/s are the most appropriate to create the initial void for stope blasting at Mine X?

Rising
Traditionally, Mine X produced the initial void for stoping using a conventional longhole rise, followed by production rings into the rise void. A rise is a vertical or steeply inclined opening, traditionally driven in the upwards direction (Tatiya, 2005). Rising uses closely spaced drill holes and uncharged reamer holes to create void for the blastholes of the rise to fire into. This is designed using burncut techniques (Orica Technical Services, 2008) as shown in Figure 1. Based on the authors’ industrial experience, rises are the traditional way in which void space is created for stoping and this is a well-established practice, which has developed significantly over time.

Slashing against paste
Recently, the new method of slashing against paste, has been trialled to open voids instead of rises, in an attempt to reduce costs associated with drilling and blasting. Slashing against paste requires firing forward dumping production rings towards a paste filled stope. The firing of the dumped production rings propels the blasted rocks forward, compressing the paste, before falling into the drive below and creating a void (Anonymous Engineer at Mine X, November 2005).
2012, pers comm). Figure 2 shows how a slashing against paste design used at Mine X looks.

Whilst slashing against paste has been practiced in several mines around the Western Australian Goldfields region, there are no literatures that could be found about this method, other than personal communication with engineers who have used the method.

Objectives
To compare the aforementioned two methods, the following objectives were investigated:

- How rises are used as the current method?
- How has slashing against paste been used?
- How do both methods perform compared to their design?
- How do the drill and blast costs associate with each method compare?
- In what specific situation each method can be used?
- What benefits can be achieved from combining the two methods?

Methodology
To complete this project, several steps were taken as follows:

- data collection from Mine X
- comparison between blast designs and results
- determine which factors affected success
- calculate costs
- compare limitations
- analyse combining methods in one stope.

Data collected
The collected data was for all stopes completed in the ‘T area’ of the mine during January 2012 and July 2013. This included
27 rise designs and nine slashing against paste designs. The data required were stope notes, drill plans and charge plans, stope reconciliations, stopes design files, paste data and drilling and explosives costs.

**Compare design to results**
To determine whether both methods have worked well at Mine X and which method has worked better, the success rate for each method was calculated. This was determined by comparing the stope shape designed in Surpac™ and the cavity monitoring system (CMS) survey taken after stope extraction. As all stopes will have some amount of over- or underbreak, stopes were classified as ‘successful’ if the level of over- and underbreak experienced did not significantly contribute to underperformance for the rest of the stope. If there was a significant amount of over- or underbreak, the stope was classified as ‘unsuccessful’.

**Factors affecting success**
A variety of factors can contribute to the success of a rise or slashing against paste design. To determine which factors most impact the results a series of analyses were carried out comparing the successful and unsuccessful designs. These considered both engineering design variances as well as operational inaccuracies where possible.

The following factors were considered:
- drill hole deviation
- incorrect charging
- length of drill hole design
- angle of inclination of drill holes
- width and height of stope
- drill hole and reamer diameters
- drill rig used
- paste properties for slashing against paste.

**Cost analysis**
The drill and blast costs for the all the stopes can be calculated to determine which method is the cheapest option. The costs associated for drilling and blasting activities were obtained from Mine X and the total cost for each rise and slashing against paste design was calculated.

To allow easier comparison between different sized stopes, the tonnes of rock designed to break for each design were obtained from Surpac™ and the cost per tonne of rock calculated for both drilling and blasting.

**Limitations**
This section aimed to determine in which situations each of the methods can be used based on the case studies analysed from Mine X. It also considered whether slot drives were used and in which future stopes they are likely to be necessary for each method.

**Combination of methods**
From a previous trial, it was recognised that both slashing against paste and a downhole rise were used to open a particularly tall stope. This analysis aimed to determine what benefits were derived by combining the two methods in the one stope. The possible advantages and disadvantages were explored by following this particular stope through the previous analyses; overall success of the stope, factors affecting this success, cost analysis and limitations to the design.

**ANALYSIS AND RESULTS**

**Comparing results to design**
Whilst the rise method has been used three times more than the slashing against paste method during the trial period, the success rate for both methods is the same at 78 per cent. This was calculated from the percentage of ‘successful’ stopes during the trial period. These results suggest that both methods can be used in the right situation. It is especially interesting to note, that the slashing against method has been very successful given it has only been used for such a short time, whilst the rise method is a practiced method, used for many years in the ‘T area’, yet still has the same success rate in the time frame sampled.

**Factors affecting success**
Analysis was performed into which engineering design factors most affected the success of both rises and slashing against paste in the case studies. A number of other operational factors were briefly considered, such as drilling accuracy and charging accuracy, however, these were difficult to detect and were not considered in the analysis.

The main reasons for unsuccessful rises in this study appear to be flatter inclination of the rise, long lengths drilled with a smaller drill rig, smaller diameter reamers and the geology in particular areas. These may possibly be improved by drilling more vertical rises, using the larger drill rig with larger diameter reamers for longer rises and trying new ways to open some of the ‘trickier’ areas.

For slashing against paste, there was no real correlation between the factors analysed and the success of the stopes. This is most likely due to the large variety of drill designs and differences between designs, as the method is still being trialled and improved. Also less data was available to see any clear trends. With more trials it may be possible to gain a better understanding of which factors influence the success of slashing against paste the most.

**Cost analysis**
The drill and blast costs for the all the stopes were calculated and compared between the different methods.

For drilling costs in cost per metre of drill hole ($/m) was obtained from Mine X. It was assumed for the calculations that the cost of drilling holes was the same for both drill rigs used on-site. It was also assumed that the cost of drilling reamer holes was the same as drilling a standard blasthole two or three times, depending on the number of passes taken to ream the pilot hole. The drilling cost for the rise or slashing against paste design was determined based on the number of drill metres in the drill plans issued to the longhole drillers and the cost per tonne was then calculated for comparison. The following steps were followed to achieve this:

- total drill metres for each design obtained from engineering drill plans
- cost per design calculated (cost per design = drill metres × cost per metre)
- total tonnes per design determined from Surpac™
- cost per tonne calculated (cost per tonne = cost per design / tonnes).

From the drill and blast cost results per tonne of rock for each method shown in Figure 3 it can be seen that the cost per tonne for drilling rises is much higher than the cost per tonne for drilling slashing against paste designs. This is mostly due...
to rise designs containing more drill holes and the additional lengths required to drill reamer holes.

For blasting costs, only the cost of primers and detonators were considered in this analysis. These are the most substantial cost for explosives, and the other costs for operators, ANFO, detonator cord, etc., would be very similar for both methods. Two main types of detonators are used at Mine X: programmable electronic delay detonators (iKon) and non-electric delay detonators (millisecond/long period (MS/LP)). The cost of these different detonators is considerably different; with iKon detonators costing almost double that of MS/LP detonators.

To calculate the blasting cost, the number of detonators required for each design was obtained on the charge plan issued to the charge-up crew. From here, the cost per design was calculated. To allow comparison between different sized stopes, the cost per tonne of rock was calculated for each design. Simple steps were taken as for calculating drilling costs:

- total number of primers and detonators for each design obtained from engineering charge plans
- type of detonators obtained from charge plans
- cost per design calculated (cost per design = number of detonators × cost per detonator)
- total tonnes per design determined from Surpac™
- cost per tonne calculated (cost per tonne = cost per design / tonnes).

The average cost per rise design and slashing against paste design per tonne of rock is shown in Figure 3. Again, rising is a much more expensive option. This is predominantly due to the preference for the more expensive iKon detonators used in most rise designs. Rises require very accurate and slow timings to fire correctly as enough time must be allowed for the blasted rock from one blasthole to fall out of the rise before the next hole fires. To accommodate for this, the iKon detonators are usually used to allow for both time and flexibility in the timings. Slashing against paste however, is much faster and does not require as much accuracy and therefore the cheaper MS/LP detonators are sufficient.

**Limitations**

Two main limitations were considered for this analysis, the position of surrounding stopes and access drives compared to the new stope; and the requirement for slot drives.

Rising used at Mine X is very flexible in its use as most stopes in the ‘T area’ of the mine are paste filled. This means stoping next to a previous stope can be very similar to stoping next to a pillar. Also a variety of rises are used; uphole, downhole and blind rises depending on the position of access drives around the stope.

Slashing against paste, however, is limited in its use by its requirement to slash against a previously paste filled stope. The paste filled stope also determines the size of the new stope, as you cannot slash against a smaller or narrower stope. This would not provide void space from compressed paste next to some part of the stope, causing potential underbreak and freezing of the stope. Thus, it is necessary to stope using slashing against paste when the right sized paste filled stope is available.

Slot drives are used with rising to create a slot that spans the width of the stope in wide stopes. It can also be used in flat stopes where a rise positioned along the footwall of the stope would be placed on too shallow of an angle. An example stope where a slot drive was required to increase the angle of the rise is shown in Figure 4. Slot drives incorporate additional costs to the stope development, through the cost of developing an additional drive of the main development as well, as increased development time before the stope can be extracted. In certain situations, by moving the position of the rise, the rise is now developed half in waste and half in ore. This adds dilution to the stope once it is fired which can reduce the economic value of the stope.

For slashing against paste, the main constraint for width is the width of the adjacent paste filled stope. Therefore, if the paste filled stope is wide enough, a slot drive should not be required for the new stope. This reduces the costs associated with slot dives described previously.

**Combination of methods**

From the previous analysis carried out, it can be seen that whilst rising is the more flexible method, slashing against paste is a much cheaper option. In an effort to gain the benefits of both methods, a stope where both slashing against paste and rising were used to create the initial void was examined. The design for this stope is shown in Figure 5.

If only one method was used for this stope, slashing against paste would not be possible due to the small size of the adjacent paste filled stope, so the only other method which could be used would a rise of at least 24 m in length. This would require very accurate drilling and charging practices to successfully blast such a long rise and, based on the previous rises, has a lower chance of success. By incorporating both methods, only a 12 m downhole rise was required, with a much better chance of success. Also, the cost was greatly reduced by combining the two methods. For this design it was found that the cost per tonne for drilling and blasting was much less than the average rise cost and even less than the average slashing against paste cost in this case.

Whilst this analysis has provided some very good results and benefits, the situations where it would be best considered to combine the two methods was outside the scope of this study and should be considered in further studies.

**CONCLUSIONS AND RECOMMENDATIONS**

**Conclusions**

From the analyses and results found during this study, the following conclusions were made:

- both the rising and slashing against paste methods have worked equally well in the ‘T area’ of Mine X
- several factors such as length, angle of inclination, drill rig and reamer size selected greatly affects the success rate of rising, however, cannot always be easily chosen
• the analysis into possible factors influencing the success of slashing against paste was found to be inconclusive due to lack of data and any clear correlation in results

• the cost for drilling and blasting per tonne of fired rock for slashing against paste is less than half the cost of rising

• from the limitations, rising is very flexible however, can require additional development, whereas slashing against paste is restricted by the adjacent paste filled stope, but less often should require slot drives

• combining the two methods can be a beneficial exercise, which can gain the advantages of both methods.

Recommendations

For Mine X, the better method depends on the circumstances of the stope position. If slashing against paste can be used, it is recommended that it should be used to gain the benefits of cost and possibly less development. In all other situations, rising or a combination of both methods should be considered.

It is recommended that further study into which stopes would benefit most from using the combination of the two methods rather than a single method.

It is also recommended that with further usage of slashing against paste, the data collected from these stopes are analysed in an aim to determine the factors which most influence success.

REFERENCES

Mine X, 2011. Stope Note T870 Nth C.


INTRODUCTION

In recent times, sweeping changes have occurred with regards to the employment of Indigenous Australians. External forces driving corporate social responsibility initiatives have seen changes in public, governmental and private sector opinions on the role of Indigenous Australians. As such, many mining companies now offer specific indigenous training and employment programs that have been successful in placing Aboriginal people in a variety of mining roles. There have also been leaps and bounds made in the area of indigenous owned and operated companies, particularly contractors, who have secured work in the minerals industry.

This study analyses the indigenous employment policies and trends of three large mining companies and two mining contractors. On the basis of the reported quantitative and qualitative data, a maturity scale is advanced for indigenous employment practices. The study is of relevance to industry as the indigenous population of Australia is growing steadily, particularly in remote and regional areas where many mines are located.

The major benefits for industry are threefold: gaining information and data on their programs for internal usage in developing indigenous employment programs; increased social licence to operate as a result of corporate social action; and economic benefits of employing local staff in areas of the mine rather than fly-in fly-out (FIFO) / drive-in drive-out (DIDO) workforces. A maturity scale ranking companies who have taken part in this research has been compiled based upon data collected.

CURRENT STATE

Indigenous Australians

The indigenous peoples of Australia comprising Aboriginal and Torres Strait Islander (ATSI) people are the oldest continuing culture on earth and have lived in Australia for over 40,000 years (Albrechtsen, 2011). Prior to European colonisation they were the sole inhabitants of Australia and as such have a rich history. Mining has had a tumultuous relationship with Indigenous Australia throughout its history as described by prominent indigenous leader Professor Marcia Langton, as over the past 50 years we have seen ‘the transformation of the mining industry from the pillagers into the major investors in the indigenous world’ (Langton, 2013).

In 2006 the Australian Bureau of Statistics (ABS) census found that approximately 455,028 people, or 2.3 per cent of the population, identified as being of ATSI descent. This figure is, however, revised upwards to 2.5 per cent as a result of undercount and unknown indigenous status (ABS, 2006). The
most recent 2011 census found that this figure had increased to 669,736 or three per cent indicating a steady rise (albeit not as fast as non-indigenous population growth) over the five-year period (ABS, 2011). Figure 1 shows the indigenous population count from 1986–2006. The final statistics for the 2001–2013 period were expected to be released in mid-2013 by the ABS.

**Employment data**

Current figures for indigenous employment in the mining industry do not exist as an aggregated total. Figures for specific companies are generally published in company sustainability reports. In 2000–2001 an Australian Bureau of Agriculture and Resources Economics (ABARE) study was commissioned specifically to conduct a survey on indigenous people in mining. Unfortunately it has not since been updated.

A research paper published by the Centre for Aboriginal Economic Policy Research stated that in 2011 Non-CDEP (community development employment projects) indigenous employment in remote areas was 27 per cent. However, this is across all industries. Mining employment itself changed from employing 3300 indigenous people in 2006 to 7500 in 2011, a rise which has been largely associated with the mining boom (Gray, Hunter and Howlett, 2013).

**Community and mine locations**

As of 2008 90 per cent of indigenous peoples lived regionally or remotely in Australia as shown in Figure 2 (ABS, 2008). Analysis of the location of mineral tenements in Australia show that substantial overlap exists with the locations of indigenous populations. Thus excellent opportunities exist for indigenous employment in mining.

There has been little study into the reasons for disparity in employment levels given geographical proximity, however, socio-economic disadvantage as well as a lack of corporate policy are believed to be the primary drivers.

**Rio Tinto corporate policy**

Rio Tinto has managed to raise its indigenous workforce percentage from less than half a per cent in 1995 to just over eight per cent in 2008 (Rio Tinto, n.d.) This success is primarily attributed (by Rio Tinto to):

- family and community support programs
- prevocational training and development
- mentoring of indigenous employees
- cross-cultural education programs.

A specific case demonstrating Rio Tinto policy is highlighted by a study of the Argyle Diamond Mine located in the East Kimberley. The 2005 Indigenous Land Use Agreement (ILUA) stipulated that not only would Argyle be required to invest financially in the local area but that the indigenous community would also have a greater say in employment (Foley, 2011).

In 2011 there were over 50 indigenous apprentices and trainees employed at Argyle. Indigenous employees comprised 25 per cent of the total workforce of which 70 per cent live in the local area. Ten per cent of the frontline leadership positions at the mine were indigenous with a long-term goal for Rio Tinto Argyle to transition more indigenous people into higher level management (Foley, 2011).

**BHP Billiton corporate policy**

BHP Billiton (BHPB) released their Reconciliation Action Plan in 2008. It concentrated on three main objectives:

1. promoting cultural awareness
2. education, training and employment
3. promoting indigenous business opportunities.

BHPB set both medium- and long-term targets for the employment of indigenous people, basing their criteria on the locality of prospective employees, unemployment levels and operational requirements. Indigenous business opportunity is focused around the increased capacity of indigenous people to engage in the economy (BHP Billiton, 2008) and does not necessarily include employment directly within the minerals industry.

**Indigenous Land Use Agreements**

As seen at Argyle Diamond Mine ILUAs make a significant contribution to the employment and land use arrangements of indigenous peoples. The input of ILUAs between various State Governments and industry aims to have an additional 3000 people employed by 2020 (BHP Billiton, 2009). As of March 2012, 588 ILUAs have been registered with the National Native Title Tribunal (NNTT) (Boyer Lectures, 2012). ILUAs are very broad in scope, however, they typically involve employment and training and protocols for future development as key aspects (NNTT, 2008).

**METHODOLOGY**

**Interviews**

Industry interviews were the primary method of data collection for this research, allowing the authors to obtain one-on-one access to industry. Interviews allow experts to get their knowledge across in a way that is not possible through other methods, particularly because interviews allow a deeper insight into how people think and reflect (Folkestad, 2008).
The data collected from the interviews has been split into two categories: quantitative and qualitative data.

Quantitative data involves information such as indigenous employee numbers, dollar contract values, etc. This data was very important as it formed the basis of many of the key performance indicators (KPIs) used in developing the maturity scale.

Perspective and relevance are highly important when dealing with statistics. Quantitative data does not give a full understanding of each company’s indigenous employment maturity. Qualitative data gathered throughout the interview process enables the quantitative data to be put into perspective for comparison against other companies. Qualitative data also gives a chance for company representatives to discuss their feelings, experiences and opinions with regards to this topic. This is highly useful as indigenous issues can sometimes be polarising with more emotive responses from interviewees eliciting information that would not be able to be gathered from the quantitative data.

**Maturity scale criterion**

Using examples of maturity scales developed for other strategic partnerships (for example Lendrum, 2000), a scale and criterion was formulated for indigenous employment. Maturity levels are assessed considering four categories:

1. regressive (no strategy at all)
2. reactive (adapts to fit government legislation)
3. proactive (actively engages and is ahead of legislative change, positive outcomes achieved)
4. strategic (companies are regarded as best practice in industry, partnerships and alliances are created at an individual level as well as strategically with other companies, contractors and government).

Each of these levels is measured around a set of KPIs as well as analysis of the qualitative data presented by each company. The KPIs are:

- indigenous employment rate
- indigenous frameworks for recruitment, training and human resources
- indigenous contractor procurement
- planning for increased indigenous involvement
- specific ATSI policies in place.

**DATA**

**Rio Tinto Iron Ore**

Rio Tinto Iron Ore (RTIO) is a global iron ore producer, the second largest iron ore trader and part of one of the world’s largest mining companies. Australian operations are located primarily in the Pilbara region of Western Australia with 14 mines in operation. RTIO has multiple ATSI employment and training programs and has steadily increased its indigenous employee percentages from 2007. Table 1 shows the quantitative data gathered from RTIO.

RTIO also have taken part in a number of indigenous scholarships, cadetships, apprentices and trainees:

- 12 RTIO scholarships
- 18 RTIO cadets
- 49 RTIO apprentices
- 196 RTIO trainees (including indigenous participation program).

### Table 1

<table>
<thead>
<tr>
<th>Total employees</th>
<th>Total ATSI employees</th>
<th>ATSI percentage of workforce</th>
<th>Monthly retention rate (%)</th>
<th>Contract value ($ M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>13 309</td>
<td>1070</td>
<td>8.04</td>
<td>Not reported</td>
<td>950 (2012)</td>
</tr>
<tr>
<td>+ 519 (contractors)</td>
<td>+ 615 (2013 YTD)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.

In addition, in 2012 RTIO spent $4.85 M with 14 community partners and a number of charitable and community organisations.

**Downer EDI Mining**

Downer EDI Mining (DEDIM) is one of Australia’s largest mining contractors employing thousands of people in a variety of commodities and mining types across the country at both site and corporate level. DEDIM has a tailored Aboriginal and Torres Strait Islander Policy that sets out ‘high-level business commitments to achieving improved economic participation for Indigenous Australians’ (Downer EDI Mining, 2013). DEDIM was also the Indigenous Employment Award winner of the 2012 Australian Human Resources Institute Diversity Award. The award recognises the specialist recruitment and training methodology undertaken by DEDIM to recruit indigenous employees.

DEDIM also has a number of indigenous metrics that are measured against company progress each year and against set targets. These include: female indigenous employees showing a 42 per cent increase in the 2012 financial year; and total indigenous employment exceeding the target by 18 per cent. Table 2 shows the quantitative data gathered from DEDIM.

**MMG**

Headquartered in Melbourne, Australia, MMG is a mid-tier global mining company with operations spanning multiple continents and commodities. MMG boasts significant indigenous employment in its Australian operations including Century and Dugald River mines. Century Mine in particular is located adjacent to the community of Doomadgee and MMG has been able to employ a significant amount of the local populace. Table 3 shows the quantitative data gathered from MMG.

**Glencore**

Glencore is one of the world’s largest global diversified natural resource companies. Its Australian operations include coal, copper, cotton, grain and oilseeds, nickel, zinc and technology, with mines regionally located, some very near local indigenous communities. For the purposes of this
These open days, which was attributed partly to social norms. Only about one third of eligible students attended traineeships and cadetships) for local students in the Pilbara region. RTIO noted that they had run specifically encourage young indigenous people to take up roles in the mining industry. Glencore in particular have had success with high school programs designed to specifically encourage young indigenous people to take up roles in the mining industry. RTIO noted that they had run open-days and mine site tours (separate to the structured traineeships and cadetships) for local students in the Pilbara region. Only about one third of eligible students attended these open days, which was attributed partly to social norms.

### RESULTS

#### Industry trends

**Education and training**

All companies interviewed (with the exception of MSCU) identified education and training as key to increasing indigenous employment in the Australian minerals industry. Strong programs and policies were identified in companies that ranked high on the maturity scale indicating the importance of these programs. In particular, highly ranked companies invest in education programs that begin whilst prospective employees are still in high school. Glencore in particular have had success with high school programs designed to specifically encourage young indigenous people to take up roles in the mining industry. RTIO noted that they had run open-days and mine site tours (separate to the structured traineeships and cadetships) for local students in the Pilbara region. Only about one third of eligible students attended these open days, which was attributed partly to social norms.

### Company rankings

Analysis of all companies current practices along with a comparison to the literature (Lendrum, 2000) returned the following ranking:

- RTIO – strategic
- DEDIM – strategic
- Glencore – proactive
- MMG – strategic
- MSCU – reactive.

### Indigenous employment maturity scale

Figure 3 shows the indigenous employment maturity scale developed as the culmination of the work undertaken for this research.

### CONCLUSIONS

This paper on the research topic of indigenous employment in the Australian minerals industry has been completed with quantitative data from two primary sources: the ABS and records of various companies whose progress were examined. With the data gathered through these methods it was possible for an indigenous employment maturity scale to be developed showing the ranking of the companies who participated relative to each other and relative to industry progress.

This report would not have been possible without the generous support provided by the companies who took place. However, for future work the following recommendations are advanced:

- the scope be expanded to include interviews and data from a greater number of companies to give a greater picture of the current situation.

### TABLE 3

<table>
<thead>
<tr>
<th>Total employees</th>
<th>Total ATSI employees</th>
<th>ATSI percentage of workforce</th>
<th>Monthly retention rate (%)</th>
<th>Contract value ($ M)</th>
</tr>
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<tr>
<td>2789</td>
<td>266</td>
<td>9.55</td>
<td>Not reported</td>
<td>Not reported</td>
</tr>
<tr>
<td>1789</td>
<td>196</td>
<td>10.96</td>
<td>Not reported</td>
<td>Not reported</td>
</tr>
</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.

### TABLE 4

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<th>Total employees ('000s)</th>
<th>Total ATSI employees</th>
<th>ATSI percentage of workforce</th>
<th>Yearly retention rate (%)</th>
<th>Contract value ($ M)</th>
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</thead>
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<td>351</td>
<td>63</td>
<td>18</td>
<td>72</td>
<td>N/A</td>
</tr>
</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.

### TABLE 5

<table>
<thead>
<tr>
<th>Total employees ('000s)</th>
<th>Total ATSI employees ('000s)</th>
<th>ATSI percentage of workforce</th>
<th>Monthly retention rate (%)</th>
<th>Contract value ($ M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>40</td>
<td>0</td>
<td>0</td>
<td>N/A</td>
<td>0</td>
</tr>
</tbody>
</table>

ATSI – Aboriginal and Torres Strait Islander.
the companies that participated in the study be approached to include company wide data (only three of five company participants provided the data for all sites and operations) given further time the companies that participated in this research be re-examined in the future to gauge ongoing progress and track results over time. It was found that two of five companies examined were sufficiently mature to qualify for the ‘strategic’ maturity category with two more qualifying for the ‘proactive’ level. This is a positive sign for the mining industry and can be seen as an indicator of where the industry is heading as a whole in the future.

ACKNOWLEDGEMENTS
We would like to thank the following people for their time and support in assisting with data collection: from Rio Tinto Iron Ore – Alison Smith and Lynette Upton; from MMG – Meg Frisby and Rodney Francisco; from Glencore – Joanne Pafumi; and from Downer EDI Mining – Bevan Whitby.

REFERENCES
Comparison of Two Excavations Systems for the Mining of Lunar Regolith

M T Lucas¹ and P C Hagan²

ABSTRACT
A study was undertaken to compare the feasibility of conventional pick cutting systems and pneumatic excavation systems in an off-world environment. This involved a technical and economic analysis of these methods under certain lunar conditions including a determination of the cuttability of simulate lunar regolith in order to assess the excavation effectiveness of conventional mining machines.

The technical and economic feasibility was determined by analysing different lunar conditions and the possible effects that these conditions would have on the excavation methods. These theoretical analyses were supported by fundamental mining knowledge and current literature on the subject matter.

The estimation of cuttability was based on a newly designed, modified version of the Newcastle-upon-Tyne cuttability test that has greater sensitivity at low force levels. The sample tested was the Australian Lunar Regolith Simulant-1 (ALRS-1). The cuttability forces and abrasivity was assessed across a range of packing densities with the results used to perform a sensitivity analysis on the efficiency of pick cutting methods in excavating soil.

While more research and development is necessary to be undertaken on the prototype pneumatic excavation system, it was found that both excavation systems have the potential to remove lunar regolith across a range of densities. It was also concluded that while neither system was economically feasible for lunar excavation based on current technology, the pneumatic excavation system was shown to be more advantageous under the conditions analysed.

INTRODUCTION
While mankind has long had an interest in space exploration, it has only been in recent times that it has had the capacity to achieve this. Ever since the Apollo missions in the 1960s and 1970s there has been an interest in the creation of a lunar settlement on the moon. With this comes a need for resource extraction to support the settlement that, with today’s advances in technology, is steadily becoming a feasible option. This situation gives weight to the objectives of this study that was to analyse and evaluate the feasibility of conventional and pneumatic mining methods particularly with respect to the excavation of the lunar surface material or regolith.

Project objectives
The project entailed both a theoretical analysis and experimentation. Testing the efficiency of conventional mining systems on lunar regolith requires a determination of the cuttability of the lunar material. The tests made use of a fine-grained simulant lunar regolith, ALRS-1, and a newly designed cuttability machine. It also involved evaluation of the effectiveness of each rock breakage technique against this simulant.

In addition, a theoretical analysis of each system in a lunar environment was undertaken, focussing on seven specific conditions, which is likely to influence the system efficiency, these being: gravity, vacuum environment, soil abrasivity, electrostatic soil, soil compaction, dust suppression and transport costs.

The study evaluated the two excavation systems, and recommends the most feasible technique. It also outlines some of the issues with the use of each system and potential concerns that need to be addressed.

COMPARISON OF EXCAVATION SYSTEMS
Whilst the excavation efficiency of each method is important in analysing system effectiveness, there are also a number of lunar conditions that influence the feasibility of each system.

Gravity
Gravity is a significant condition that changes between a terrestrial and lunar environment, with one-sixth of Earth’s gravity on the lunar surface.

Conventional pick mining methods rely on the weight of the machine to provide the sumping force for material cutting. As the equation shows, the reduction in gravity reduces the effectiveness of the machine as its sumping force is reduced.

\[
\text{sumping force} = \text{mass} \times \text{gravity} \times \text{coeff friction}
\]
However, pneumatic excavation systems are not influenced by gravity and therefore there is no known effect of this on the machines efficiency in a reduced gravitational environment.

**Vacuum environment**

The main issue that the vacuum environment poses regarding use of conventional mining machines is the likely increase in surface to surface abrasive friction between the picks and regolith increasing the rate of pick wear.

In an environment lacking an atmosphere, the internal angle of friction and strength of soils is increased. It has been claimed that an increase in friction would increase the cutting forces as well as increasing pick wear (Prado, 2013; Podniesiks and Roepke, 1985). The US Bureau of Mines used experimental data to determine that the vacuum environment increased friction by between 1.5 and 60 fold compared to the friction experienced with terrestrial pick cutting (Prado, 2013; Podniesiks and Roepke, 1985).

Additionally, the lack of an atmosphere in a lunar environment would adversely affect the rate of heat dissipation. As heat cannot be dissipated at the same rate as on earth, the cutting picks would quickly build up heat and thereby reducing the relative difference in hardness between the cutting tool and lunar material.

This increase in friction would not only reduce the efficiency of the pick cutting, but also costs of cutting with much higher pick consumption.

Pneumatic excavation systems would require the use of a to excavate the regolith. In a vacuum environment, these systems could only operate in a self-contained atmosphere. The system would require creation of this atmosphere sealing off the excavation hole through the use of an expanding ‘packer’ as well as grout (Bernold, 2013). If an efficient seal cannot be made, this self-contained atmosphere could not be created, as gas could constantly escape, which would make this excavation system partially redundant.

**Regolith abrasivity**

The main issue with abrasion for conventional mining methods is pick wear. A significant issue in relation to abrasivity of the regolith is lubrication. In a lunar environment, due to the vacuum nature, standard wet lubrication cannot be used (Honeybee Robotics, 2013). Instead, another means such as use of a dry or recyclable lubricant would be required.

There are also several possible issues that the abrasivity of lunar regolith may have on the pneumatic excavation system. The first being it may cause wear on the inside of the suction piping. While the quantitative effect of this is not known, it will lead to faster wear that if not planned for will eventually create holes diminishing the effectiveness of the system, as the pneumatic gas would just escape from the closed system. More regular replacement of the piping would be required then similar systems used on earth. Another potential concern is damage to the inflatable packers that are used to seal the hole. The soil abrasion will wear or cut the rubber packer, which will also require more regular replacement (Geopro, 2013).

**Electrostatic regolith**

The lunar regolith has electrostatic properties that would affect conventional mining machines, as it would tend to coat the picks and exposed moving parts, increasing component wear. It would also increase thermal adsorption, due to the dark material, which was observed on previous lunar equipment, which continually overheated (Bell and Phillips, 2005).

The pneumatic excavation system could also be influenced by the electrostatic properties, as the piping could be coated reducing the suction and hence efficiency of the system. The heat issues described previously would also apply to this equipment.

**Regolith compaction**

Whilst compaction of the regolith is likely to have a greater effect on excavation efficiency, it also influences the feasibility of these operations.

One issue that would affect both systems is a loss of traction over loose soil, which was observed during the Apollo 15 mission (Beale, 2009). Additionally, this would affect the coefficient of friction for conventional machines, further reducing the sumping force and excavation efficiency.

Also, soil compaction would influence the efficiency of pneumatic excavation systems, as blasting may be required in denser regolith to allow the system to operate. This would further increase the transportation costs and reduce its economic feasibility.

**Dust suppression**

Dust suppression is a significant concern, as the lower gravity and vacuum environment cause much slower dust settlement.

Conventional mining methods often create a large amount of dust during excavation. As water is not readily available on the moon, and continual supply from the earth would not be practical, alternate means of dust suppression would be required.

As the pneumatic excavation system operates within an enclosed atmosphere, no regolith is expelled from the excavation and no dust suppression would be needed, making this system more beneficial in regards to dust suppression.

**Transportation costs**

Transport costs are a significant factor in the feasibility of both operations. Mackey, Gaskins and Lally (1996) claimed that the transportation of material to the moon is likely to be between $54 000 and $121 000/kg of payload. Hence both operations would have a significant initial capital outlay. As pneumatic machines do not rely on mass for efficient operation, the mass of these machines could be minimised through the use of lightweight materials or other design methods to reduce transportation costs. Conventional machines are required to make a compromise between cutting efficiency and transport cost, as the reduced mass reduces the sumping force of the machine, but would also reduce the transport cost.

Both machines would require replacement parts for maintenance, which would be influenced by the operational reliability of each system. Additionally, conventional mining systems would require replacement picks, which would be a significant cost due to the higher pick wear, as well as a possible continual supply of material for dust suppression, if no other option is determined.

Pneumatic excavation systems may require replacement compressed gas, depending on the effectiveness of gas recycling in the closed-circuit system, as well as grout and replacement funnel heads.
EXPERIMENTAL APPARATUS

Requirement for new test facility

Due to the nature of ALRS-1 lunar regolith being an unconsolidated material, it is likely to have low cutting forces relative to rocks. An established test procedure developed to assess the cuttability of rocks is the Newcastle-upon-Tyne test (Roxborough, 1987). Its use in this case was limited, as it did not have sufficient sensitivity to measure the expected low forces in cutting the lunar regolith hence an alternate approach was required.

Apparatus

The experimental apparatus designed and constructed for the tests is shown in Figure 1. Essentially it is a scaled-down version of the Newcastle facility. The apparatus comprises a pneumatically driven arm to provide the motive force to the cutting tool mounted on a steel support frame, and associated electrical components. Attached to the front of the piston is a detachable cutting head with force transducers.

Lunar regolith sample preparation

The Australian Lunar Regolith Simulant, also known as ALRS-1 or ALS-1, was developed by Professor Leonhard Bernold at the UNSW Department of Civil and Environmental Engineering comprised mainly of basaltic soils obtained from the northern region of Sydney, Australia (Garnock and Bernold, 2012). The basalt material was sieved so that the resultant particle size distribution was in accordance with actual lunar soil samples.

Sample preparation involved water being added to the regolith before being compacted into a cylindrical shape. Zeng et al (2010) performed the Proctor compaction test on a similar lunar regolith simulant, JSC-1A, and determined the optimum moisture content for compaction was around 13.5 per cent. This was used as a baseline for estimating the amount of water to add to each sample for compaction.

Compaction was achieved using a ram and cylindrical mould with sufficient material added to the cylinder equivalent to a third of the height each time. This process was repeated until the mould was filled to the required level. The sample was then be left to dry in a drying oven at 500°C for up to three hours. After drying, each sample was weighed and the dimensions measured to determine the sample density.

EXPERIMENTAL PROGRAM

Testing methodology

Sample density tests

The apparatus was operated in a similar method to the Newcastle-upon-Tyne cuttability facility, with the cutting head driven parallel to the axis of the sample at the required depth as shown in Figure 1. The cutting and normal forces were measured using force transducers. After each test, the sample was rotated 90° and the test repeated with up to four test replications undertaken with each sample. Each test was performed using a 10 mm wide cutting head at a cutting depth of 5 mm. Tests were conducted on regolith samples with densities varying between 1.6 and 2 t/m³.

Cutting depth tests

In this series of tests, the effect of cutting depth in cutting was examined. Initially tests at three depths of 3, 5 and 10 mm were to be undertaken. It was later found that the 3 mm tests had to be eliminated due to the load cell interfering with the sample during cutting.

Abrasivity

After each regolith sample was tested, the cutting head was removed and weighed. The difference in mass was used to determine the level of abrasivity of the material at different densities, in units of milligrams of wear per metre of cutting.

Results

Cutting forces

Figure 2 shows the variation in forces with regolith density. All tests were conducted a constant depth of 5 mm with three test replications at each level of density. Over the range of densities considered, there is a linear increase in cutting and normal force with density.

Cutting head depth

The results for the cutting head depth tests are summarised in Table 1. These results showed that the forces of the soil increased as expected, and that the specific cutting forces, given as the amount of force per millimetre depth of cut, were as theoretically calculated.

Regolith abrasion

The results from the soil abrasion measurements are shown in Table 2. These results indicate that the soil has minimal wear on the pick which in this study was not tungsten carbide as used in the standard cutting test but made of aluminium, a much softer metal that would exaggerate the wear rate. However, this wear is likely to be higher under lunar conditions, as discussed previously.

Analysis

These results were used to determine the effectiveness of conventional mining machines in soil excavation. Of these machines, continuous miners are generally used to excavate
low strength material. These machines are either limited by torque or thrust. For the purpose of quantifying its effectiveness in this application a JOY 14CM9A Continuous Miner, with machine specifications as shown in Table 3, was considered in cutting a lunar regolith having a density of 1.8 t/m³.

A sensitivity analysis was undertaken based on the machines specifications for both the torque limited and thrust limited scenarios.

Analysis of the torque limited scenario considered the impact of changes in several machine parameters on cutting depth and machine mass the results of which are graphed in Figures 3 and 4 respectively. The horizontal dashed line in each of these two graphs indicates the ceiling above which the machine changes from being torque limited to thrust limited.

These two graphs indicate that a reduction in either motor power or efficiency, or conversely an increase in cutter speed would reduce cutting depth and effective machine mass. The reduction in mass is as a result of the reduction in depth, as a smaller cutting depth would require a lower sumping force.

The sensitivity analyses for the thrust limited scenario considered depth of cut and motor power. The effect of these changes can be seen in Figures 5 and 6.

In this thrust-limited scenario, changes in cutting speed and motor efficiency had no effect on cutting depth as shown in Figure 6, unlike the torque-limited scenario. However, reducing motor power increased the depth of cut.

---

### Table 1
Specific cuttability forces of Australian Lunar Regolith Simulant (ALRS-1).

<table>
<thead>
<tr>
<th>Regolith density (t/m³)</th>
<th>Specific cutting force (N)</th>
<th>Specific normal force (N)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>mean</td>
<td>max</td>
</tr>
<tr>
<td>1.6</td>
<td>1.8</td>
<td>5.4</td>
</tr>
<tr>
<td>1.8</td>
<td>7.2</td>
<td>11.4</td>
</tr>
<tr>
<td>1.9</td>
<td>11.0</td>
<td>17.8</td>
</tr>
<tr>
<td>2</td>
<td>13.2</td>
<td>17.4</td>
</tr>
</tbody>
</table>

---

### Table 2
Summary of soil abrasion results.

<table>
<thead>
<tr>
<th>Soil density (t/m³)</th>
<th>Mass loss (mg/m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.5</td>
<td>0.1</td>
</tr>
<tr>
<td>1.6</td>
<td>1.6</td>
</tr>
<tr>
<td>1.8</td>
<td>3.0</td>
</tr>
<tr>
<td>2.0</td>
<td>2.6</td>
</tr>
</tbody>
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---

### Table 3
Joy 14CM9A Continuous Miner specifications.

<table>
<thead>
<tr>
<th>Machine factor</th>
<th>Joy 14CM9A</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass</td>
<td>53.6 t</td>
</tr>
<tr>
<td>Motor power</td>
<td>150 kW</td>
</tr>
<tr>
<td>Radius of cutting drum</td>
<td>432.5 mm</td>
</tr>
<tr>
<td>Bit tip speed</td>
<td>3.28 m/s</td>
</tr>
<tr>
<td>Cutting drum width</td>
<td>3350 mm</td>
</tr>
<tr>
<td>Pick spacing</td>
<td>57 m</td>
</tr>
</tbody>
</table>

---

**FIG 2** – Variation in cutting and normal forces with regolith of differing densities.

**FIG 3** – Sensitivity analysis in the torque limited scenario on the change in depth of cut.

**FIG 4** – Sensitivity analysis in the torque limited scenario on the change in mass.
in Figure 5. Consequently the machine would effectively remain above the ceiling and hence would always be torque limited. Only reductions in machine mass and the coefficient of friction of more than 20 per cent would cause the machine to become thrust limited, these two parameters being related to sumping force.

In terms of motor power, Figure 6 shows power varies inversely, as would be expected, with motor efficiency. However even at the limit of an increase in efficiency of 25 per cent, the machine remains torque limited. Reductions in the other machine parameters of machine mass and cutter speed of more than 20 per cent would cause the machine to become thrust limited.

As machine mass is a significant factor in the effectiveness of material excavation for conventional mining machines, an analysis was undertaken on the mass required to excavate the lunar regolith across a range of densities, from a minimum density of 1.5 to 2 t/m$^3$. The results of this analysis can be seen in Figure 7.

As the graph shows, conventional mining systems can excavate lunar regolith without any significant challenge. A study by Bernold (2013) has found that pneumatic excavation systems can adequately excavate lunar regolith up to the maximum density. Consequently systems have the capability to excavate lunar regolith.

CONCLUSIONS

It has been found that both conventional and pneumatic excavation systems have the potential of excavating lunar regolith. In terms of technical and economic feasibility, the pneumatic excavation system is likely to be more advantageous for excavation in a lunar environment.

The feasibility of each system considered seven lunar conditions; gravitational force, vacuum, soil abrasion, electrostatic soil, soil compaction, dust suppression and transport costs. Of these conditions, the pneumatic excavation system was more adaptable in all conditions except for soil compaction. Transportation costs for both systems remains the main impediment in terms of overall economic sustainability.

The efficiency of conventional pick cutting on lunar regolith was determined using a newly design cuttability test facility and a regolith simulant material. The facility confirmed a low level of excavating forces over a range of compaction densities. The level of abrasion of the aluminium cutting tool indicated minimal amount of wear compared to conventional rock cutting.

ACKNOWLEDGEMENTS

The authors would like to acknowledge the support during this project of Professor Leonhard Bernold, UNSW Civil Engineering, and Associate Professor Serkan Saydam, UNSW Mining Engineering. Professor Bernold provided the ALRS-1 sample and information on his prototype pneumatic excavation system.

The authors also acknowledge the assistance of Mr Kanchana Gamage, Laboratory Manager, in the design and construction of the test cutting facility.

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Using Clustering Methods for Open Pit Mine Production Planning

H Ren¹ and E Topal²

ABSTRACT

Typical mine planning process includes creating a mining block model, applying the ultimate pit limit analysis and creating mining cuts for production scheduling. Mixed integer linear programming (MILP) has been used extensively for optimal mine production scheduling of open pit mines. One main obstacle for large scale mine scheduling is the size of the problem. In large scale open pit mines, the number of blocks are too numerous to allow an optimal solution to be developed in a reasonable amount of time frame. However, the problem can be simplified by aggregation of mining blocks to create the mining cuts. The objective of this research is to analyse and validate a clustering algorithm for mining block aggregation. Popular cluster algorithms are reviewed and compared for the effectiveness of clustering for production scheduling purposes. Fuzzy C-Means (FCM) cluster method is used to partition mining benches and it is tested against the efficiency and practicality of mine production scheduling. The block aggregation algorithm is validated with a case study of a copper deposit. The net present value (NPV) of the production schedule which is created by using clustering algorithm is $2.25 M higher than the traditional pushback based production schedule.

INTRODUCTION

Mine planning is critical and directly influences the value of the project which is important to the shareholders and the community. A typical mine planning process involves finding the ultimate pit limit from a resource block model, developing a number of mining cuts and deriving a mine schedule which maximises the net present value (NPV). Mixed integer linear programming (MILP) has been used for optimising the production schedule of open pit mines since 1960s (Hustrulid and Kuchta, 2006). The objective function of the MILP formulation is to maximise the NPV of the mining operation subject to a set of constraints such as mining, milling and refining capacities, sequencing, blending requirement and minimum mining width (Askari-Nasab, Awuah-Offei and Eivazy, 2010). For a practical size mining block model, the time to find the optimal block-by-block mine life schedule on a powerful computer is too long or sometimes it is impossible to obtain a solution. Many researchers have studied the problem of mine production scheduling in open pit mines. Some of the approaches to the scheduling problem include heuristics, Lagrangian relaxation, parametric methods, dynamic programming techniques, mixed integer linear programming techniques and the application of artificial intelligence algorithm such as simulated annealing, genetic algorithms and neural networks. All of those approaches have limitations. Due to the complexity and size of the problem, most of these approaches suffer one or more of the following limitations: they cannot provide what is needed for most of the constraints that arise; they yield suboptimal solutions in most cases; or they can only handle small sized problems (Caccetta and Hill, 2003).

The constraints and variables can be reduced by aggregation of mining blocks. Block aggregation is a very important step in the preprocessing of data sets before optimisation. If it is done properly, despite the loss of detail, it enables difficult problems to be simplified without material loss of model integrity (Stone, 2013, pers comm). Therefore, one solution to reduce the computation time to find the optimum solution in scheduling is aggregation of mining blocks. Clustering algorithm technique has been used widely for the categorisation of large data set. There is a wide range of cluster algorithms that have been developed in psychology, biology, control and signal processing, information theory and data mining (Kogan, 2007). The clustering analysis seeks to group objects of a similar properties or value into different categories. It reveals conceptual, spatial and/or temporal and representational aggregation (Lee, Qu and Lee, 2012). Askari-Nasab et al (2010) have used the Fuzzy C-Means (FCM) clustering algorithm in creating mining cuts for production schedule. This algorithm is among the latest aggregation approaches available. The study focuses further on developing a deterministic MILP formulation, implementing and documenting the details of the MILP models in TOMLAB/CPLEX (by MATLAB® environment, verifying and validating the MILP production scheduler by a case study against GEOVIA Whittle™ (by GEOVIA, Dassault Systèmes). The MILP scheduling result which was obtained using the data from FCM algorithm produced a NPV of $50.4 M or three per cent higher value than Whittle Milawa Balance algorithm from GEOVIA Whittle™ (Askari-Nasab et al, 2010). Further research were conducted on a two-stage clustering algorithm for block aggregation in

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open pit mines. The two-stage clustering method is based on hierarchical clustering for block aggregation where the size of the maximum cluster size is controlled as an input. The second step involves improving the mining cuts that is created from the mining cuts with Tabu Search, which is a local search algorithm that balances between the quality of clusters and the number of binary constrains (Tabesh and Askari-Nasab, 2011).

The aim of this research is to evaluate the mine planning process in creating mining cuts for production scheduling and compare the NPV value of the schedules based on block aggregation technique versus that derived from classic pushback design.

**METHODOLOGY**

The approach in the analysis and the completion of the research required several steps to be taken. The major project tasks and activities include:

- familiarising with the most popular clustering algorithm for high dimensional data
- applying a suitable clustering algorithm for a block model data and revision of the block aggregation algorithm
- evaluating a case study by creating a long-term mine scheduling with clustering algorithm and compare it with classic pushback design production schedule.

Two main clustering algorithms, Hierarchical Cluster Algorithm and Fuzzy C-Means Cluster Algorithm were considered for the clustering of mining blocks. Brief descriptions and comparisons of these methods can be found in the following section.

**Hierarchical cluster algorithm on generic block model**

Hierarchical clustering is a method that groups data based on similarity by creating a cluster tree or dendrogram. The general procedure of these types of algorithms is reviewed as follows (Johnson, 1967):

The logic of the algorithm starts by considering each object as a cluster, in other words; if there are $N$ objects, then there are $N$ clusters. The similarities between clusters are the same as the similarities between the objects they contain. The major steps of the hierarchical cluster algorithm is summarised as follows:

- find the most similar pair of clusters and merge them into a single cluster
- compute similarities between the new cluster and the rest of the clusters
- repeat steps until all objects are merged together and a cluster with size of $N$ is created.

**Fuzzy C-Means cluster algorithm**

Dunn (1973) developed the FCM algorithm. Bezdek (1981) introduced an improvement on traditional clustering methods. The technique assumes each data point belongs to a cluster to some degree, which is specified by a membership grade. The iterations minimise the objective function in Equation 1 that represents the distance from any given data point to a cluster centre.

$$J_m(U, v_1, \ldots, v_K) = \sum_{i=1}^{n} \sum_{k=1}^{K} (u_{ik})^m d^2(x_i, v_k)$$   \hspace{1cm} (1)

where:

- $u_{ik}$ is the degree of membership of $x_i$
- $x_i$ is the $i$th object in the measured data
- $v_k$ is the centre of the cluster
- $d^2(x_i, v_k)$ is the square Euclidean Distance between $x_i$ and $v_k$.

The exponent $m$ is chosen from $(1, \infty)$ in advance; it is used to determine the degree of fuzziness of the clustering, $m$ between 1.1 and 5 is typically reported as the most useful range as recommended by Bezdek (1981).

Sato, Sato and Jain (1995) evaluated conventional clustering as classifying the given data into exclusive subsets. They described that the technique can clearly determine whether an object belongs to a cluster or not. However, this kind of partition is not sufficient to represent many real situations. The research team offered a fuzzy clustering method to construct clusters with uncertain boundaries. The FCM is one of the methods of fuzzy clustering. Figure 1 shows the scatter plot of the data. In this case, the FCM is trying to divide the data set into two sets based on their location. Figure 2 depicts the result of the FCM results. As can be seen from Figure 2, the data set marked x has centre X, whilst the data set o has centre O.
These two cluster methods are both available for clustering block model. Pros and cons of both methods are summarised in Table 1. Both have been used in mining application for clustering mining block models, for this research FCM is used as the method for block aggregation. This is because it has high flexibility to cluster the blocks and it can control cluster sizes. More importantly, the calculation steps to find the solution are shorter than the traditional hierarchical algorithm.

<table>
<thead>
<tr>
<th>Advantages</th>
<th>Hierarchical</th>
<th>Fuzzy C-Means</th>
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</thead>
<tbody>
<tr>
<td>Can handle categorical</td>
<td>Can handle categorical variables.</td>
<td>Fewer steps to find solution.</td>
</tr>
<tr>
<td>Variables.</td>
<td>Produces better clusters.</td>
<td>Can control cluster sizes.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Highly flexible.</td>
</tr>
<tr>
<td>Disadvantages</td>
<td>Has more steps to find similarity.</td>
<td>It is not designed to partition categorical</td>
</tr>
<tr>
<td></td>
<td>Slower when dealing with larger data sets.</td>
<td>variables.</td>
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</table>

**IMPLEMENTATION OF THE CLUSTERING ALGORITHM FOR BLOCK AGGREGATION**

**Case study of a copper block model**

The FCM algorithm is implemented on a copper block model to demonstrate the practicability and the high project value to produce mining cuts. The algorithm of the FCM is described in Equation 1. The criterion for aggregating the block is based on the Euclidean distance between each block. For this copper block model, the ultimate pit limit has been calculated which includes 8196 blocks with a total of 19 benches. Blocks in this pit shell are exported to a CSV (comma-separated values) file for cluster analysis.

There are several steps to cluster the entire block model. The data has to be processed bench-by-bench. For block aggregation with clustering method, every bench is partitioned into a number of mining cuts starting from the bottom bench. The bottom bench has 17 blocks, forming the first mining cut. The next bench has 58 blocks which is clustered into two mining cuts. Clustering GUI Tool in MATLAB®, version R20131 (by MathWorks) has been used to perform FCM algorithm. The GUI can perform tasks as follows:

- load and plot the data
- clustering the data with FCM algorithm
- apply different control parameters such as cluster number (varies each bench), maximum iterations (100), exponent = 2 and minimum improvement 1e-005 for clustering
- save the cluster centre.

After performing the FCM clustering with a fixed number of centres, each block in the original CSV has a centre which indicates which mining cut number it belongs to. The centres are imported back to the CSV file and eventually to the original block model which can be viewed in 3D with mining software. The same process is carried out bench-by-bench by assigning the mining cut number to the block in the model block. Figure 3 presents the Bench 2240 in the copper block model before the clustering algorithm is applied. The same bench after the clustering algorithm is demonstrated in Figure 4. Overall 81 clusters have been created as the total clusters with this process.

Scheduling is done using Chronos software (Maptek Pty Ltd, 2012) which utilises a spreadsheet to store, manipulate and report scheduling data. Conventionally, the reserve is broken down by bench and user defined variables and scheduling constraints are given to the optimisation engine CPLEX (ILOG, 2005).

The overall NPV value of the scheduling from 81 mining cuts is $215.74 M. Figure 5 demonstrates the yearly production schedule for the life of the mine using FCM.

The same block model was used to derive the optimal production schedule based on conventional approach in the second step. For the pushback design three pit shells are selected from the pit-by-pit graph produced by the Pit Optimizer Tool in GEOVIA Whittle™ using the Lerch-Grossman algorithm. The reserve blocks in the block model are aggregated into mining cuts with the following method:

- Every reserve block is categorised into either Pushback 1, Pushback 2 or Pushback 3 based on their pit shell that is produced from Lerch-Grossman algorithm and criteria for minimum mining width problem.
- For each pushback, blocks that are at the same elevation belong to the same mining cuts. This process has produced 48 mining cuts which are less than the mining cuts produced from FCM algorithm. Figure 6 presents the corresponding annual ore and waste schedules. The schedule of the traditional pushback design yielded a
The production plan which was produced using the mining cuts from the FCM algorithm has a $2.2 M higher NPV value than the classical pushback method.

The MILP solution time is the same for both methods which was under three minutes under the same input parameters. However, the traditional pushback design involves the several steps to find a series of pushbacks and to eliminate the minimum mining width problem. On the other hand, the cluster design involves manually exporting and importing the block models into MATLAB® for analysis. Both methods can be time consuming for average users who wish to create a production schedule from a raw block model. At this stage, it is acceptable to conclude the FCM is a valid method to create a practical production schedule. More clustered mining cuts can be created to test for the potential of yielding higher NPV.

CONCLUSIONS AND RECOMMENDATIONS
The objective of this research is to analyse and validate a clustering algorithm for block aggregation. Popular cluster algorithms are reviewed and compared for the goodness of clustering for mining purpose. The FCM cluster method is used to partition benches of a case study and it is tested against the goodness of clustering and practicality of life of mine production scheduling. The result shows the FCM can partition blocks on every bench into different clusters without losing significant details. By using this cluster algorithm, the case study block model which has 8158 blocks has been clustered into 81 mining cuts. On the other hand, the conventional pushback design has created 48 mining cuts from the same number of blocks. The mining cuts formed from clusters resulted in an increase in NPV of $2.2 M as compared to pushback design.

More clustered mining cuts can be created to test for the potential of yielding higher NPV. However, the time and effort to apply the FCM process on a block model is too high. If this process can be automated within the mine design software, it would be more efficient for users to find the production schedule from the mining cuts that are created automatically. The coding or the automatic process could be future research areas.

REFERENCES


Optimising Cut-off Grade Strategy for Seabed Massive Sulfide Operations

S A Skorut¹ and S Saydam²

Polymetallic seabed massive sulfide (SMS) deposits represent a significant resource base which can help bridge the supply of critical minerals to society. Analysis is needed to apply strategic mine planning decisions in an SMS mining context. The selection of cut-off grade is a key strategic decision and consequently, can greatly influence a project’s net present value (NPV).

This project demonstrates the application of an optimal dynamic cut-off grade strategy to a hypothetical SMS deposit. A generalised cut-off grade optimisation algorithm was derived and applied to a case study, which produced a 16.5 per cent improvement in the present value when compared to a break-even cut-off grade. The influence of in situ grade variance was found to be relatively small, with approximately a 1 per cent loss in present value expected on average when following an optimised plan.

Furthermore, methods and techniques pertaining to the application of the cut-off grade algorithm as a strategic tool are developed. Practical implications of optimal strategies in various scenarios are explored. These developments enable mine planners and decision makers to develop economically viable life of mine plans.

The algorithm presented can be used as a foundation for further developing strategic mine planning approaches for SMS operations.

INTRODUCTION

With the increasing use of metals in society, the average grade of mines is declining, despite improving mining, processing and refining technologies. The mining industry is seeking alternative resources to traditional land based deposits to fill the gap in supply. One of these alternatives is seabed massive sulfide (SMS) mineralisations, which are high-grade, polymetallic mineralisations containing copper, zinc, lead and tin as well as gold and silver. Analogous to the petroleum industry moving offshore to meet demand, the global occurrences of SMS mineralisations provide an opportunity for the mining industry to also move offshore.

Massive sulfide mineralisations are deposited from hydrothermal venting systems, which form at a diverse range of tectonic settings. The majority of hydrothermal systems occur along fast spreading mid ocean ridges, however the largest deposits occur at intermediate and slow spreading centres, at ridge axis volcanoes, in deep back arc basins and in sedimented rifts adjacent to continental margins (Hannington, De Ronde and Petersen, 2005).

The ore formation process is almost identical in each case, however the mineral composition varies due to the different composition of host volcanic rocks (Herzig and Hannington, 1995). More than 300 sites of hydrothermal activity exist, with over 100 sites forming massive sulfide deposits as indicated in Figure 1.

Herzig (1999) claims marine mining will be feasible under conditions of:

- high gold and base metal grades
- site location close to land and ideally within territorial waters of a coastal state
- relatively shallow water depth, not significantly exceeding 2000 m.

Hannington et al (2011) mentions that easily accessible SMS mineralisations have tonnages estimated in the order of 600 Mt, containing 30 Mt of copper and zinc. Consequently, the mining of SMS mineralisations can help secure the supply of critical minerals in the future as the mining industry continues to face declining average grades.

Mining of polymetallic deposits is soon to be a reality with two companies; Nautilus Minerals and Neptune Minerals; developing mining systems to extract SMS deposits at depths of 1000–2000 m below sea level, as shown in Figures 2 and 3, respectively.

Many potential environmental benefits of SMS over land based mining were cited by Fouquet and Scott (2009).

In depleting minerals from the earth’s crust it is paramount that these minerals are extracted in a manner that optimises the projects’ objectives, as they can only be mined once.

Maximising the economic value of a deposit, specifically, net present value (NPV) is the most common objective of mining operations. A key variable in maximising project NPV is cut-off grade. Cut-off grade is the criteria used to define ore and thus, differentiates between material scheduled for treatment and material considered to be waste (Lane, 1964). Cut-off grade is a dynamic, strategic mine planning decision which has profound effects on the size of reserves and hence, all operational decisions influenced by project size, such as

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throughput considerations. Therefore, there is a need to optimise cut-off grade decisions in order to maximise the economic value derived from mineral properties.

Optimising cut-off grades for terrestrial open pit and underground operations is now standard practice in industry. Most cut-off grade strategies are derived from the pioneering work by Lane (1964), which identified the need to maximise the NPV of the project when optimising cut-off grades. Whilst Lane’s Algorithm presents a generalised mine model based upon both open pit and underground operations, further research has indicated that cut-off grade selection for underground mining needs to be more carefully determined due to mining selectivity (Yi and Sturgul, 1987). Similar to the way in which underground operations optimise cut-off grade strategies differently from open pit operations; SMS deposits require analysis and research in order to augment, modify and apply traditional cut-off grade theories.

**METHODOLOGY**

**Seabed massive sulfide strategic mine design**

Strategic mine design begins with a geological block model of the mineralisation, which is a representation of a deposit as an array of blocks. Each block has properties assigned, such as metal grades, density and geological domains. In its entirety the block model provides an indication of a deposit size, shape, grade distribution and other characteristics. Knowledge of these parameters enables engineers to evaluate the likely size and costs involved in mining the deposit. From this start point SMS mine planning, predominantly follows the traditional open pit planning process as illustrated in Figure 4.

Initial cost estimates combined with geotechnical data are used to design an ultimate pit. The selection of an ultimate pit is largely the same as with traditional open pit mining, using...
Lerchs-Grossman pit optimisation algorithm to determine a viable final pit.

In a standard open pit scenario pushbacks are defined. Pushbacks outline nested pits which are sequentially excavated in order to maximise cash flows by delaying waste movement. This process also applies to SMS mine design, however, SMS operations will generally mine bench by bench and thus, pushback design is not entirely necessary. In a traditional open pit scenario mining bench by bench generally produces the ‘worst case’ for cash flows. However, in an SMS context, deposits are flat lying with little overburden removal required. Therefore, comparisons between mining bench by bench versus utilising pushbacks may not lead to large differences.

At this point, the bench excavation cuts are designed with respect to the cut-off grade strategy selected. A detailed production schedule is then developed from the selected cut-off grade strategy. The production schedule sets forth the timing of bench excavations creating a schedule specifying the waste and ore tonnes mined as well as the ore grade per a period. From this point the process is completed again from start as production and economic inputs initially estimated have been refined and further developed.

**Generalised seabed massive sulfide operational model**

In order to frame the optimisation sequence, a generalised SMS operation model needed to be created in order to validly model the mining sequence. Lane’s (1988) three stage mine model is used to correctly classify each activity into the correct throughput category. Activities are separated into mining, treating and marketing activities as shown in Figure 5.

Utilising the throughput classifications, a generalised profit function can then be defined as in Equation 1 (Lane, 1988).

\[ P_t = (p - k)Q_t - hQ_t - mQ_t - f_t \]  

where:

- \( \tau \) = length of time period (quarters)
- \( P_t \) = profit per a period ($US millions)
- \( p \) = selling price of final product ($US/t)
- \( k \) = selling cost of final product ($US/t)
- \( h \) = treatment cost per a tonne of ore treated ($US/t)
- \( m \) = mining cost per a tonne of mineralised material mined ($US/t)
- \( f \) = fixed cost per a time period ($US)

**FIG 3** – Neptune Minerals proposed subsea mining system (Parenteau et al, 2013).

**FIG 4** – Traditional open pit circular planning process (Dagdalen, 2001).

**FIG 5** – Classification of SMS operational activities.


\( Q_m \) = quantity of material mined (Mt)

\( Q_h \) = quantity of ore treated (Mt)

\( Q_f \) = quantity of final product sold (kt)

**Objective function**

The objective of cut-off grade optimisation is to maximise the NPV of a project, as in Equation 2.

\[
NPV = \sum_{t=0}^{T} \frac{P_t}{(1 + d)^t}
\]

where:

\( t \) = life of mine (quarters)

\( d \) = discount rate (equivalent annual discount rate)

For a cut-off grade strategy to be optimal, it must be optimal in every time period (Lane, 1988). Thus the objective function is reduced to the maximisation of present value per a period. The difference in value between two time periods is given by Equation 3.

\[ v = P_t - F \tau \]

where:

\( v \) = difference in value between time periods ($US)

\( F \) = opportunity cost of remaining future value ($US)

Thus, the final objective function is defined as in Equation 4, which states that per a unit of resource depletion, the value derived from that unit is maximised.

\[ \text{Max}(v) = \text{Max}[(p - k)Q_h - hQ_m - mQ_f - (f + F)\tau] \]

**Cut-off grade solutions**

When maximising the objective function, three closed form solutions occur. These are the three limiting economic cut-off grades which occur when either the mine, treatment or market limits throughput. Three more cases arise when more than one capacity limits throughput, these are balancing cut-off grades. The following discussion is drawn from Lane (1988).

**Economic limiting cut-off grades**

The cut-off grade resulting from the mine limiting case is shown in Equation 5.

\[ g_m = \frac{h}{(p - k)y} \]

where:

\( g_m \) = mine limiting cut-off grade (%)

\( y \) = yield in the ore treatment process (%)

The mine limiting cut-off grade is solely dependent on economics, thus no reference to present values exist. When the treatment limits throughput Equation 6 shows the corresponding cut-off grade.

\[ g_h = \frac{h + (f + F)}{(p - k)y} \]

where:

\( g_h \) = treatment limiting cut-off grade (%)

\( H \) = treatment capacity (Mt)

The treatment cut-off grade is dependent on opportunity cost and thus future present values. The market limiting cut-off grade shown in Equation 7, is also dependent on opportunity cost.

\[ g_r = \frac{h}{p - k - (f + F)} \]

where:

\( g_r \) = market limiting cut-off grade (%)

\( R \) = market capacity (kt)

**Balancing cut-off grades**

In mining operations where more than one component is limiting throughput the utilisation of capacities is determined by the grade distribution, as well as cut-off grade. By computing the ratios between capacities and comparing this value to the cumulative grade distribution of the deposit, a balancing cut-off grade can also be determined. Thus three more cut-off grades can be calculated corresponding to the balancing of each pair of throughputs as shown in Table 1.

For balancing cut-off grades to be valid the grade distribution of mineralised material must be known with reasonable certainty. Preferably, the grade distribution for each pushback or phase has been developed. Therefore, balancing grades are applicable in both strategic and tactical decision (Lane, 1988).

**Effective optimum cut-off grades**

Lane (1988) has shown that the optimum cut-off grade is one of the six cut-off grades. By graphically plotting the value derived from each cut-off grade an optimal grade can be found algebraically as shown in Figure 6. An effective optimum cut-off grade can be found for each pair of cut-off grades. The optimum cut-off grade is then the median of these three effective cut-off grades.

**Optimisation algorithm**

With the optimum cut-off grade now defined, the dynamic method of determining a cut-off grade strategy which maximises the objective function can be developed. The algorithm for determining optimum cut-off grades in each period is shown in Figure 7. The algorithm was validated against that developed by Dagdalen (1993), producing a result within 0.1 per cent.

**CASE STUDY**

<table>
<thead>
<tr>
<th>Cut-off grade</th>
<th>Limiting pair</th>
<th>Ratio</th>
<th>Graph</th>
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<tbody>
<tr>
<td>( g_{mb} )</td>
<td>Mining and treatment</td>
<td>H/M</td>
<td>Cumulative grade distribution ((x))</td>
</tr>
<tr>
<td>( g_{m} )</td>
<td>Mining and market</td>
<td>R/M</td>
<td>Recoverable mineral per a unit of mineralised material ((xg))</td>
</tr>
<tr>
<td>( g_{r} )</td>
<td>Market and treatment</td>
<td>R/H</td>
<td>Recoverable Mineral per a unit of ore ((kg))</td>
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</tbody>
</table>

The optimisation algorithm was applied to a theoretical case study. The case study was constructed from publicly available data pertaining to Nautilus Minerals Solwara 1 deposit; in order to closely resemble reality. The block model contains grade estimations for a total of 22,471 blocks containing copper, gold, silver and zinc. Blocks have dimensions of...
10.0 m in the Easting and Northing directions and 0.5 m in the vertical dimension. The vertical extent was selected to match the corresponding minimum mining unit and to match Nautilus Minerals Solwara 1 block model dimensions. At an average dry bulk density of 3.0 t/m³ the total mineralised region contains 3.4 Mt.

The deposit will be mined using Nautilus Minerals’ proposed seafloor mining system as described by Jankowski et al (2010).

The SMS mineralisation has a total resource tonnage of 3.4 Mt, containing copper and gold. It is planned to be mined at 1.2 Mt/a, with a 0.8 Mt/a treatment rate, resulting in an estimated life of mine (LOM) of 2–5 years.

**Grade distribution data**

A block model was created from publicly available drill hole data on Nautilus Minerals’ Solwara 1 deposit (Lipton et al, 2012). The block model is shown in Figure 8. Gold grades were converted to an equivalent copper grade for grade estimation using the method set out by Ataei and Osanloo (2003a).

The resultant grade-tonnage distribution data is presented in Table 2.

**Input data**

Operational and economic input data are shown in Table 3. Values used in the optimisation were either obtained or back
calculated from Nautilus Minerals’ Cost Study (Jankowski et al., 2010).

Importantly, some parameters stray significantly to those usually encountered in the traditional mining systems. The discount rate is relatively high at 25 per cent, this value is valid given the unproven nature of SMS operations. Treatment recovery is 100 per cent as the equivalence grades already include recoveries.

Treatment costs are relatively high to reflect the categorisation of specific activities into the treatment category. Finally, the cost associated with running a production support vessel regardless if mining is occurring or not, supports the high fixed cost used. Note that the optimisation has been completed in quarters to highlight less obvious findings, due to the short LOM. An equivalent quarterly discount rate was applied to ensure validity.

**CASE STUDY FINDINGS**
The results of the algorithm were compared to the use of break-even cut-off grades and Heuristic methods of cut-off grade determination. The present values are used to indicate value, as all methods have the same capital expenditure.

**Optimisation algorithm results**
Output from the optimisation algorithm, predicated from the case study, is shown in Table 4.

The strategy yields a present value of $462.9 M over a LOM of 12 quarters.

The outlined strategy results in a balanced cut-off grade of 2.76 per cent from the 1st to the 8th quarter, resulting in the mine and treatment capacities both limiting overall system throughput. From the 9th to the 12th quarter the treatment capacity is the limiting throughput and thus a declining cut-off grade ensues during this period.

**Break-even cut-off grade comparison**
The algorithm’s output was compared with results obtained from using break-even cut-off grade formula. The break-even strategy achieves the schedule shown in Table 5, resulting in a present value of $397.3 M over a LOM of 16 quarters. The cut-off grade remains fixed throughout the LOM at 0.83 per cent resulting in 11.6 kt of equivalent copper metal being refined each period.

---

**TABLE 2**
Copper equivalent grade-tonnage distribution data.

<table>
<thead>
<tr>
<th>Increment (%</th>
<th>Increment grade (%)</th>
<th>Increment tonnage (t)</th>
<th>Cumulative tonnage (t)</th>
<th>Average grade above cut-off (CuEq%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0–1</td>
<td>0.24</td>
<td>241 500</td>
<td>3 370 650</td>
<td>5.34</td>
</tr>
<tr>
<td>1–2</td>
<td>1.46</td>
<td>569 250</td>
<td>3 129 150</td>
<td>5.74</td>
</tr>
<tr>
<td>2–3</td>
<td>2.45</td>
<td>412 500</td>
<td>2 559 900</td>
<td>6.69</td>
</tr>
<tr>
<td>3–4</td>
<td>3.43</td>
<td>307 200</td>
<td>2 147 400</td>
<td>7.50</td>
</tr>
<tr>
<td>4–5</td>
<td>4.48</td>
<td>216 000</td>
<td>1 840 200</td>
<td>8.18</td>
</tr>
<tr>
<td>5–6</td>
<td>5.48</td>
<td>286 200</td>
<td>1 624 200</td>
<td>8.67</td>
</tr>
<tr>
<td>6–7</td>
<td>6.52</td>
<td>229 950</td>
<td>1 338 000</td>
<td>9.35</td>
</tr>
<tr>
<td>7–8</td>
<td>7.50</td>
<td>252 900</td>
<td>1 108 050</td>
<td>9.94</td>
</tr>
<tr>
<td>8–9</td>
<td>8.49</td>
<td>206 100</td>
<td>855 150</td>
<td>10.67</td>
</tr>
<tr>
<td>9–10</td>
<td>9.40</td>
<td>223 050</td>
<td>649 050</td>
<td>11.36</td>
</tr>
<tr>
<td>10–15</td>
<td>11.78</td>
<td>383 250</td>
<td>426 000</td>
<td>12.38</td>
</tr>
<tr>
<td>15–20</td>
<td>16.55</td>
<td>34 350</td>
<td>42 750</td>
<td>17.81</td>
</tr>
<tr>
<td>20+</td>
<td>22.93</td>
<td>8400</td>
<td>8400</td>
<td>22.93</td>
</tr>
</tbody>
</table>

**TABLE 3**
Case study input data.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper price</td>
<td>$/t</td>
<td>$7000</td>
</tr>
<tr>
<td>Selling cost</td>
<td>$/t</td>
<td>$1000</td>
</tr>
<tr>
<td>Discount rate</td>
<td>%/a</td>
<td>25%</td>
</tr>
<tr>
<td>Treatment recovery</td>
<td>%</td>
<td>100%</td>
</tr>
<tr>
<td>Mining cost</td>
<td>$/t material mined</td>
<td>$20</td>
</tr>
<tr>
<td>Treatment cost</td>
<td>$/t ore treated</td>
<td>$30</td>
</tr>
<tr>
<td>Mining capacity</td>
<td>Mt/a</td>
<td>1.2</td>
</tr>
<tr>
<td>Treatment capacity</td>
<td>Mt/a</td>
<td>0.8</td>
</tr>
<tr>
<td>Market capacity</td>
<td>Mt/a</td>
<td>Not limited</td>
</tr>
<tr>
<td>Capital cost</td>
<td>$M</td>
<td>$400</td>
</tr>
<tr>
<td>Fixed cost</td>
<td>$M/a</td>
<td>$50</td>
</tr>
<tr>
<td>Estimation tolerance</td>
<td>$M</td>
<td>$0.10</td>
</tr>
</tbody>
</table>
The optimisation algorithm produced a present value of $462.9 M, a 16.5 per cent improvement over the break-even strategy. The improvement in present value is a result of the opportunity cost of the break-even strategy; lower grade material is being mined when higher grade material is available. The algorithm explicitly values opportunity cost and thus implements a higher cut-off available. The algorithm explicitly values opportunity cost of the break-even strategy. The improvement in present value by 16.5 per cent increase in present value over the use of Heuristic methods.

When compared with break-even and Heuristic methods of cut-off strategies the algorithm achieves a greater present value by 16.5 per cent and 5.5 per cent, respectively. In both cases the algorithm created strategies which:

- produced greater present values
- mined a higher average grade over LOM
- classified and hence treated less total ore
- refined and sold less final metal
- mined for less time (lower LOM).

### TABLE 4
Optimisation algorithm cut-off grade schedule.

<table>
<thead>
<tr>
<th>Qtr</th>
<th>Cut-off grade (%)</th>
<th>Average grade (%)</th>
<th>Qh (Mt)</th>
<th>Qr (Mt)</th>
<th>Qh (Kt)</th>
<th>Profit ($M)</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>53.9</td>
</tr>
<tr>
<td>2</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>51.0</td>
</tr>
<tr>
<td>3</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>48.3</td>
</tr>
<tr>
<td>4</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>45.6</td>
</tr>
<tr>
<td>5</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>43.2</td>
</tr>
<tr>
<td>6</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>40.8</td>
</tr>
<tr>
<td>7</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>38.6</td>
</tr>
<tr>
<td>8</td>
<td>2.76</td>
<td>7.48</td>
<td>0.30</td>
<td>0.20</td>
<td>15.0</td>
<td>57.0</td>
<td>36.5</td>
</tr>
<tr>
<td>9</td>
<td>2.73</td>
<td>7.46</td>
<td>0.30</td>
<td>0.20</td>
<td>14.9</td>
<td>56.8</td>
<td>34.4</td>
</tr>
<tr>
<td>10</td>
<td>2.57</td>
<td>7.32</td>
<td>0.29</td>
<td>0.20</td>
<td>14.6</td>
<td>55.3</td>
<td>31.6</td>
</tr>
<tr>
<td>11</td>
<td>2.41</td>
<td>7.19</td>
<td>0.28</td>
<td>0.20</td>
<td>14.4</td>
<td>53.9</td>
<td>29.2</td>
</tr>
<tr>
<td>12</td>
<td>2.24</td>
<td>7.07</td>
<td>0.19</td>
<td>0.07</td>
<td>5.4</td>
<td>19.2</td>
<td>9.8</td>
</tr>
<tr>
<td>Totals</td>
<td>7.43%</td>
<td>3.37</td>
<td>2.27</td>
<td>0.37</td>
<td>168.8</td>
<td>$61.5</td>
<td>$462.9</td>
</tr>
</tbody>
</table>

### TABLE 5
Cut-off grade schedule using break-even formula.

<table>
<thead>
<tr>
<th>Qtr</th>
<th>Cut-off grade (%)</th>
<th>Average grade (%)</th>
<th>Qh (Mt)</th>
<th>Qr (Mt)</th>
<th>Qh (Kt)</th>
<th>Profit ($M)</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>36.7</td>
</tr>
<tr>
<td>2</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>34.8</td>
</tr>
<tr>
<td>3</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>32.9</td>
</tr>
<tr>
<td>4</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>31.1</td>
</tr>
<tr>
<td>5</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>29.4</td>
</tr>
<tr>
<td>6</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>27.8</td>
</tr>
<tr>
<td>7</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>26.3</td>
</tr>
<tr>
<td>8</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>24.9</td>
</tr>
<tr>
<td>9</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>23.5</td>
</tr>
<tr>
<td>10</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>22.2</td>
</tr>
<tr>
<td>11</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>21.0</td>
</tr>
<tr>
<td>12</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>19.9</td>
</tr>
<tr>
<td>13</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>18.8</td>
</tr>
<tr>
<td>14</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>17.8</td>
</tr>
<tr>
<td>15</td>
<td>0.83</td>
<td>5.82</td>
<td>0.21</td>
<td>0.20</td>
<td>11.6</td>
<td>38.9</td>
<td>16.8</td>
</tr>
<tr>
<td>16</td>
<td>0.83</td>
<td>5.82</td>
<td>0.18</td>
<td>0.17</td>
<td>9.9</td>
<td>19.2</td>
<td>9.8</td>
</tr>
<tr>
<td>Totals</td>
<td>5.82%</td>
<td>3.37</td>
<td>2.17</td>
<td>0.21</td>
<td>132.4</td>
<td>$61.5</td>
<td>$462.9</td>
</tr>
</tbody>
</table>

### FIG 9
Break-even and optimisation algorithm strategies.

**Heuristic cut-off grade method comparison**

Heuristic methods involve using break-even formulae but may add additional provisions over periods of time. The Heuristics method resulted in a present value of $438.9 M over a LOM of 15 quarters as shown in Table 6.

The cut-off grade declines in steps over time in correspondence to the assumptions used as shown in Figure 9. The total present value of the optimisation algorithm achieves a 5.5 per cent increase in present value over the use of Heuristic methods.

When compared with break-even and Heuristic methods of cut-off strategies the algorithm achieves a greater present value by 16.5 per cent and 5.5 per cent, respectively. In both cases the algorithm created strategies which:

- produced greater present values
- mined a higher average grade over LOM
- classified and hence treated less total ore
- refined and sold less final metal
- mined for less time (lower LOM).
A summary of results for the different cut-off grade strategies is show in Table 7.

**DISCUSSION**

**Optimal capacities**

Based on the current SMS mining systems proposed, the limiting throughput is likely to be either mine, treatment or both mine and treatment limiting in balance. This is a consequence of the unique logistics of SMS operations and also SMS mineralisations characteristically low strip ratios.

The determination of an optimal capacity can be solved quantitatively by considering cut-off grades as a production driver. The cut-off grade optimisation algorithm can then be used as a strategic mine planning tool. By providing a capital cost matrix and resultant variable costs, various mine plans can be evaluated within the algorithm. Each scenario was evaluated using the algorithm, with the resultant NPV contours shown in Figure 11.

There is a clear plateau illustrated by Figure 11 of economically feasible mine plans, with this knowledge mine planners can focus on systems corresponding to those capacities.

The algorithm can also be used to identify risk–reward situations via sensitivity analysis in designing mine plans. Mine plans can be evaluated on their ability to handle adverse changes in commodity prices. A robust mine plan can then be triangulated which achieves the highest possible expected outcome given a set of economic conditions. Using the augmented case study data, a down case as represented by a -10 per cent change in copper price and an up case as represented by a +10 per cent change in copper price were evaluated against the various mine plan combinations as shown in Figures 12 and 13 respectively.

The identification of relative mine plan combinations which outperform can empower mine planners to make robust strategic decisions. The output for each scenario can be compared to identifying changing characteristics in the mining system and also what course of action to take.

<table>
<thead>
<tr>
<th>TABLE 7</th>
<th>Summary of cut-off grade strategies.</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>LOM (quarters)</td>
</tr>
<tr>
<td>Break-even formula</td>
<td>16</td>
</tr>
<tr>
<td>Heuristic algorithm</td>
<td>15</td>
</tr>
<tr>
<td>Optimised algorithm</td>
<td>12</td>
</tr>
</tbody>
</table>
provides a relatively rapid and valid basis for making mine planning decisions in a changing economic environment.

Grade control
Grade control is a necessary activity for any mining operation and often adds significant cost. In an SMS setting grade control issues become more contentious due to poor core recoveries and high drilling costs. Consequently SMS operations are likely to have highly variable head grades when compared to expected or predicted grades.

An optimal strategy is based upon the assumption that the grade-tonnage curve is correct and grade descriptors will be correctly and validly applied in practice. Consequently strategic decisions will be presumed optimal from the data, but non-optimal when compared to what is actually in the ground.

Monte Carlo analysis was conducted to determine the potential loss in value of following a mine plan which has been based off a grade-tonnage distribution which is truly variable in nature. One thousand iterations were completed, with metal grades the only stochastic variable. The variability was achieved assigning a normal distribution for the interval tonnages, with an expected mean equal to the base case tonnages and a standard deviation of 20 per cent of the mean applied. Table 8 shows resultant descriptive statistics.

The variance is surprisingly small, with a mean present value loss of $4.1 M and a maximum loss of $16.6 M. In terms of the initial present values a percentage spread is more meaningful. Descriptive statistics are shown in Table 9, the mean confers with a 0.9 per cent loss and a maximum loss of 4.3 per cent of present value. Subsequently on average, following a perceived optimal plan will produce a relatively small loss of approximately 1 per cent of an operations present value. This result in itself makes the use of cut-off grade optimisation worthwhile, as it produces quite robust mine plans.

The actual variance in the optimal strategy versus the perceived optimal strategy was also compared. Figure 15 illustrates the range of optimal strategies produced from the stochastic grade-tonnage distribution. The dashed lines represent the distribution of optimal cut-off grades with one standard deviation (68 per cent) of values falling between the dashed lines.

The optimal cut-off grades differ by up to 1 per cent equivalent grade from the perceived optimal strategy. The spread is clearly larger in early periods and converges at the end of LOM; a reflection that over time, the ability to influence a projects present value is diminished.

It is tempting to conclude that extensive grade control is not required from the perspective of strategic decision making. However, grade control is entirely necessary operationally in order to identify on a tactical level what is being mined; a particularly difficult task in a deep sea mining environment.

Practical considerations and limitations

Grade distribution
Cut-off grade optimisation assumes that any ore block has the same grade distribution specified by the grade-tonnage curve. Consequently the schedules produced in the analysis are inadequate for detailed mine planning. Additionally it is assumed that there is sufficient mineralised material uncovered for mining.

To overcome the limitation the orebody can be divided in panels, each with its own grade-tonnage distribution as described by Dowd (1976). A sequential mining sequence can then be generated.

<table>
<thead>
<tr>
<th>TABLE 8</th>
<th>Descriptive statistics of Monte Carlo analysis.</th>
<th>Present value ($M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>461.3</td>
<td></td>
</tr>
<tr>
<td>Minimum</td>
<td>336.4</td>
<td></td>
</tr>
<tr>
<td>Maximum</td>
<td>551.4</td>
<td></td>
</tr>
<tr>
<td>Range</td>
<td>215.0</td>
<td></td>
</tr>
<tr>
<td>Standard deviation</td>
<td>34.7</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>TABLE 9</th>
<th>Potential value loss descriptive statistics.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Present value loss differential (% of optimal)</td>
<td></td>
</tr>
<tr>
<td>Mean</td>
<td>0.91</td>
</tr>
<tr>
<td>Minimum</td>
<td>0</td>
</tr>
<tr>
<td>Maximum</td>
<td>4.35</td>
</tr>
<tr>
<td>Range</td>
<td>4.35</td>
</tr>
<tr>
<td>Standard deviation</td>
<td>0.78</td>
</tr>
</tbody>
</table>
Stockpiling intermediate grades
Optimised strategies lead to overall lower mineral recovery. As a consequence of implementing a higher cut-off grade in early periods, material that would have been defined as waste may be economical later in the operations LOM when the cut-off grade is lowered. Intermediate material is usually stockpiled and fed to the mill during periods of low utilisation, for blending purposes and after mining has ceased. Seafloor stockpiles of fragmented intermediate material are improbable as they will require upkeep over the LOM. It is unlikely such methods will be economically feasible and consequently intermediate ore will most likely become sterilised.

Equivalent grades
The optimisation uses equivalence grades in order to convert minor economic mineral grades to a single, primary mineral. This method of handling polymetallic deposits becomes invalidating if: no definitive correlation exists between the mineral occurrences, one or more metals is limited at the market level, or input parameters significantly change in value from that used to calculate the equivalence grades.

The limitations regarding equivalence grades can be resolved by utilising ‘Grid Search’ or ‘Golden Section’ methodologies as describe by Lane (1988) and Ataei and Osanloo (2003b) respectively. The methods extend the operational model and hence the objective function to include two metals, instead of converting one metal to another.

Fixed treatment yields and prices
In practice treatment yields will vary with the average head grade entering the mill. Furthermore mill facilities are generally designed and optimised for a specific range of head grades. As a result, it is invalidating to assume a low-grade block will achieve the same recovery yield as a high-grade block.

The implementation of parametric equations into the grade estimation process can assist in relieving the assumption of fixed treatment yields. By doing so yields will be able to vary with the estimated block grade, thus providing further validity to the model and hence the optimisation.

CONCLUSIONS
This paper has presented models, algorithms and considerations to apply existing cut-off grade strategies to an SMS operation. The application of cut-off grades to SMS operations was achieved by augmenting existing cut-off grade theories.

When applied to the theoretical case study the strategy produced by the optimisation algorithm provided significant increases (165 per cent) in project present value over break-even cut-off grade strategies; and a modest increases (5 per cent) in present value over the use of Heuristic methods. Techniques for using the cut-off grade optimisation algorithm to evaluate mine plans were also presented.

Importantly, in a SMS context, grade control is a contentious issue. It was shown that, by following a perceived optimal plan, a relatively small loss of approximately 1 per cent of an operations present value on average can be expected. Additionally practical implementation issues and limitations such as grade distribution, equivalence grades, stockpiling and fixed yields were discussed.

By utilising optimisation algorithms, mine planners can readily identify operational characteristics, which can be exploited to ultimately add value to an operation. As a key component in strategic mine planning, cut-off grade optimisation can have a profound influence on the economic value derived from mineral properties.

RECOMMENDATIONS
This research and subsequent findings have facilitated the formation of several recommendations aimed at improving strategic mine planning in SMS operations. Importantly the algorithm and conclusions presented can be used as a foundation for further development and implementation of advanced techniques in an SMS mining setting. The findings include the following recommendations:

- implement grid search or golden section methodologies to improve optimisation validity
- incorporate a stochastic framework for grade estimation in order to minimise the influence of modelled geological variability on mine planning decisions.

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A Critique of Selective Mining Unit Sizing at Century Mine to Optimise Productivity with Dilution

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ABSTRACT
The mining industry of the modern era is at a fever pitch with innovation driven by the motivation to keep sales margins high amidst an ever-volatile commodities market. Autonomous mining, hybrid ore mining methods and increasingly larger and more mobile machinery continue to have a greater impetus than in the past. Amidst this flourishing innovation, the underlying principle of mining remains true; an operation is running at its most optimal point when it can attain the highest net present value possible from the in situ ore within its lease boundary. There are numerous drivers that stipulate the discounted cash flow of an operation. Getting swept up in, and fixating on, just a select few of these drivers, is a critical flaw in some aspects of modern mine planning because other important value drivers can be undermined and significantly compromised.

This paper acknowledges and focuses on the selective mining unit (SMU) sizing selection at Century Mine in far north-west Queensland to demonstrate that an intricate optimisation can be achieved between all key value adding drivers in a mining project. Presently, the mine as an operation is faced with numerous challenges; not only must the site remain profitable throughout a trough in the industry, it must also ensure that logistical challenges such as ore stockpiling are acted on before the situation turns dire. Appropriate SMU sizing is crucial as it stipulates the selectivity of ore excavation, and in turn impacts the extraction rate of this ore. Downstream effects of this process ultimately tie in the whole operation with the SMU size selected. This optimisation study aims to elucidate an SMU size that simultaneously maximises the discounted cash value and improves the ore stockpiling of the open pit base metals operation. It will be shown that net present value improvements up to 1 per cent over a 30 month period are achievable, and run-of-mine stockpiles can be improved by as much as 1400 t/day. Furthermore the project highlights that there is a potential to improve the discounted cashflow of other surface hardrock mines, by fixating on the selection of an appropriate SMU size for the operation in subject.

INTRODUCTION
At the cornerstone of hardrock mine planning is an ore block model; it dictates both the tonnage and grade of mineralisation, and it is adapted into computer aided drawing packages in order to develop and design the progression of all forms of mines. Within this block model are individual blocks, all with a designated width, length and height. Due to the multidirectional facet of mining, the length and the width of these blocks are generally equal to each other, and their height is based on numerous factors such as the equipment on-site and their vertical mining capabilities. The dimensions of these blocks stipulate how selectively material must be excavated, and therefore each individual block is coined as a selective mining unit, or SMU.

The task was set forth to assess the impact that an SMU has on six key value drivers within any mining operation. These drivers are as follows:

- mining:
  1. production rate
  2. extracted grade
  3. production cost.
- milling:
  4. throughput rate
  5. milled recovery
  6. milling cost.

Century Mine was used as a case study for the foundations of this analysis. Century Mine is an asset owned and operated by MMG Limited in north-west Queensland, and produces upwards of 470 kt/a of zinc in concentrate form. Commencing operations in 1998, the ore reserve is projected to be fully exhausted in 2016, leaving the mine with a remaining life of approximately 30 months. Even in the later stages of the mining phase there is an emphasis on optimising the operation. In assessing the various impacts that the SMU size

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has on a mining operation, the analysis was scoped in a way to delineate an SMU size that would:

- maximise the net present value (NPV) of the in situ orebody
- improve ore stockpiling logistics
- be more practical for operators to abide by.

This paper aims to elucidate the findings of the project by investigating the current state of knowledge, sharing the project methodology undertaken and discussing the findings and results of the project overall.

CURRENT STATE OF KNOWLEDGE

Selective mining

Noble (2011) defines selective mining as a process that specifically focuses on the accurate and precise extraction of economically optimal material in an effort to make the greatest return on a finite deposit. It is synonymous for its difference to bulk mining in the following ways (Swanepoel, 2003):

- mine plans are generally more intricate, and blast patterns are more precise
- exploration core is continually assayed for geological data, rather than just geotechnical data
- excavating equipment is generally smaller and more flexible
- excavation is focused more on precision than total movement
- haul trucks may be smaller to optimise cycle times with excavators, however this is not always the case
- mining unit costs are higher
- mining production rates are lower.

Kruger and van der Westhuizen (2005) conducted a study on the Thabazimbi iron ore mine in South Africa and found that significant economic improvements could be made on the operation by:

- adjusting the mining direction to follow the dip of the orebody and separate waste material more freely
- downsizing the equipment on-site to mitigate dilution and adhere to proper grade control measures
- reducing the bench height from 15–10 m to allow for an effective delineation between ore and waste.

Importantly the general consensus within the industry is that selective mining should ideally optimise the point between productivity and precision with regards to ore extraction.

Ore dilution

Bertinshaw and Lipton (2007) divides the roof causes of dilution into the following categories:

- Dilution due to geometry whereby the operation of excavation equipment cannot be physically matched with the shape and size of the orebody. Such an issue is commonly related to vein style ore deposits being extracted by surface methods.
- Dilution due to uncertain knowledge of the orebody whereby sampling and assaying precision is either limited or inadequate.
- Dilution due to blast movement whereby ore is thrown or heaved during the blasting process.
- Dilution due to mining errors and poor grade control where ore is incorrectly marked out on benches.

Importantly, dilution is measured as the percentage change between the in situ grade and the extracted ore grade, rather than the magnitude of the change between these two values.

To assess the influence on the extracted ore grade that various block sizes has, a block model assembled with the geostatistical method of kriging is encouraged. From there, the blocks can be manipulated into various shapes and sizes by using a volume weighted average technique. The analysis conducted in this report utilised this method.

Selective mining unit sizing

Leuangthong, Nenfeld and Deutsch (2002) describes the conventional definition of the SMU as the smallest volume of material on which ore waste classification is determined. The author however stresses that such a convention is impractical, as the sheer size of mining equipment generally negates the practicality of freely separating ore from waste material. The paper then introduces the concept of optimising the SMU size whereby, instead of focusing on a perfect extraction of the ore as per its original definition, a point is determined that economically optimises the extraction of ore, even if such a practice is dilutive.

Leuangthong, Nenfeld and Deutsch’s (2002) refined definition of an SMU is therefore the block model size that would correctly predict the tonnes of ore, tonnes of waste, and diluted head grade that the mill will receive with anticipated grade control practice. It is important to note that while the author of the paper disagrees with the conventional definition of an SMU, he does agree that the selective capacity to mine ore deprecates with an increasing SMU size, a theory widely proven in multiple case studies (Szmiegiel, 2005). Ultimately the optimal selection of an SMU is an iterative process that encompasses and relies upon numerous economic factors (Leuangthong, Nenfeld and Deutsch, 2002; Hekmat, Osanloo and Asi, 2008), including:

- equipment size
- mining method
- mining costs
- ore dilution
- ore and waste tonnage variances
- mill recovery.

Net present value sensitivity to value drivers

Accounting for the time value of money and discounting cash flow accordingly is crucial in order to conduct a fair economic assessment. The discount relative to time that is influenced by numerous factors in Australia:

- consumer price inflation
- employee industrial action, especially in coal mining operations
- uncertainty surrounding future Australian taxation policies
- non-profitable commodity prices being realised
- foreign exchange rate fluctuations
- general project execution risks (Harrison, 2010).

Although the theory behind a discount rate is relatively straightforward, there is still great conjecture over what discount rate to use in certain scenarios. For example the present day average discount rate for Australian projects ranges between 8 per cent and 12 per cent, however there is no definitive, publicly accepted figure that the industry turns to when in doubt of what rate to specifically use. It may pay to be conservative and use a higher discount rate, but therein lies its own issue (Quiggin, 1997) in that potentially profitable projects may be neglected due to having a negative NPV at one discount rate while still remaining positively valued at a slightly lower rate.
The following key strengths exist in the current state of knowledge. There is:

- ample evidence to suggest that an increased SMU size increasingly dilutes ore grade
- significant proof that selective mining has the ability to generate a more efficient ore mining operation over bulk mining
- an industry wide recognition of the risks of increasing milling throughput
- a general acceptance that Australian mining cashflow is discounted at 8 per cent to 12 per cent/annum in the current economic climate and at this point in time.

Having said this, there are also key weaknesses that exist in the current state of knowledge. There is:

- very limited research conducted on the manner which various specific SMU sizes simultaneously react with mined grade and mining productivity
- no current analysis conducted that simultaneously optimises SMU sizing with both mining and milling to maximise the value of in situ ore.

**METHODOLOGY AND ANALYSIS**

A combination of on-site and off-site work at Century and in Brisbane university offices respectively was required in order to undertake the analysis required. Contingency plans were put in place from the outset in order to ensure that the project’s progression was not compromised. At Century, an operational survey was conducted to assess the operational practicality of various SMU sizes relative to the excavation machinery on-site. A formal data collection process also allowed for the author to obtain crucial data related to historical production, future budgeted production and operational costs.

In Brisbane, significant progress was made on amalgamating, sorting and validating the obtained data. A formal analysis process then proceeded, allowing for relationships to be drawn between SMU sizes and the aforementioned key value drivers for Century Mine. A vast majority of the analysis was conducted in Microsoft Excel using an optimisation spreadsheet that drew data from various arranged sources within the workbook.

**RESULTS AND DISCUSSION**

**Century Mine**

**Practical optimisation**

The current SMU block size in place at Century has dimensions of 5 m × 5 m × 3 m, equivalent to a volume of 75 bank cubic metres (bcm). This size was originally chosen as it is considered somewhat of a convention within the industry. An operational survey was conducted on-site and it was determined the smallest practical SMU size was in fact 9 m × 9 m × 3 m. This is due to the following reasons.

Firstly, Komatsu (2009) specifications for the three excavators on-site, PC1800, PC2000 and PC3000, denote that the average dig reach is approximately 9 m in length, as seen in Figure 1.

Further analysis on the dig span revealed that the Komatsu excavators at Century require a lateral working area of at least 9 m as well. The excavators cannot selectively delineate material within this dimensional range, therefore limiting their selective capacity to a length of 9 m and a width of 9 m.

Finally, the bucket specifications for the excavators depict a bucket depth of 3 m on average, as seen in Figure 2. Once again, this limits the ability of the excavator to freely separate ore from waste material within this 3 m boundary.

Thus, it can be seen that any SMU size smaller than 9 m × 9 m × 3 m is impractical for the operation at Century.

**Economic optimisation**

Data has been amalgamated and assessed for the impact that it has on the six key value drivers for Century Mine’s operation. It was found that:

- mined grade increasingly decays further with an increasing SMU size and this is due to a decreasing selectivity
- mine production rates increase linearly up to the equipment threshold with an increasing SMU size
- ore mining costs decrease linearly with an increasing SMU size due to improved equipment utilisation
- milling rates must increase with an increasing SMU size in order to offset the decreased mined grade and produce the same quote of metal as the status quo
- in turn, milling recovery drops with an increased SMU size, as float resonance and grinding cycle times decrease
• milling and asset management costs increase with an increased SMU size as wear and tear on milling equipment increases with an increased throughput rate.

A NPV optimisation was conducted, incorporating these findings and the current foreign exchange and commodity prices for the Australian dollar and zinc spot price respectively. It was found that, for a 12 per cent/annum discount rate, over the remaining mine life of approximately 2–2.5 years at Century, the SMU size that will maximise the NPV of the in situ ore the greatest is 250 bcm. At this SMU size, compared to an SMU size of 75 bcm:

• mined grade is diluted by 0.93 per cent
• mine production rates rise by 8.04 per cent
• mine operating costs drop by 5.78 per cent
• milling rates must increase by 0.96 per cent
• mill recovery will fall by 0.18 per cent
• milling and asset management costs will increase by 0.33 per cent.

Figure 3 denotes the relationship between the percentage change in NPV and the SMU size selected.

Most importantly, the NPV increase is within the realm of 1 per cent for the remaining mine life, which will boost the project’s discounted value by approximately $5 M to $7 M. The reason for this finding can be summarised by the optimisation between ore dilution and the pace of ore extraction. This is highlighted by the fact that an SMU size of 600 bcm implemented over a 30 month period will decrease the NPV of the in situ ore by almost 5 per cent. Ultimately, it can be seen that an SMU size of 250 bcm is ideal for the proposed time period of implementation.

Stockpiling optimisation

Figure 4 denotes that the most ideal SMU size to optimise stockpiling, and improve it by the greatest extent is 525 bcm. However, for the remaining mine life this SMU size will have a detrimental impact on the project’s NPV by as much as 3 per cent. The economically optimised SMU size of 250 bcm will improve ore stockpiling by as much as 1400 t/day, significantly helping the operation mitigate stockpiling issues.

Hypothetical scenarios

The economic optimisation conducted earlier can be generalised for longer time periods, riding on the entirely hypothetical situation that the Century deposit had the ore to provide for this scenario to pan out. This is useful for the purposes of showing this study’s relevance to the wider mining industry because it reflects on other likely industry situations. It can be seen in Figure 5 that the most optimal SMU size increases with an increasing duration of implementation. This concept can be explained by the time value of money, whereby with an increased time period, a greater demand is placed on the procurement of a product now than in the future. To achieve this, an increased SMU size is required, boosting production rates significantly.

Even more interesting is the potential increase in NPV that could be achieved in these hypothetical situations with the SMU sizes proposed for various time lengths. Figure 6 illustrates that improvements of excess of 10 per cent are quite plausible.

FINDINGS FROM ANALYSIS

Significance to Century Mine

Most importantly, there is a potential to improve the NPV of the in situ ore by as much as 1 per cent which will equate to a gross pre-tax discounted cash flow improvement of approximately $5 M to $7 M. At a time when the price of zinc
A Critique of Selective Mining Unit Sizing at Century Mine to Optimise Productivity with Dilution

In hindsight it is easy to say that the Century project could have been even more profitable if this study was conducted sooner, however there is still 30 months of mining that this proposed SMU size can be implemented to help have a positive financial impact. A re-evaluation of the ultimate pit limit could also be investigated, as the mine’s economics do adjust with this altered SMU size. Due to the reasonably exhausted orebody it is expected that any pit limit changes would be minimal.

Significance to the mining industry

On a wider level, this study highlights that there is an equilibrium point between dilution and production. Too often there is the misconception held that ore mining must be as selective as possible and that dilution is an unfathomable outcome. Excessive focus on selectivity hampers mining rates significantly, and serves to delay ore extraction. To put an analogy to the situation, using a hand shovel to extract ore is particularly quick, whereas using a waste hydraulic shovel to extract ore is significantly, and serves to delay ore extraction. To put an analogy to the situation, using a hand shovel to extract ore is particularly quick, whereas using a waste hydraulic shovel to extract ore is significantly, and serves to delay ore extraction. To put an analogy to the situation, using a hand shovel to extract ore is particularly quick, whereas using a waste hydraulic shovel to extract ore is significantly, and serves to delay ore extraction.

Importantly, there are some key downstream effects out of the open pit if the SMU size is adjusted. This serves to make the study even more complicated and, therefore, extensive modelling unique to each site must be conducted if an assessment of this type was to be conducted at various other hardrock minds. One key take away message is that the faster delivery of ore can deliver significant double digit percentage improvements in the NPV of an orebody, and, such a study, if conducted at the feasibility stage of a project, could help to significantly boost a project’s economic credentials.

A further important message from this project is that the most optimal SMU size increases in line with the time period of implementation. This reinforces the time value of money wherein a tonne of zinc metal in the hand today is worth more than one tonne of zinc metal tomorrow, assuming prevailing conditions. As the effect of discounting compounds itself, the time value of money is stressed even more, up to the point in the long-term where it is more beneficial to mine the ore with methods more akin to bulk mining as opposed to a selective manner.

Ultimately, this project shows that often the most significant improvements in profitability can be achieved by revisiting the ground roots of mining and working on optimising the basics. Flamboyant innovation is definitely needed in the industry to maintain technological progression, but it is also important to have an appreciation and understanding of the basic principles of mining to ensure that value is continually added to an operation through day to day decisions.

CONCLUSIONS

The author set about to fill the void in the current state of knowledge by tackling and attempting to solve an issue at Century Mine related to maximising the NPV of the in situ ore that remains to be mined over the next 30 months. The task involved simultaneously optimising numerous critical conditions that impact on a mine site’s economic profitability including mining rates, mining costs, mined grades, mill throughput, mill recovery and milling costs. In the act of conducting this analysis, two key conclusions can be drawn. The mine engineering department at Century should:

7. utilise an SMU size of 9 m × 9 m × 3 m to maximise the NPV of ore at Century; or
8. utilise an SMU size of 13 m × 13 m × 3 m to maximise ore stockpiling at Century.

If the first pathway is chosen, an NPV improvement of approximately 1 per cent will be drawn on the remaining orebody, as the ore will be brought forward in the mine life as fast as feasibly possible while also maintaining a tight control on costs. Ore stockpiling will improve by approximately 1400 t/day on average. If the second pathway is chosen, ore stockpiling will improve dramatically at Century, in the order of upwards of 2500 t of ore/day. Unfortunately this is not a cost effective method over the longer term, as the NPV of the orebody will drop by approximately 2 per cent compared to the best case economic outcome.

Ultimately the SMU size should be increased from 75 to 250 bcm in order to maximise and optimise the value of the orebody at Century Mine. It was shown that due to the time value of money, the improvements that can be made through this form of implementation compound and improve upon each other for longer time periods. Hypothetically this form of optimisation could aid in long-term mines being increasingly more profitable from a discounted cashflow perspective.

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Selection of Haulage Fleet at Daisy Milano Gold Mine

B Yiu¹ and A Halim²

ABSTRACT

In the underground mining industry, one of the long-term operational issues involves longer haulage routes as the mine gets deeper with the life of mine. This longer travel route has a great impact on the productivity and cost of the operation. Choosing the best haulage fleet will result in improved productivity and lower cost per tonne of material moved. For this reason, it has motivated Daisy Milano Gold Mine, owned by Silver Lake Resources, to analyse their current Toro 50 Plus trucks against the Atlas Copco MT5010 truck. During a three month trial of both haulage options, trucking parameters were collected from a specifically designed trucking plod. This research used a time motion study to perform productivity calculations and cost analysis of the two haulage options.

Based on the assumptions made in this research, the conclusion is that the Atlas Copco MT5010 should be utilised as the preferred truck option at Daisy Milano. The Atlas Copco MT5010 has a higher average productivity of 254 tkm/hr (tonnes kilometre per hour) compared to the Toro 50 Plus with an average productivity of 160 tkm/hr. In terms of cost ($ per tkm) the MT5010 is the cheaper option by over 35 per cent when compared to the Toro 50 Plus ($0.85 versus $1.31).

INTRODUCTION

Like many other mining operations, tonnage and grade reports are presented daily at the management meeting and daily production in ounces are required to meet the production targets. In terms of production cycle, whilst the grade of the deposit cannot be changed, the quantity of material hauled can be controlled.

The study is based on Daisy Milano, an underground gold mine owned by Silver Lake Resources. The mine is located 55 km south-east of Kalgoorlie in the Mount Monger Goldfields. The current 600 kt annual production is mainly from the Haoma orebody which is steeply dipping and mined using narrow vein stopping methods such as selective longhole avoca stopping and hand-held airleg stopping. Currently, the haulage equipment used are six Sandvik Toro 50 Plus trucks. However, a three month trial of an Atlas Copco MT5010 truck (MT5010) was introduced in 2012 to compare trucking performance with the existing fleet. All underground equipment are second hand and hired from on-site contractors.

The aim of this study was to determine the better haulage option, Toro 50 Plus or MT5010, in terms of truck productivity and cost at Daisy Milano Gold Mine. The objectives were to address the following questions:

- What is the industry measurement of trucking productivity?
- What are the factors that affects trucking productivity?
- What is the difference in trucking productivity of the MT5010 compared to Toro 50 Plus?
- What are the economic costs associated with each haulage option?

This study was limited to analysis of truck performance in terms of productivity, rental and fuel costs and ventilation requirement. Maintenance and the associated costs, including tyres was not analysed as this data was not available.

METHODOLOGY

The approach as outlined in this design is critical in obtaining conclusions for this project. This project was carried out with the following steps:

- literature review to identify industry measurement of trucking productivity
- data collection with a designed ‘truck operator plod’
- determining travel distance of trucks, using mine planning software Surpac™
- determining truck productivity, setting up and using Microsoft® Excel spreadsheets
- cost analysis with mine site truck cost data
- other considerations to examine such as ventilation requirements and truck maintenance.

Industry measurement of trucking productivity

The literature review helped identify the current best industry practice of measuring trucking productivity. Even though limited literature on this topic is available, tonnes kilometre per hour (tkm/hr) was confirmed as the industry accepted measurement of trucking productivity (Carrick, Guilfoyle and Robertson, 2002; Robertson, Ganza and Noack, 2005).

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As this topic is based on a real operation, applications and case studies were explored to provide guidance to the approach method that others have used to achieve similar outcomes. Case studies involving Kanowna Belle (Kerr, 2002), Gwalia (Savage, 2008) and Hope Downs (Soegri, 2010) mines were used and provided methodology to select haulage fleets.

**Data collection**

During summer employment at Daisy Milano Gold Mine from December 2012, a ‘truck operator plod’ was generated and utilised by day and night shift truck operators to record parameters as part of the trucking cycle. The recorded parameters include the times and locations of when the truck is loaded and other comments. A copy of a trucking plod sheet is shown in Figure 1. The plods were utilised by about half of the truck operators, many simply just forgot about them or ignored them. The plod data collected was in the period of 4 December 2012 to 1 January 2013 and 1 February 2013 to 19 February 2013, totalling 44 days. The plod data was collected and collated in a database.

It is in this part that the greatest amount of error can be introduced, with the plods easily filled out incorrectly. Plods found to be incorrectly reported were excluded from the analysis in order to improve the accuracy of results.

**Determining travel distance of trucks**

At Daisy Milano Gold Mine, there are currently no design files to provide the centreline for the decline from the portal down to the bottom of the mine. There are survey pickups of each of the levels and separate sections of the decline in string files. All of the string files were merged, removing unnecessary access drives and generated one string only for the entire decline as shown in Figure 2.

**Determining truck productivity**

After all the data was entered and Surpac™ work created, the trucking productivity calculations could begin. The cycle time of each truck load was then calculated using an Excel spreadsheet.

Following this the truck productivity in tkm/hr was determined. From this information, comparison of trucking productivity of every hour was analysed.

**Cost analysis**

To perform discounted cash-flow analysis (DCF), the truck costs associated are required. The cost data required to undertake simple DCF analysis should include:

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**FIG 1** – Silver Lake Resources truck operator plod.

**FIG 2** – Schematic of Daisy Milano decline.
• truck rental cost
• truck operating cost
• truck fuel cost
• maintenance cost, including parts.

The DCF analysis will be based on the net present cost (NPC), this allows the factor of time to be incorporated into the analysis of the cost information. By completing a DCF analysis, the NPC would provide an estimate of the current cost and the cost to replace Toro 50 Plus with the MT5010. Many mining operations are reluctant to provide any cost data due to the sensitive nature of the information. However, Silver Lake Resources were willing to provide most of the cost information, regarding this trucking project only.

DATA INPUT

Operational assumptions

At Daisy Milano, the operation is divided into two 12-hour shifts: day and night, every 24 hours. Day shift begins from 6:00 am until night shift commences at 6:00 pm until the start of the next day shift.

According to the manufacturer specification handbook of the haulage options, both trucks have a designed payload of 50 t. However, to perform trucking reconciliation at Daisy Milano, tonnages of truck load are not weighed and the payload of each truck load is taken as a constant of 32 t. This value is agreed by the engineers and geologists on-site given past data and reconciliations. This value is critical in the calculation of the productivity of the truck as described later.

Due to the limited decline access and pathways, both truck options are assumed to have used the same travel routes to reach the same destinations.

From the existing Toro 50 Plus fleet at Daisy Milano, five trucks were utilised in this study. They are named as ‘UT001, UT002, UT006, UT007 and UT008’. During the trial period, the MT5010 was named as ‘UT010’.

Trucking plods

The trucking plod, as shown in Figure 3, when filled out properly, represents the time motion of the truck. It provides the time and location when the truck is filled with material to when the material is dumped onto either the run of mine (ROM) or into the backfill stope.

The schematic of the two haulage truck cycles, to ROM or to the backfill stope is shown in Figures 4 and 5, respectively. For this study, the path of the main truck cycle concerned was the travel of the truck after it has been loaded and prior to dumping of material. The difference in loading and dumping time for the two options considered is negligible based on site observation that both trucks performed these tasks in a similar time frame. Therefore it has not been included in the analysis. According to the truck specifications given by the respective manufacturer’s handbooks and based on site observations, both trucks achieve very similar performance whilst travelling empty.

Travel time

Using the information entered into Excel, the travel time of each load was calculated from the time at which the truck was loaded and when the truck dumped the material. Travel time information was used as a direct indicator of production as a constant payload of 32 t was utilised. It can be summarised that travel time is inversely proportional to production, when comparing the same travel routes with constant payload. For example, Truck ‘X’ travel time to complete Route ‘A’ is less than Truck ‘Y’ travel time to complete Route ‘B’.
than that of Truck ‘Y’ on the same route. Therefore, Truck ‘X’ is more productive.

RESULTS AND DISCUSSION

Productivity

From the trucking data entered into Excel, calculations to obtain productivity in tkm/hr were performed. As discussed previously, tkm/hr is an approach used in this study to analyse productivity and uses the following formula:

\[
\text{tkm/hr} = \frac{t \times km}{hr}
\]

where:
\( t \) = payload in tonnes
\( km \) = distance travelled in kilometres
\( hr \) = time taken in hours

The assumption of the 32 t payload constant was utilised in this study for all of the truck loads. Travel time and distance are utilised as discussed previously.

Toro 50 Plus

The tkm/hr for all Toro 50 Plus truck loads during the period was calculated and categorised into the 12 working hours/shift. The average load tkm/hr for each hour was graphed for both day and night shift as shown in Figure 6. In Figure 6, it is evident that the number of loads was low during three main hours of the shift, first hour, sixth hour and the last hour. This low number of loads would be a reflection of the start of shift, crib time and end of shift, respectively. It is also worth mentioning that the number of loads obtained for the first hour and the last hour is very low, 15 and one respectively. This would provide a less accurate indication of the productivity at these hours compared with other hours with many loads. The trend for both day and night shift productivity is very similar and both provide the result of 160 tkm/hr as the average productivity of Toro 50 plus.
Selection of Haulage fleet at Daisy Milano Gold Mine

MT5010
The same analysis was performed for the MT5010, calculating the productivity per hour, as shown in Figure 7. The same comment can be made in regards to the accuracy of the calculated productivity of the first and last hour about the MT5010, given that only data from two loads were recorded during each of these hours. The trend for both day and night shift, like Toro 50 Plus, are very similar, peaking between the first hour until sixth hour for crib time and between the seventh hour after crib time and the end of shift (last hour). It can be seen from the ninth hour to the eleventh hour that the number of loads recorded from day shift is significantly greater than that of night shift. However the productivity at these hours were lower for day shift than night shift. This is a good example of how this design to analyse productivity of the truck is not biased by a greater number of loads recorded. The average productivity of the MT5010 calculated was 254 tkm/hr.

Comparison
To compare the productivity of each of the truck options, the average productivity from all the loads was analysed. In Figure 8, the productivity in tkm/hr for both Toro 50 Plus and MT5010 truck for each hour of the day is plotted, with an average line representing the productivity of each of the options.

MT5010 has a higher average productivity of 254 tkm/hr compared to Toro 50 Plus with an average productivity of 160 tkm/hr. This shows that the MT5010 is more productive than the existing Toro 50 Plus. Based on productivity analysis alone, the MT5010 is the preferred haulage option.

Other considerations
Ventilation requirement
According to the Western Australian Mines Safety and Inspection Regulations 1995 (WA Government, 1995), Regulation 10.52 (7):

A diesel unit referred to in subregulation (5)(b) must have a ventilation volume rate of not less than 0.06 cubic metres per second per kilowatt of the maximum rated engine output specified by the manufacturer for the fuelling and timing configuration at which the engine has been set.

This is based on the limited data available and that for both truck options the subregulation (5)(b) applies to all second hand equipment.

From the individual manufacturer’s specification for each of the trucks, it is to note that the MT5010 has an additional 93 kW of engine power than that of Toro 50 Plus. Whilst this additional power is expressed as faster truck speed, additional ventilation requirements as a result will need to be considered.

The additional ventilation requirement was calculated as follows.

\[
\text{Total ventilation requirement (m}^3/\text{s)} = \text{Airflow per kW} (\text{m}^3/\text{s/kW}) \times \text{total engine power (kW)} = 0.06 \times 93 = 5.58 \approx 6 \text{ m}^3
\]

From the above, an additional 6 m³ of ventilation will be required per truck if a MT5010 is to replace a Toro 50 Plus truck. This additional ventilation should be included in the ventilation design and monitored as per normal ventilation.

Maintenance
Maintenance data is obtained for two different reasons, performance evaluation and measurement of reliability. By examining the maintenance information on the trucks, information that is incorporated as ‘downtime’ can be examined in detail. For example, downtime for a truck can be due to a faulty part not being repaired properly. This would show up as many maintenance repairs and problems arise due to that certain part. The reliability of the truck, whether it is Toro 50 Plus or MT5010, can be evaluated given the...
Maintenance downtime that it incurs. It can be expressed that the less downtime due to mechanical failure means that the more reliable the equipment is.

For this study, no maintenance data could be obtained from the mine site and therefore was not considered.

Financial analysis

In this part of the study, the costs associated with the truck options were analysed and compared. At Daisy Milano, all of the underground equipment including trucks are hired, therefore capital cost is not considered. To operate the truck for one month, the cost for the truck operators are the same to both options of $55 000/month.

Rental cost

The cost associated with Toro 50 Plus truck is based on the flat rental rate of $70 000/month. This flat rental rate includes the base rental of the truck and operating cost minus fuel and maintenance cost, regardless of the number of hours that the truck is operated for.

The cost associated with the MT5010 is split into fixed and operating cost. The fixed cost of $34 000/month is considered as the ‘base rate’ or the cost to rent the truck, regardless of the number of hours that the truck completes. The operating cost is charged at $86/hour and is based on the number of ‘operating hours’ that the truck accumulates over the monthly periods. For budgeting reasons and during the use on-site during the trial period, 420 operating hours/month is utilised. The total cost associated with the MT5010 can be summarised as $70 120/month (as budgeted).

Fuel cost

Diesel fuel cost analysis was performed in this study for both of the haulage options. Due to the limited fuel data obtained, fuel data in litres was only available for certain times of the study period. Operating hours of this period was assumed to be a ratio of the monthly operating hours including utilisation and availability. Fuel consumption per hour for this period was calculated from the fuel consumed divided by the operating hours. This can be simplified as follows:

\[
\text{Fuel consumption/hour (L/hr)} = \frac{\text{Fuel usage (L)}}{\text{Operating hours (hr)}}
\]

For calculating fuel cost, the cost of diesel was obtained from the mine. The full price of diesel is $1.38/L, however the mine only pays $1.03. This is due to the deduction of 35 cents from off-road use government rebate. Fuel cost per month was calculated as follows.

\[
\text{Fuel cost per month ($)} = \text{Cost of fuel per litre ($)} \times \text{fuel consumption (L/hr)} \times \text{operating engine hours (hr)}
\]

Fuel is a large component of the haulage cost and was analysed in this study for the two options. The recorded number of operating engine hours for Toro 50 Plus and MT5010 obtained was different during the study period, due to the biased nature of the haulage fleet trial. To ensure that the normal cost of fuel per month was more accurately estimated, a constant 420 operating engine/month was utilised. Using $1.03 as the cost of diesel, a summary of the monthly fuel cost for each truck option is summarised in Table 1.

Discounted cash-flow

By using NPC analysis, the time value of money can be included. The net present values can then be compared as discounted cash-flow analysis. In this study, no capital costs are associated and therefore NPC analysis could not be included.

<table>
<thead>
<tr>
<th>Truck</th>
<th>Fuel consumption (L/hr)</th>
<th>Fuel usage per month (L)</th>
<th>Cost per month</th>
</tr>
</thead>
<tbody>
<tr>
<td>MT5010</td>
<td>47</td>
<td>19 740</td>
<td>$20 332</td>
</tr>
<tr>
<td>Toro 50 Plus</td>
<td>42</td>
<td>17 640</td>
<td>$18 169</td>
</tr>
</tbody>
</table>

TABLE 1
Monthly fuel cost for both haulage options.
**Cost comparison**

By comparing the truck costs of two options, $70,000 for Toro 50 Plus and $70,120 for the MT5010, the cost difference of $120/month is almost negligible in the context of haulage cost. The break-even cost between the two options occurs when the MT5010 costs the same flat rate as Toro 50 Plus truck of $70,000/month. With analysis as shown in Figure 9, the break-even cost would occur when the MT5010 accumulates about 418 operating hours/month.

Figure 9 also shows that the cost of the MT 5010 will be less than that of Toro 50 Plus if the operating hours are less than 418 hours/month. Operating hours per month far above the 418 break-even hours could see that Toro 50 Plus would be preferred, given its lower cost.

The total monthly cost of the two options is the sum of the rental cost and fuel cost for each month as summarised in Table 2. This is calculated by the formula:

\[
\text{Total monthly cost (\$)} = \text{rental cost (\$)} + \text{fuel cost (\$)}
\]

To compare the two options with different productivity, the cost per tkm was calculated. The cost per tkm is calculated from the total monthly cost divided by the total tkm per month. Using the assigned operating hours of 420 engine hours/month, the total tkm per month was calculated. A summary of the cost per tkm is shown in Table 3.

**Comparison**

Whilst the sum of the rental cost and fuel cost provides a monthly cost of the truck, it does not provide any information of how productive the truck is in the month. In terms of rental and fuel cost per month, Toro 50 Plus truck is $2273 cheaper than that of MT5010 ($88,169 versus $90,442). However, using productivity data as obtained, the cost per tkm was determined using the assigned 420 engine hours/month. The result concludes that MT5010 is the cheaper option by $0.46/ tkm when compared to Toro 50 Plus ($0.85 versus $1.31). This is a 35 per cent cost savings/tkm when MT5010 trucks are used.

It is advised that further study and analysis using field maintenance cost data should be undertaken. This will provide a more comprehensive and reliable cost analysis for the two options.

**TABLE 2**

Comparison of the total monthly cost for both options.

<table>
<thead>
<tr>
<th>Truck</th>
<th>Rental cost</th>
<th>Fuel cost</th>
<th>Total cost per month</th>
</tr>
</thead>
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<tr>
<td>MT5010</td>
<td>$70,120</td>
<td>$20,322</td>
<td>$90,442</td>
</tr>
<tr>
<td>Toro 50 Plus</td>
<td>$70,000</td>
<td>$18,169</td>
<td>$88,169</td>
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<tr>
<td>Difference</td>
<td>$120</td>
<td>$2,153</td>
<td>$2,273</td>
</tr>
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</table>

**TABLE 3**

Summary of cost per tonnes kilometre (tkm) per hour for both options.

<table>
<thead>
<tr>
<th>Truck</th>
<th>Total cost per month</th>
<th>tkm per month</th>
<th>Cost per tkm</th>
</tr>
</thead>
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<tr>
<td>MT5010</td>
<td>$90,442</td>
<td>106,680</td>
<td>$0.85</td>
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<tr>
<td>Toro 50 Plus</td>
<td>$88,169</td>
<td>67,200</td>
<td>$1.31</td>
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<td>Difference</td>
<td>$2,273</td>
<td>39,480</td>
<td>$0.46</td>
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**CONCLUSIONS**

This study has found an indication that MT5010 is the better haulage option, over the existing Toro 50 Plus, in terms of truck productivity and cost at Daisy Milano Gold Mine.

MT5010 has a higher average productivity of 254 tkm/hr compared to Toro 50 Plus with an average productivity of 160 tkm/hr. This represents MT5010 as the more productive haulage fleet over the existing Toro 50 Plus. Based on the productivity analysis alone, the MT5010 is the preferred haulage option.

The financial analysis based on 420 monthly operating hours concludes Toro 50 Plus has a $2,273 cheaper monthly rental and fuel cost than the MT5010 ($88,169 versus $90,442). Based on rental and fuel cost alone, Toro 50 Plus is the preferred haulage option.

However, analysis of productivity and economics of the truck separately does not provide an indication of how productive the truck is and how much that productivity would cost. Therefore an economic value based on productivity was determined, as cost per tkm. The result concludes that MT5010 is the cheaper option by $0.46/tkm when compared to Toro 50 Plus ($0.85 versus $1.31).

Based on the assumptions of this study, as stated previously, the project for Daisy Milano Gold Mine concludes that:
MT5010 truck compared to Toro 50 Plus is more productive and more cost effective per tkm.

MTS010 should be utilised as the preferred haulage option, replacing the existing Toro 50 Plus.

ACKNOWLEDGEMENTS

This research study would not have been possible without the support and assistance of several people and parties. The authors would like to thank Daisy Milano Gold Mine for providing the data necessary for this study, in particular acknowledges the assistance provided by Mr Nicholas Chernoff for his time and assistance. The authors would also like to thank Ms Kirstan Lee for her support throughout the study.

REFERENCES


Soegri, R S, 2010. Analyse the availability and utilisation of the production fleet at Hope Downs, BEng thesis (unpublished), Western Australian School of Mines, Curtin University, Kalgoorlie.

# APPENDIX 1 – PROGRAM 2013 MEA STUDENT CONFERENCE

**Date:** 2013 MEA Student Conference, 21 October 2013  
**Venue:** School of Mining Engineering, University of New South Wales  
Rm G51, Old Main Building, Kensington campus

<table>
<thead>
<tr>
<th>Time</th>
<th>Activity</th>
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<tbody>
<tr>
<td>08:00 – 08:30</td>
<td>Arrival Tea and Conference Registration</td>
</tr>
<tr>
<td>08:30 – 08:45</td>
<td>Welcome &amp; Opening of the Symposium by P. Hagan &amp; P. Dowd</td>
</tr>
<tr>
<td>08:45 – 10:30</td>
<td>Session 1</td>
</tr>
<tr>
<td>08:45 – 09:00</td>
<td>Eric Lou (UNSW): The effects of particle stabilisers on bubble generation, coalescence and breakup</td>
</tr>
<tr>
<td>09:00 – 09:15</td>
<td>Charles Byron Smith (UQ): A Critique of SMU sizing at Century Mine to optimise productivity with dilution</td>
</tr>
<tr>
<td>09:15 – 09:30</td>
<td>Mark Kong (WASM): Aboriginal land rights agreement – A Western Australia case study</td>
</tr>
<tr>
<td>09:30 – 09:45</td>
<td>Graham Ball, Jessee Clark, Michele Gifford, Rajendra Rathod (UA): Probabilistic stability analysis of rock excavations</td>
</tr>
<tr>
<td>09:45 – 10:00</td>
<td>He Ren (WASM): Using clustering method for block aggregation in open pit mine planning</td>
</tr>
<tr>
<td>10:00 – 10:15</td>
<td>Nicholas Anthony Butel (UQ): Estimation of in-situ rock strength using geophysics</td>
</tr>
<tr>
<td>10:15 – 10:30</td>
<td>Prudence Grace Fischer (UNSW): Identify and optimise parameters influencing blast performance within basal till material with groundwater inflow</td>
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<tr>
<td>10:30 – 10:50</td>
<td>Morning Tea in Room G38</td>
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<tr>
<td>10:50 – 12:35</td>
<td>Session 2</td>
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<tr>
<td>10:50 – 11:05</td>
<td>Tyler Bastian, Brendan Connolly, Cassandra Lazo Olivares, Natalie Tziantzis (UA): The study of the strength and deformability of rocks under random and</td>
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<tr>
<td>11:20 – 11:35</td>
<td>Gareth Thomas Swail (UQ): Analysis of heat reduction opportunities for the George Fisher north expansion project decline</td>
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<tr>
<td>11:35 – 11:50</td>
<td>Stefan Andrez Skorut (WASM): Optimising cut-off grade strategy for seabed massive sulphide operations</td>
</tr>
<tr>
<td>11:50 – 12:05</td>
<td>Yu Yang (WASM): Surface fauna as a new mining exploration method – fact or fiction?</td>
</tr>
<tr>
<td>12:20 – 12:35</td>
<td>Jake Lees Small (UNSW): Time benefit analysis of through-seam blasting</td>
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<tr>
<td>12:35 – 13:45</td>
<td>Lunch in the Chancellors Garden–Fountain</td>
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<tr>
<td>13:45 – 15:15</td>
<td>Session 3</td>
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<tr>
<td>13:45 – 14:00</td>
<td>Yunchao Hu (UQ): Workforce hazard mapping and its optimum support design for North Goonyella Coal</td>
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<tr>
<td>14:00 – 14:15</td>
<td>Leigh Bowen, Matthew Harding, Ashlee Kiss, Samuel Locken (UA): Deep open-pit mining – Rock haulage Optimisation</td>
</tr>
<tr>
<td>14:30 – 14:45</td>
<td>Kristian Lee (WASM): Comparison of methods for opening initial voids in longhole open stoping blasting</td>
</tr>
<tr>
<td>14:45 – 15:00</td>
<td>Daniel Boswell (WASM): Using Artificial Neural Network (ANN) to predict stope overbreak at Platino Gold Mine</td>
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<tr>
<td>15:00 – 15:15</td>
<td>Jingyu Chen, Menghua Gu, Jiachang Liu, Ke Zhang (UA): Simulation and Animation of a Surface Gold Mine</td>
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<tr>
<td>15:15 – 15:30</td>
<td>Afternoon Tea &amp; Meeting of the Judging Panel</td>
</tr>
<tr>
<td>15:30 – 16:00</td>
<td>Announcement of Prizes &amp; Closing of the Conference</td>
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### APPENDIX 2 – PRESENTERS AT MEA STUDENT RESEARCH CONFERENCES, 2009–2013

<table>
<thead>
<tr>
<th>Year</th>
<th>University</th>
<th>Student presenter(s)</th>
<th>Presentation title</th>
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<tr>
<td>2013</td>
<td>UoA</td>
<td>Paul Bryson</td>
<td>Deep drilling performance estimation using rock mass characterisation</td>
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<td></td>
<td></td>
<td>Jingyu Chen</td>
<td>Simulation and animation of a surface gold mine</td>
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<td>Jesse Clark</td>
<td>Probabilistic stability analysis of rock excavations</td>
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<td></td>
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<td>Mathew Harding</td>
<td>Deep open-pit mining: rock haulage optimisation</td>
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<td></td>
<td></td>
<td>Cassandra Lazo Olivarès</td>
<td>The study of the strength and deformability of rocks under random and systematic cyclic loading</td>
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<td>UNSW</td>
<td>Prudence Fischer</td>
<td>Identify and optimize parameters influencing blast performance within basal till material with ground water inflow</td>
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<td></td>
<td></td>
<td>Jamie Jongeblod</td>
<td>Coal project valuation — a hedonic pricing approach</td>
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<td></td>
<td></td>
<td>Eric Law</td>
<td>The effects of particle stabilisers on bubble generation, coalescence an breakup</td>
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<tr>
<td></td>
<td></td>
<td>Stefan Skorut (equal third prize)</td>
<td>Optimising cut-off grade strategy for seabed massive sulphide operations</td>
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<td>Jake Small (second prize)</td>
<td>Time benefit analysis of through-seam blasting</td>
</tr>
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<td>Nicholas Butel (fourth prize)</td>
<td>Estimation of in situ rock strength using geophysics</td>
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<td>Yanchao Hui</td>
<td>Workface hazard mapping and its optimization support design at North Goonyella</td>
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<td></td>
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<td>Robert Lucas</td>
<td>Optimisation of waste-dump lift heights for pre-stripe operations at Meandu Mine, Queensland</td>
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<tr>
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<td>Charles Smith</td>
<td>A critique of SMU sizing at Century Mine to optimize productivity with dilution</td>
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<td>Gareth Steel</td>
<td>Analysis of heat reduction opportunities for the George Fisher north extension project decline</td>
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<td></td>
<td>WASM (Curtin)</td>
<td>Daniel Boxwell</td>
<td>Using artificial neural network (ANN) to predict overbreak at Plutonic Gold Mine</td>
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<td></td>
<td></td>
<td>Mark Kong</td>
<td>Aboriginal land rights — a Western Australia case-study</td>
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<td>He Ren (equal third prize)</td>
<td>Using clustering method for block aggregation in open pit mining</td>
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<td></td>
<td>Kirstan Lee (first prize)</td>
<td>Comparison of methods for opening initial voids in longhole open-stoping blasting</td>
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<td>Yu Yang</td>
<td>Surface fauna as a new mining exploration method - fact or fiction?</td>
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<td>Dermott Sundquist and Davood Jafari</td>
<td>Multi-objective optimisation of mining-metallurgical systems</td>
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<td>Paul Carmichael</td>
<td>An investigation into semi-intact rock mass representation for physical modelling of block caving mechanics zones</td>
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<td>Prediction and modelling of blast vibration and its effects at Glendell Colliery</td>
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<td>Strategic project risk management for an emerging miner</td>
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<td>Vanessa Collins</td>
<td>Maximising production rates at Brockman 4 by minimising truck delays at the crusher</td>
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<td>Casey Costello (first prize)</td>
<td>Grizzly modifications at Ridgeway Deeps block cave gold mine</td>
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<td>Brenton Goves (second prize)</td>
<td>Continuous surface miner operations at Fortescue</td>
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<td>WASM (Curtin)</td>
<td>Seung Hyeon Lee</td>
<td>Comparing neural networks with JKSimblast prediction model in purpose of optimal ground vibration induced by blasting</td>
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<td></td>
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<td>Vadim Strukov</td>
<td>A comparative study of truck cycle time prediction methods</td>
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<td>Weilin Wang</td>
<td>ANSYS for stress analysis of underground structures</td>
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<td>James Boffo</td>
<td>Acoustic emission monitoring of impregnated diamond drilling for deep exploration</td>
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<td>Owen Riddy (first prize)</td>
<td>Developing a truck allocation model for Bengalla Mine</td>
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<td>Brendan Murphy</td>
<td>Development of an underground coal scheduling and simulation program</td>
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<td>Colin Thomson</td>
<td>Stope analysis of Golden Grove under high stress conditions</td>
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<td>Adam Schwartzkopf and Daniel Hardean</td>
<td>Comparisons between three-dimensional yield criteria for fractured and intact rock</td>
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<td>James Tibbett</td>
<td>Failure criteria of three major rock types affecting Ridgeway Deeps</td>
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<td>An investigation into final landform criteria required for a safe, stable, sustainable and non-polluting landform in the Bowen Basin</td>
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<td>Nathan Colli</td>
<td>Analysis of the principles for sound waste dump design and placement</td>
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<td>Kali Dempster</td>
<td>Quantifying time-dependent rock behaviour at Ridgeway Deeps block cave operation</td>
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<td>Michael Baque</td>
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<td>Paste fill for stope stability at the Challenger Gold Mine</td>
</tr>
</tbody>
</table>
## AUTHOR INDEX

<table>
<thead>
<tr>
<th></th>
<th>Authors</th>
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</tr>
</thead>
<tbody>
<tr>
<td><strong>B</strong></td>
<td>Ball, G H</td>
<td>1</td>
</tr>
<tr>
<td></td>
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<td>7</td>
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<tr>
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<td>Berriman, E L</td>
<td>15</td>
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<td></td>
<td>Bosompem, M K</td>
<td>63</td>
</tr>
<tr>
<td></td>
<td>Boxwell, D</td>
<td>21</td>
</tr>
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<td><strong>C</strong></td>
<td>Clark, J B</td>
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<td>Connelly, B J</td>
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<td><strong>H</strong></td>
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<td>15, 39</td>
</tr>
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<td>Halim, A</td>
<td>27, 69</td>
</tr>
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<td><strong>J</strong></td>
<td>Jang, H</td>
<td>21</td>
</tr>
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<td><strong>K</strong></td>
<td>Knights, P</td>
<td>33</td>
</tr>
<tr>
<td><strong>L</strong></td>
<td>Lazo Olivares, C S</td>
<td>7</td>
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<tr>
<td></td>
<td>Lee, K</td>
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<td>39</td>
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<td>63</td>
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<td>Rathod, R</td>
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<td>Ren, H</td>
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<td>Saydam, S</td>
<td>51</td>
</tr>
<tr>
<td></td>
<td>Skorut, S A</td>
<td>51</td>
</tr>
<tr>
<td></td>
<td>Smith, C B</td>
<td>63</td>
</tr>
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<td>Taheri, A</td>
<td>7</td>
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<tr>
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<td>Topal, E</td>
<td>21, 45</td>
</tr>
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<td><strong>X</strong></td>
<td>Xu, C</td>
<td>1</td>
</tr>
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<td><strong>Y</strong></td>
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