Load-Haul-Dump operations in underground mines

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Mining and Rock Engineering
Load-Haul-Dump operations in underground mines

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The research work presented in this thesis is part of the Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future (I² Mine) project, WP 2 subtask 2.1.1, Optimized Medium Range Mine Operation Scheduling (OMMS) project and Sustainable Intelligent Mining Systems (SIMS) project, WP 4 Integrated process control. The research was carried out at the Division of Mining and Geotechnical Engineering at Luleå University of Technology. The projects cover a wide range of mining operations and mining systems. In this thesis, the focus is on Load-Haul-Dump operations, a common unit in underground mines. Before moving on, I would like to thank many people for sharing their knowledge, experiences, and reflections. I would like to thank my supervisor, Associate Professor Jenny Greberg, for her support and technical advice and for allowing me to take part in the three projects. My thanks go to Dr. Abubakary Salama for his advice and cooperation during and after the projects. It would have been difficult to manage without others who gave me advice and shared their experience, data, and knowledge throughout the study, namely those at AtlasCopco (Epiroc AB), LKAB, ABB and Boliden.

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ABSTRACT

The research presented in this thesis addresses several aspects of loading operations in underground mines, in particular tools and equipment selection. It also addresses the flexibility of the fleet when subject to substantial disturbances, such as ore pass loss, and proposes integration of the scheduling system with discrete event simulation. The thesis begins with a study of discrete event simulation (DES) tools for loading operations in an underground mining system. The results show the benefits of using simulation but also the drawbacks. The thesis presents an analysis of energy consumption and exhaust gas emissions from diesel and electric LHDs. The results show the potential energy savings with the use of electric LHDs. Next, it focuses on the LHD operations affected by long-term ore pass loss (unavailability). It shows the effects on the production system (the ventilation requirements, production and waiting times when too many LHDs operate in the area affected by an ore pass loss) and highlights the need for a flexible solution and a mitigation strategy. Finally, the thesis studies the integration of ABB’s Ability Operations Management System (OMS) with the SimMine simulation model and how this affects LHD operations. The results show the benefits of using the joined platform as a testbed and decision support system.

**Keywords:** LHDs, DES, simulation, underground mines, energy consumption, ventilation requirements, ore pass loss, production scheduling, process control
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The aim of this chapter is to introduce the reader to the research problem, research scope and research significance. It explains why the study was performed.

Mining was one of the earliest human activities and continues to play an important role in human existence by supplying basic resources (Hartman and Mutmansky, 2002). Mining refers to the extraction of naturally occurring mineral substances. If extraction is carried out from the earth surface, the mining is termed surface mining; if the extraction is carried out beneath the earth’s surface, it is called underground mining (Hartman and Mutmansky, 2002). With time, underground mines extract mineral deposits at greater depths moving from surface mining to underground mining. This is when the depth of the deposit or the stripping ratio of overburden to ore becomes excessive for surface exploitation (Hartman and Mutmansky, 2002). With increasing mining depth, efficient mining processes are extremely important because distances, stresses, seismicity, energy consumption, ventilation requirements and travelling distances will increase. To cope with the numerous challenges of the future, more attention must be paid to automation, to flexibility and to the transition from diesel to electric equipment.

A typical, deep underground mine uses both vertical and horizontal systems. Material handling is usually performed in steps; the most common unit operations are drilling, blasting, loading, hauling and hoisting of the material. One popular way to transport the material over short distances from the face is to use underground rubber-tired loaders such as Load-Haul-Dump (LHD) machines. These machines have been used to move the material from the face to the dumping locations for over half a century. The first loader was produced by Wagner (today Epiroc) in 1959, followed by Teletram in 1962 and Scooptram in 1963 (Chadwick, 1996). Now, more than 75% of all underground metal mines use LHDs (Taitya, 2013). Responding to the need to improve the productivity of the
machines, equipment manufacturers have introduced different types of LHDs that vary in type, size or profile (low/high). Their selection in a mine planning process is based on three important factors: tunnel clearance, tire life and production capabilities (Tatiya, 2013).

The majority of LHDs currently have diesel engines because of their flexibility, durability and high efficiency (Pronk et al. 2009). However, diesel-powered machines emit a significant amount of CO₂, contribute significantly to ventilation costs and produce harmful or excessive quantities of gasses that pollute the air in the mine. Given the strict ventilation requirements (stated in regulations), high environmental impact and high operational costs, replacing old diesel equipment with newer machines may not be enough. It may be necessary to select an alternative power source.

With more stringent regulations on mine air quality and CO₂ emissions, one alternative is to use electric equipment; as well as complying with regulations and being a more sustainable solution, it may generate major ventilation cost savings (Paraszczak et al. 2013, Chadwick, 2008). The power for the electric machines can come from a trailing cable that can be deployed or retracted (tethered), overhead cable (trolley-line) or other electrical equipment, such as batteries (Paraszczak et al. 2014; Varaschin and Souza, 2015). The disadvantages of tethered cables include reduced versatility, cable faults, cable relocation issues, limited haul range, restricted movement and cable wear (Chadwick, 1996; Paraszczak et al. 2013; Paterson and Knights, 2012; Paraszczak et al. 2014). An overhead power line does not provide flexibility because it has a fixed route (Paraszczak et al. 2013). The drawback of batteries is a low energy storage capacity that results in a need for power stations and frequent recharging (Paraszczak et al. 2013). An electric machine with a 165 kW battery and average payload of 6.8 tonnes can run for around 4 hours (Epiroc, 2018), whereas a similar diesel machine can run for around 10-12 hours before needing to refuel. However, many equipment manufacturers are striving to increase the battery storage capacities and solve the
need to frequently recharge batteries by, for example, using two batteries and charging one while other is in use.

With a wider palette of LHDs available on the market, the selection of the right type of machine and the optimal fleet size is critical. In the mining environment, especially in areas which are difficult to access or extremely hazardous, reducing manual and repetitive work, increasing overall efficiency and increasing productivity by increasing the level of automation are obvious goals (Chadwick, 2008). However, the wrong choice of machine could result in increased emissions, increased operational costs and increased operational time. Often there are many alternative solutions, making the choice even more difficult; for example, when choosing whether to buy fewer machines with larger bucket sizes or more machines with smaller bucket sizes. The first option has the advantage of reducing the congestion and costs related to, for example, buying and maintaining fewer LHDs. The second option enables flexibility in cases where many production areas are in operation; however, using more LHDs in fewer active areas might result in vehicle queuing, leading to LHD underutilisation. The problem continues when allocating the selected machines to the right production locations; the current infrastructure must support the fleet integration, and the exposure to disturbances must be minimised. In cases of a major disturbance, production could be jeopardised. An example of such a disturbance is the failure of the closest unit structure to the LHD operation, for example, an ore pass loss or tunnel collapse.

To accommodate the dynamic nature of the mining environment and increasing environmental regulations, mining operations need flexible and responsive systems. An automatic scheduling system provides the production planner with fast and flexible solutions to problems and offers automatic and quickly readjusted plans for the system.

Optimal performance requires optimal solutions. However, the continuous changes occurring in the dynamic environment of a mine complicate things, making it a challenge to identify the optimal solution. For these type of situations, it is often very
difficult to have only one alternative and to make a decision without performing any analysis or studying alternative scenarios. One option is to perform experiments in the mine, but this is usually expensive and not very realistic. Another option is to use simulation. A simulation is the artificial representation of a studied system or process over time (Banks et al. 2013). Simulation is used as an analysis or design tool to predict the effects or performance of a system under varying sets of circumstances (Banks et al. 2013). To develop a simulation model, the operating characteristics are determined by observing the processes of the studied system and gathering historical data on it (Banks et al. 2013). The validated simulation model can be used to answer ‘what-if’ questions and provide a detailed analysis of the performance of the system by capturing its stochastic and dynamic character. Stochastic processes are characterised by random output variables (Banks et al. 2013). In stochastic modelling, the user studies the system’s response to the input.

1.1 Problem statement

With increased mining depth, distances, stresses, seismicity, energy consumption, ventilation requirements and travelling distances will also increase. This will affect the loading equipment and loading operations. To adapt to the dynamic nature of the mining environment and strict environmental regulations, efforts must be made to select the right type of equipment, find flexible solutions and minimise operational costs.

1.2 Objectives

The objectives of this research were to study and evaluate alternative solutions to loading operations in underground mines. More specifically, the research sought to:

- Identify the main challenges of and study alternative approaches to LHD operations;
- Identify suitable analysis tools and methods for studying the mining system, with a focus on LHD operations;
• Determine a way to optimise the fleet and manage LHD operations;
• Study the LHD operations when affected by major disturbances in the mine production system;
• Study the need for a flexible system and present an analysis of the effects of a major disturbance on the system.

The study was limited to hard rock underground mines using caving or cut-and-fill as a method of extraction. It was also limited to specific underground mines with specific operating characteristics.

1.3 Research scope

To fulfill the research objectives, the following research questions were formulated:

**RQ1** – What are the major challenges related to LHD operations in underground mining?

After answering RQ1 the following research questions were derived:

**RQ2** – How can the use of DES improve the LHD operations?

**RQ3** – How can the use of electric LHDs change the loading operations?

**RQ4** – How can automated scheduling influence the loading operations?

Table 1 shows the relationship between the appended papers and the research questions.

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Paper A shows how different simulation tools perform in analysing LHD operations. Paper B presents an analysis of the energy consumption and gas emissions for different LHD equipment in Kiirunavaara mine using DES. In Paper C the effects of an ore pass loss on LHD operations in Malmberget mine is examined. Paper D presents an analysis the ventilation requirements and costs related to LHD operations using DES. In Paper E, the concept of using a mine simulation model as a testbed platform for a mine scheduling system in which LHDs are part based on operations in Kristineberg mine is presented. The research is based on information collected from existing mines.

1.4 Significance of the research

The main contribution of this research is to increase the understanding of problems related to LHD operations and to propose relevant tools and methods and conceptual solutions for LHD operations in underground metal mines. This is done by identifying the problems related to LHD operations, using DES to improve the management of the LHD operations and studying mining disturbances and their implications for fleet strategy and equipment and software selection. The use of DES to optimise operations enables alternative simulation solutions, answers ‘what-if’ questions and provides a detailed analysis of the performance of the system by capturing its complex stochastic and dynamic character.
The aim of this chapter is to provide the reader with a better understanding of the theoretical part of the subject under study and with the practical information related to the study. This will develop the background for the next chapter where the research methodology is discussed.

2.1 Underground mining

Underground mining is the process by which various mining methods (supported, unsupported and caving methods) are employed to extract naturally occurring mineral deposits from underground. In these mines, different infrastructure and transportation systems such as truck, rail or hoisting systems are used to move or remove the material (Figure 1).

![Figure 1. Example of underground mine infrastructure (courtesy of Epiroc AB).]
An important part of the mine transportation system is loading and haulage. The loading and hauling equipment must be able to cope with sharp curves and the various conditions of the mining environment. LHDs are rubber-tired loaders used in underground mining to move the material from the face to the dumping locations designed for such conditions.

2.1.1 Challenges related to LHD operations in underground mining

The problems related to LHD operations in underground mining were identified through interviews, questionnaires and a literature review, using several case studies within the I²Mine project. Altogether 15 mines, located in Canada, USA, Chile, Australia, Zambia, Finland and Sweden were studied (Greberg and Salama, 2016). The study found that the most critical problems in deep underground mining operations are: high stresses and seismicity, rock support and ground control, high energy consumption, ventilation limitations, communication and flow of information, fragmentation, logistics, loading operations, problems with ore passes, temperature, media access and longer haul distances (Greberg and Salama, 2016). In addition, "Smart Mine of the Future" (Bäckblom et al., 2010) identified the following future challenges: safer mining, leaner mining, greener mining (energy efficiency and CO₂ reduction), increased ore recovery and reduced impact of the generated waste rock. Other challenges addressed by Mielli (2011) are the lack of specialised personnel, sustainable mining and extreme and remote mining. Wilkinson (2012) identified five challenges: capturing the true complexity of mineral deposits, generating accurate production and budget forecasts, minimizing impact of staff turnover, capitalizing on quick changing market and operational conditions and streamlining the flow of information between geologic modelling, mine planning and scheduling. Brake (2013) identified the following list of hard rock ventilation challenges: the competence of hard rock mine ventilation professionals, ventilation practices, value creation, new ventilation systems for new mining methods and equipment, training in case of emergency, increasing power costs, managing and reducing exposure levels and globalisation of the world’s
mining industry. Based on the studies above, where issues related to the rock mass transportation and its effect on production, cost and emissions were emphasized by a majority of the studied mines the following challenges related to LHD operations are being addressed in this research:

- Select the right type of equipment, find flexible and optimum solutions and minimise operational costs. Poor selection and management of an LHD fleet will result in high operational costs, high gas emissions and significantly low equipment performance.
- Automation, flexibility and to the transition from diesel to electric equipment.
- Strict ventilation requirements (stated in regulations), high environmental impact and high operational costs. Replacing old diesel equipment with newer machines may not be enough. It may be necessary to select an alternative power source.
- Production disturbances - critical for deep underground mines and affects the mining operations. With increased ventilation requirements and deeper mines, larger and more powerful ventilation systems are required, resulting in high energy costs and high ventilation requirements.

2.2 Load-Haul-Dump operations

The first trackless rubber-tired loader for underground mines was produced in 1959, followed by Teletram in 1962 and Scooptram in 1963 (Chadwick, 1996). A typical LHD used for face haulage in underground mining is shown in Figure 2. The use of LHDs is increasing because of their continuous improvement in productivity, their flexibility and their success in the harsh mining environment (Dubois et al. 2007).
To fit into the existing underground openings and the mine infrastructure, the choice of LHDs is regulated by minimum working clearance (minimum one-metre operating clearance between vehicle and sidewalls) and ventilation requirements (Dubois et al. 2007). When selecting an LHD, it is important to consider the balance between static breakout force and hydraulic breakout force, as to reduce the spinning of the wheels if the hydraulic breakout force is excessive (Dubois et al. 2007). Breakout force is the combination of the forces coming from the machine’s own weight (static breakout force – tipping capacity) and hydraulic breakout force exerted on the boom by the machine’s operator (Dubois et al. 2007). The choice of bucket size is based on the loose density of the material (weight/m³) to be moved. The bucket sizes range from 3 to 11.6 m³ (Bloss et al. 2011). For lighter material, a larger bucket is recommended, whilst for heavier material, a smaller bucket size is recommended (Dubois et al. 2007). Overloading is not recommended, as it results in excessive tire wear, higher operating costs and shorter working life for the machine (Dubois et al. 2007). Overall, LHDs offer:

- The possibility to install autonomous systems, such as MineGem system from Caterpillar, AutoMine system from Sandvik or Scooptram Automation system from Epiroc.
• Versatility and flexibility with a variety of different uses in mining, such as road cleaning, material haulage or loading trucks (Dubois et al. 2007).

• The bucket shape ensures optimised fill factor (shape factor of 0.92) and Z-bar geometry.

• The capability to climb steep gradients; this is effective up to 20% but is best up to 14% (Bloss et al. 2011). Tramming distances are efficient up to 200 m (Bloss et al. 2011). Haulage distance from stope to orepass is usually between 50 m and 400 m; when trucks are being loaded, the distance is usually less than 100 m (Dubois et al. 2007).

• Diesel-powered LHDs with drive motors of up to 350 kW can reach a payload of 21 tonnes (Wennmohs, 2014). The LHD’s payload is around 50% higher than for Front-End-Loaders (FELs) (Dubois et al. 2007). Diesel-powered machines travel 30 to 50% faster than electric machines (Paraszczak et al. 2013). However, newer electric machines are able to travel as fast as or even faster than diesel machines. Electric-powered LHDs reach payloads of up to 25 tonnes with drive motors of up to 350 kW for the largest series (Wennmohs, 2014). Both diesel and electric models are available on the market. Different designs offer the possibility of working in areas from small and narrow orebodies to the largest of open stopes (Dubois et al. 2007).

• The LHD canopy/cabin (depending on the type) is driver friendly (ergonomic, air-conditioning and sound insulation) and meets the following regulations: Roll Over Protection Systems (ROPS), Falling Object Protective Structures (FOPS), European Commission (EC) and Mine Safety and Health Administration (MSHA) regulations with installed fire suppression systems (Dubois et al. 2007).

2.2.1 Fleet dynamics, dispatch and scheduling

Fleet dynamics refer to the allocation of vehicles to multiple faces. The ability to flexibly and optimally allocate and reallocate vehicles enhances the potential to improve production and increases the efficiency of the vehicles, especially when onsite disturbances
interfere with the predefined plans. Therefore, dynamic dispatch systems are used to control the fleet. Dispatch (real-time management) systems help to achieve production targets and make decisions using updated information on expected future states of the system rather than making each vehicle assignment independent of such a forecast (Yingling, 2011). The dispatching systems are run according to production plans, typically short-term plans that are regularly updated and based on the production scheduling models. Scheduling is a decision-making process that deals with the allocation of resources to tasks over a given period of time (Pinedo, 2008). The goal of scheduling is to optimise one or more objectives (Pinedo, 2008).

Traditionally, to achieve production targets and handle unexpected errors, a scheduling plan has been created by the production planner based on his or her own experience and personal judgment (Song et al. 2015). However, production scheduling systems can be used to aid the production planner in decision making. Different types of routines, dispatch rules or optimisation techniques can be used in the decision-making process. New systems or changes to an existing scheduling system are often required.

To aid in the scheduling process, Song et al. (2015) proposed a schedule optimiser for mobile mining equipment that uses a set of algorithms and rules to minimise the total times required to finish certain activities. Truck-dispatching systems to maximise production and minimise material handling of the transportation in mining were studied by Song et al. (2013), White and Olson (1992) and Saayman et al. (2006). Song et al. (2013) used linear programming, Saayman et al. (2006) used different dispatching strategies simulated in MatLab environment, and White and Olson (1992) used a combination of network models, linear programming, and dynamic programming. Gamache et al. (2005) used Dijkstra’s algorithm for routing single mobile machinery. Beaulieu and Gamache (2006) used dynamic programming to solve the routing problem related to the movement of the vehicles in an underground transportation system. This allowed them to minimise the travelling time and congestion. Similarly, Yu et al. (2011) studied
the vehicle routing problem using fast local search and parallel computing of the genetic algorithm to minimise the distance, time and number of vehicles in operation. Weintraub et al. (1987) used linear programming based on a heuristic approach to solve the truck routing problem. This allowed them to minimise queueing time at loading points and resulted in about an eight percent increase in productivity (Weintraub et al. 1987).

Nehring et al. (2010) scheduled a loader-truck transportation system using Mixed Integer Linear Programming (MILP). Schulze et al. (2016) scheduled mobile machines in an underground mine using CPLEX based on formulated Mixed Integer Programming (MIP). Matamoros and Dimitrakopoulus (2016) used Stochastic Integer Programming (SIP) for shovel-truck allocation, together with a production extraction sequence. Salama et al. (2014a) used a combination of DES and MIP; DES was used to estimate the production of different haulage systems, and MIP was used to generate the optimal production schedule and mine plan. Salama et al. (2014b) used simulation to determine the expected operating costs associated with haulage options at different main haulage levels; they then used MIP to optimise production scheduling to generate an operating NPV for each option. McKenzie et al. (2008) used the shortest-path method via dynamic programming to optimise the location of the conveyor, resulting in a 14% reduction in operating costs. Nehring et al. (2012) used MIP formulation to optimise net present value. Jawed (1993) and then Winkler (1998) used LP to determine the amount of ore to be extracted. O’Sullivan and Newman (2015), Martinez and Newman (2011), Newman et al. (2005), Kuchta et al. (2004), Dagdalen et al. (2002), Kuchta (2002) and Topal (1998) studied long and short-term production scheduling in LKAB’s Kiruna mine scheduling when certain areas had to be extracted, with the goal of maximising production and improving resource allocation.

### 2.2.2 Equipment performance and production calculations

In this study, performance measures are used to determine the performance of the equipment. The equipment is selected based on
the single machine or combinations of machines that should be capable of moving a specified amount of material over a given period of time. The steps in the selection process are to first look at the costs and design of the equipment and then to determine the required production and haul paths. The next step is to calculate the cycle times, capacities (production rate = capacity x no. of cycles/unit of time) and efficiency factors (productivity = production rate x efficiency factors). To improve the productivity, the process should be further iterated. Once this is done, the fleet size is calculated and iterated to reduce owning and operating costs (Sweigard, 1992). After calculation, the user has to remember that as job conditions become more severe (deeper trench, more obstacles, etc.), the machine may slow down, affecting the cycle time (Sweigard, 1992). Other factors that should be considered are planned shutdowns, downtime losses, speed losses and quality losses. Planned shutdowns should be excluded from the efficiency analysis since they cover activities such as breaks, lunch, scheduled maintenance or periods where there is nothing to produce (Vorne Industries, 2002-2008). Downtime losses are also related to production stoppages caused by, for example, equipment failures, material shortages or changeover times. Speed loss can be related to the experience of the operator but also to machine wear or sub-standard materials (Vorne Industries, 2002-2008). Quality loss is related to pieces that do not meet standards (Vorne Industries, 2002-2008).

To determine the performance of rubber-tired equipment the following factors should be considered (Frank and Filas, 2009):

- Drawbar pull - horizontal force available at the drawbar.

\[
DBP = \frac{(EBP \times EFF \times UCF)}{Vtr}
\]

(1)

where:

\[
DBP = \text{Drawbar pull (kg)},
\]

\[
EBP = \text{Engine brake power (kW)},
\]

\[
EFF = \text{Efficiency in converting engine power to drawbar power (decimal)},
\]
UCF = Unit conversion factor, 102 (m·kg)/(kW·s),

\[ V_{tr} = \text{Tractor speed (m/s)} \]

- Traction – varies depending on the type of road surface. For example, when driving on a gravel, packed and oiled road, a 0.55 – 0.85 coefficient of traction should be used, whereas when driving on loose earth road, a 0.45 coefficient of traction should be used.

\[
\begin{align*}
\text{UDBP} &= \text{CT} \times \text{GWT} \\
\text{URP} &= \text{CT} \times \text{GWD}
\end{align*}
\]

where:

- UDBP = Usable drawbar pull (kg),
- GWT = Gross weight of tractor (kg),
- URP = Usable rim pull (kg),
- GWD = Gross weight on drive wheels (kg),
- CT = Coefficient of traction (decimal).

- Total resistance – the sum of the grade resistance, i.e., the tractive force required to overcome gravity (Sweigard, 1992), and rolling resistance, i.e., the tractive effort required to overcome by the tires in contact with the ground (Sweigard, 1992). Rolling resistance is surface related.

\[
\begin{align*}
\text{RR} &= \text{RRF} \times \text{GMW} \\
\text{GRF} &= \text{GCF} \times \%\text{Grade} \\
\text{GR} &= \text{GRF} \times \text{GMW} \\
\text{TR} &= \text{RR} \times \text{GR}
\end{align*}
\]

where:

- RR = Rolling resistance (kg),
- RRF = Rolling resistance factor (kg/t),
- GMW = Gross machine weight (t),
- GRF = Grade resistance factor (kg/t),
- GCF = Grade conversion factor = 10 (kg/t),
%Grade = (+/- vertical distance/horizontal distance) x 100,
GR = Grade resistance (kg),
TR = Total resistance (kg).

- Rim pull - maximum pulling force that the engine has to deliver to the tires.
  \[ RP = \frac{(EBP \times EFF \times UCF)}{V_{eq}} \] (8)

where:
- \( RP \) = Rim pull (kg),
- \( EBP \) = Engine brake power (kW),
- \( EFF \) = Efficiency in converting engine power to drawbar power (decimal),
- \( UCF \) = Unit conversion factor, 367 (km-kg)/(kW-h),
- \( V_{tr} \) = Equipment speed (km/h).

Rim pull and speed can be also tabulated as shown in Figure 3 in steps 1-4.

![Figure 3. Rim pull speed curve for LHD (Courtesy of Caterpillar).](image)
• Maximum and average speed – calculation of the average speed requires incorporation of acceleration, deceleration, shifting, and braking, as well as steep curves, grades, and congested areas (Frank and Filas, 2009).
• Altitude correction – loss of engine power occurs at higher altitudes. The use of derating tables is recommended (Frank and Filas, 2009).

To determine production, loading, hauling and dumping, the following factors should be considered (Frank and Filas, 2009):

• Density and swell;
• Operating efficiency (availability and utilisation);
• Bucket fill factor;
• Load cycle time;
• Loading production;
• Travel time;
• Haulage cycle time;
• Haulage production.

2.2.3 Fuel and power consumption

Improving vehicle usage by reducing the fuel and power consumption will generate energy savings and will often lower the operating costs (EEO - Transport, 2010). The efficiency of the vehicle and its fuel consumption and energy use will depend on factors such as the following (EEO - Mining, 2010):

• Design of the vehicle;
• Driver loading, hauling and dumping practices – aggressive driving can raise fuel consumption by 30%, poor tuning by between 4% and 40%, suboptimal tire inflation by 3% and clogged air filters by 3% - 10% (EEO - Transport, 2010);
• Scheduling practices;
• Gradient and surface features of the road on which the vehicle travels;
• Size and shape of the material being transported.

Various models can be used to calculate the energy consumption of the system, including Energy Savings Measurement Guide,
Energy Efficiency Opportunities, Energy Mass Balance, or Mining Industry Energy Bandwidth (Bise, 2003). These models are not as accurate as on-site measurements but they are less expensive (Tatiya, 2013). The Energy Mass Balance (EMB) model is used to determine the efficiencies of processes and equipment (EEO – Transport, 2010). The EMB shows how the energy flows and influences the parts of the system being studied and helps to identify how much energy is being used or lost. The development of a mining EMB consists of five steps:

- Step 1 - Develop an EMB plan;
- Step 2 - Collect the data for the EMB;
- Step 3 - Analyse the EMB model and identify potential energy efficiency opportunities;
- Step 5 - Identify the next versions of the EMB model and repeat the process until the EMB is sufficiently detailed and accurate.

The examination of influencing factors should include external factors (conditions at mine site, regulations and standards, market demand, market value of resources), business factors (demand and energy using equipment, business operations, employee attitudes and behaviour, operating procedures) and site-specific factors (mine design and characteristics, selection of energy using equipment and installation, operation of equipment, rate of resource extraction) (EEO – Mining, 2010).

On-site measurements are the most accurate way to determine fuel and power consumption, but they are very expensive because continuous monitoring is required. In cases where there are not enough data from the field measurements or field measurements are not possible, the fuel consumption can be estimated based on the following equation (Frank and Filas, 2009):

\[ C_f = EBP \times F_l \times UCF \]  

(9)

where:

\[ C_f \] = Fuel consumption (L/h),
\[ EBP \] = Engine brake power (kW),
$F_{L} = \text{Engine load factor (decimal)},$

$UCF = \text{Unit conversion factor 0.3 (L/kWh)}.$

It can also be estimated using the equation by Kecojevic and Komljenovic (2010):

$$F_c = \frac{(K \times \text{GHP} \times \text{LF})}{\text{KPL}}$$  \hspace{1cm} (10)

where:

$F_c = \text{Fuel consumption (L/h)},$

$K = \text{kg of fuel used per brake horsepower per hour},$

$\text{GHP} = \text{Gross engine horse power (kWh)},$

$\text{KPL} = \text{Weight of fuel (kg/L)},$

$\text{LF} = \text{Load factor (%)}.$

Standard engineering calculations of the vehicle energy model can be written as (EEO - Transport, 2010):

$$E = E_{\text{idling}} + \frac{1}{\eta_u} (E_{\text{aero}} + E_{\text{roll}} + E_{\text{accel}} + E_{\text{climb}} + E_{\text{brake}} + E_{\text{ancill}})$$  \hspace{1cm} (11)

where:

$E = \text{Total energy used by the vehicle},$

$E_{\text{idling}} = \text{Energy used by idling},$

$\eta_u = \text{Energy efficiency of the engine and transmission},$

$E_{\text{aero}} = \text{Aerodynamic resistance},$

$E_{\text{roll}} = \text{Rolling resistance},$

$E_{\text{accel}} = \text{Energy required for acceleration (overcoming inertia)},$

$E_{\text{climb}} = \text{Energy required for climbing},$

$E_{\text{brake}} = \text{Energy dissipated in braking},$

$E_{\text{ancill}} = \text{Energy consumed by ancillaries (e.g. alternator, air compressor)}.$

To convert the fuel consumed (FC) into kilowatt hours (kWh), the following equation can be used (Packer, 2011):
\[ E_D = \frac{(FC \times A)}{B} \]  \hspace{1cm} (12)

where:

- \( E_D \) – Diesel energy consumption (kWh),
- \( A = 38.6 \text{ MJ/L} \) – Amount of heat released by the combusted fuel (Engineering Conversion Factors),
- \( B = 3.6 \text{ MJ} \) – Heat used to produce 1 kWh (Engineering Conversion Factors).

The energy consumption can be derived from the Energy Mass Balance model, estimated using the loading cycles (from the loading point to the dumping point and back to the loading point):

\[ E = \frac{(TR \times g \times (V_W + B_C) \times V_L + (V_W \times V_E))}{1000 \times t} \]  \hspace{1cm} (13)

where:

- \( E \) – Energy consumption (kWh),
- \( TR \) – Total resistance (t),
- \( g \) – Acceleration due to gravity (m/s\(^2\)),
- \( V_W \) – Gross vehicle weight (t),
- \( B_C \) – Bucket capacity (t),
- \( V_L \) – Vehicle speed when loaded (m/s),
- \( V_E \) – Vehicle speed when empty (m/s),
- \( t \) – Loading cycle time (h).

The power to overcome the rolling resistance is proportional to vehicle speed, but the energy used is not affected by vehicle speed, as \( \text{Energy} = \text{Power} \times \text{Time} \), and time for a trip is inversely proportional to average speed (EEO - Transport, 2010). The power to overcome aerodynamic resistance is proportional to the cube of velocity, and the aerodynamic drag is proportional to the square of the vehicle speed (EEO - Transport, 2010).
The ventilation system is necessary to remove the noxious gases after the blast, stabilise the temperature and provide fresh air. When an underground mine reaches greater depths, the cost of ventilation increases. With an increasing depth, the air has to travel longer distances, requiring larger and more powerful ventilation systems to pump the air down to the production areas. According to De la Vergne, the ventilation in underground mines accounts for one-third of the electrical power costs (De la Vergne, 2003), whilst according to Paraszczak et al. (2013), it reach approximately 40% of the electrical power costs. According to Brake (2013), the ventilation in typical underground hard rock mines when trucking up to surface consumes up to 80% of electrical power. In this study, the focus is on the ventilation costs related to the LHD equipment. A recent study comparing a diesel fleet with a battery-driven electric fleet by Fox et al. (2018) shows a potential reduction in operating costs related to ventilation of up to 58.3% and a reduction in heat of up to 53.4% (Allen, 2016).

In the absence of on-site measurements, the loader component of the ventilation required can be calculated based on the following equation derived from Jacobs (2013) after Tuck (2011):

\[
C = \frac{(P \times OT \times EC)}{NoM} \tag{14}
\]

where:

\begin{align*}
C &= \text{Ventilation costs per vehicle ($/day/unit)}, \\
P &= RQ^3 \text{ (kW)}, \\
R &= \text{Mine resistance (Ns}^2\text{m}^2), \\
Q &= VR \times NoM \times M.
\end{align*}

where:

\begin{align*}
Q &= \text{Flow rate (m}^3\text{/s)}, \\
VR &= \text{Ventilation requirements (m}^3\text{/s)}, \\
NoM &= \text{Number of machines}, \\
M &= \text{Motor power (kW)},
\end{align*}
OT = Operating time (hours/day),
EC = Electricity costs ($/kWh).

When many diesel machines are in operation, high levels of diesel exhaust are generated (Pronk et al. 2009). If the working areas are not ventilated properly, workers will be at risk of being affected by noxious gases and carcinogenic agents (Pronk et al. 2009). There is a need to provide oxygen to personnel and provide oxygen for combustion, to dilute fumes from the diesel engines, diesel particle matter (DPM) and blasting, to regulate heat and improve visibility (Tuck, 2011). The ventilation requirements (diesel exhaust and noxious gasses) in Sweden specify levels of nitrogen dioxide (NO₂) and carbon monoxide (CO) (Roiste et al. 2013). Carbon dioxide (CO₂) is used to indicate diesel exhaust exposure (Arbetsmiljöverket, 2010). CO₂ emissions from diesel fuels can be calculated based on the diesel conversion factors published by the United States Environmental Protection Agency (2005):

\[
\text{CO}_2 = FC \times CC \times 10^{-6} \times 0.99 \times [44/12] \quad (15)
\]

where:

CC – Carbon content for the diesel fuel (g/L),
0.99 – Oxidation factor (kg/kW/h),
44/12 – Ratio of the molecular weight of CO₂ to the molecular weight of carbon.

2.3 Discrete event simulation

Simulation is the artificial representation of a real-world system or process over specified period of time (Banks et al. 2010). Simulation models are used to study the behaviour of the system, represented by mathematical equations and symbolic notations (mathematical model). The model can be a scaled down or enlarged version of the studied system (physical model). The combined use of simulation and analytical calculations enables a more accurate estimate of the parameters of concern. In cases where randomness is involved, mathematical/numerical analyses are difficult to perform and simulation can be used.
Discrete Event Simulation (DES) is recognised as a suitable method for analysing dynamic complex systems, such as a mining operation. DES is defined by Banks (1998) as a simulation technique where variables change at discrete points at which the events occur. Examples of events are the arrival, departure or breakdown of the machine. The methods by which those events are controlled in the simulation are divided into five categories (Derrick et al. 1989):

- Event scheduling - advance is based on events;
- Activity scanning/two-phase approach – advance/scan is based on objects performing activities;
- Three-phase-approach - advance is based on activities bound to occur or conditionally stored variables;
- Process interaction - advance is based on emulation the flow of an object;
- Transaction flow - advance is based on time and state relationship as in process interaction, but the movement and generation of objects differ.

Discrete event simulation models are not always used to model discrete systems, and continuous simulation models are not always used to model continuous systems. For example, conveyor belts can be modelled discretely, even though the belt is continuously moving. The choice of the type of simulation model to use depends on the characteristics of the system and the objective of the study (Banks et al. 2013).

The user has various options to build a simulation model. The first option is to write the code from scratch in a general-purpose programming language. The second is to use simulation oriented software. Some simulation software enables the user to write the code from the scratch in an existing simulation language. Today, there are numerous simulation software packages exist, the user has a wide palette of software to choose from. Writing the simulation models in general languages is time-consuming and requires a considerable amount of programming knowledge. Therefore, writing the program code with in-built simulation language or using simulation oriented software is recommended.
Writing the program code using one of the numerous simulation languages takes less time than writing the model in a general purpose language; the user can select one of the built-in pre-programmed tools, thus permitting more focus on the task (interactions between the vehicles, output or input values) without needing to build the structure that will handle the functions, environment, etc. Depending on the choice of simulation software, the model development time can be greatly reduced. It is advisable to involve the customer in model conceptualisation. Involving the customer will enhance the quality of the resulting model and increase confidence in the model’s application. Using existing oriented software is the easiest option, but software packages often have limitations related to, for example, the input requirements, or constraints related to their capabilities. For instance, a simulation model developed specifically for a surface mine may not incorporate the functionality required for an underground transportation system. The simulation oriented software is limited in cases where extra functionality is required to, for example, implement functions that are not in the package.

To develop a comprehensive simulation that will adequately model the studied system and generate a credible output, a series of simulation cycle steps should be employed (Figure 4). Each step helps to assure that the simulation model is created to the appropriate level of detail. The level of detail in the model has to be appropriately balanced; otherwise, the time and cost of generating the model could significantly increase. Developing an over-detailed simulation model does not improve the answer to formulated problems. In fact, this type of model is likely to generate insufficient results because there is an insufficient amount of input data; this, in turn, may obscure the readability of the model, making it hard to follow and validate. However, if designed and studied appropriately, a simulation can provide a detailed analysis of the performance of the system (Banks et al. 2013)
During the experimental design, decisions need to be made on the length of the initialisation period, the length of simulation runs, and the number of replications (Banks et al. 2013 after Sargent, 2011). Documentation and reporting are always necessary. If the program is going to be used again by the same or a different analyst, it could be necessary to understand how it operates. The results of all analyses should be reported clearly and concisely in a final report. The success of the implementation phase depends on how well the previous steps have been performed.
### 2.3.1 Discrete event simulation for studying LHD operations

Discrete event simulation (DES) is a suitable method for analysing dynamic (the system changes over time) and complex rock transportation systems in the mining industry. Recent research on LHD operations using DES can be divided into the following categories:

- **Selecting equipment** – Skawina et al. (2015a-c, 2016) performed a simulation to select the LHD fleet for an underground mining environment. Salama (2014b) simulated different underground environments to evaluate and analyse different haulage systems, including diesel and electric loaders and trucks, shafts and belt conveyors. Yuriy and Runciman (2013) presented and discussed Vale's Canada experience with WITNESS, Quest, SimMine, ProModel and AutoMod simulation tools from the point of view of their functionality, capability and user-friendliness towards underground mining operations;

- **Studying the effects of disturbances** – Skawina et al. (2016) studied the effects of ore pass disturbances on loading operations;

- **Scheduling, sequencing and optimising processes** - Puhakka and Kainulainen (2000) used the OPTIMINE simulation tool to maximise performance and optimise a mobile fleet, schedule work and fine-tune layout. Salama et al. (2014b) used MIP to optimise production scheduling to generate an operating NPV for each option. Usmani et al. (2014) used the ARENA simulation tool to determine the optimal number of LHDs required to meet the production target of a block cave panel operations. Dugardin et al. (2007) used a simulation-based optimisation technique to optimise the machines in a workshop. Rippenhagen and Krishnaswamy (1998) discussed the application of simulation-based scheduling using a real-time dispatch tool. Potradi et al. (2002) used DES to generate schedules for the machines and a plan for the start date of material extraction;
• Assessing, managing and evaluating the studied system - Bailey et al. (2012) used Arena software to model the first part of the ore handling system (face haulage) at Olympic Dam. Gregg (1986) used the GPSS/H simulation tool to develop a model to compare the productivity of diesel and electric face-haulage vehicles (Novak et al. 1987). Similar average shift production was achieved by both types of machines, but with increased haul distances, diesel vehicles out-produced the electric vehicles. A combination of an AutoMod simulation model and a predictive reliability assessment model was used to study and analyse a mine’s equipment system by Yuriy and Vayenas (2008);

• Testing alternative approaches/scenarios/input generation - Pereira and Mech (2010) compared the scale-up issues and constraints of autonomous vehicles and a manual system using ExtendSim software. Hunt (1994) used SLAMSYSTEM software to evaluate what-if scenarios for the ore transport system in Henderson mine. The project broadened and expanded the mine’s procedures by using simulation. Turner (1999) used the ARENA simulation tool in Finsch diamond mine to test a new concept called Quadrant loading. The results were positive: the LHDs’ production rate had the potential to increase by 16%, and the tramming distance could be reduced by 38%.

2.3.2 Goals and performance measures

The typical expectation when using a simulation model is to obtain the performance measures in form of numeric values, such as the production rate under a given set of conditions. Common performance measures are throughput, cycle time, utilisation, bottlenecks, queuing and delays, storage and personnel requirements. Another is effectiveness of the studied system (Banks et al. 2013). The performance measures, together with the visualisation through animation and a graphics interface, contribute to the model’s credibility and acceptance of the model’s numeric output values.

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2.3.3 Modelling downtimes and failures

It is often necessary to include downtimes and failures in the model to achieve accurate and valid simulation models. It is also necessary to decide whether to use actual system data or a statistical model for different input values. Downtimes can be unscheduled and random or scheduled, for example, for preventive maintenance. Failures refer to machine breakdowns and scheduled downtimes refer to preventive maintenance.

There are a number of possible approaches to modelling random unscheduled downtimes (Banks et al. 2013):

- Alternative 1: Ignore downtimes;
- Alternative 2: Model the downtimes by increasing the times using the appropriate proportion;
- Alternative 3: Use constant values;
- Alternative 4: Use statistical distributions.

For cases when the production or plant is down for a long period due to rare or catastrophic events, such as general power failures, snow storms, cyclones and hurricanes, the user can justify ignoring the downtimes (Alternative 1). Adjusting the processing time to each job is a possibility if the pure processing time is almost never accurate; this type of deterministic adjustment might provide reasonably accurate estimates of overall system throughput (Alternative 2).

Constant duration of time to failure and time to repair could be used in cases when the disturbance occurs on a fixed schedule, such as preventive maintenance (Alternative 3). In all other cases, statistical distributions are recommended (Alternative 4).

Time to failure can be measured by a clock, by machines or equipment, by the number of cycle times or by the number of items produced (Banks et al. 2013). Breakdowns or failures can be based on clock time, actual usage or cycles (Banks et al. 2013). Time to repair can be modelled as a pure time delay or as a wait time for a resource together with the time to repair (Banks et al. 2013). If the input variables are not based on the constant variables, the goal is to determine the probability distribution for the performance
variables. Since simulation models do not provide direct information on which distribution to use, the user has to use multiple or extended runs and statistical inference (process in which the properties of the probability distributions are deduced) to describe the performance variable/s and then select the appropriate distribution for an input value (Yingling, 2011).

2.3.4 Verification and validation

Verification and validation tests must be performed to ensure the correctness and credibility of the model. Verification tests are responsible for ensuring that the conceptual model is accurately transformed and translated into a computer model. Validation tests ensure that the simulation model accurately represents the studied system in problem formulation and objectives (Kleijnen, 1995; Balci, 1998; Banks et al., 2010).

Verification and validation tests aim at increasing confidence in a model’s credibility and decreasing the possibility of errors. Thus, they are important parts of simulation model development (Sargent, 2011; Kleijnen, 1995). It is also important to mention that simulation models are created to fulfil certain predefined objectives and used to fulfil predefined conditions. Thus, any changes to those objectives will require additional verification and validation tests. In some cases, models have to be modified and changed, requiring additional work.

To model confidence, Balci (1998) and Sargent (2011) proposed a credibility model (Figure 5). A point is reached at which the costs and time become very high, and it is not necessary to further increase the value of the model to user since it does not result in any significant improvement.
Verification and validation techniques that test and analyse the model can be informal, static, dynamic or formal (Table 2).

**Table 2. Examples of verification and validation tests modified after (Balci, 1998).**

<table>
<thead>
<tr>
<th>Informal:</th>
<th>Static:</th>
<th>Dynamic:</th>
<th>Formal:</th>
</tr>
</thead>
<tbody>
<tr>
<td>- Inspections</td>
<td>- Cause-effect graphing</td>
<td>- Debugging</td>
<td>- Mathematical calculus</td>
</tr>
<tr>
<td>- Reviews</td>
<td>- Data analysis</td>
<td>- Execution testing</td>
<td>- Logical deductions</td>
</tr>
<tr>
<td>- Document checks</td>
<td>- Control analysis</td>
<td>- Acceptance analysis</td>
<td>- Predictive calculus</td>
</tr>
<tr>
<td>- Audits</td>
<td>- Fault analysis</td>
<td>- Comparison testing</td>
<td>- Proof of correctness tests</td>
</tr>
<tr>
<td>- Face validation</td>
<td>- Interface analysis</td>
<td></td>
<td>- Compliance testing</td>
</tr>
<tr>
<td></td>
<td>- Semantic analysis</td>
<td></td>
<td>- Object flow testing</td>
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<td></td>
<td>- Symbolic analysis</td>
<td></td>
<td>- Interface testing</td>
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<tr>
<td></td>
<td>- Syntax analysis</td>
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<td></td>
<td>- Traceability analysis</td>
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</tr>
<tr>
<td></td>
<td>- Structural analysis</td>
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</tr>
</tbody>
</table>

**Figure 5. Credibility model (after Sargent, 2011).**
In many cases, there is insufficient information, and a subjective judgment about model validity must be made based on expert opinion. The final decision on whether the simulation model is valid is usually made by an internal group. An additional measure is to involve a third party group to independently test and validate the model.

2.3.4 Scheduling and simulation

Altogether, there are three different ways of using simulation with connection to a scheduling system. The first is to use simulation as a scheduler (Fowler et al. 2006). Potradi et al. (2002) used DES to generate schedules for the machines. Dugardin et al. (2007) used an ARENA simulation model and the optimisation algorithm to optimise machines in a workshop. Sivakumar (2001) used DES based on the dynamic scheduling system to test and optimise manufacturing environment. Rippenhagen and Krishnaswamy (1998) discussed the application of simulation-based scheduling using a real-time dispatch tool.

The second option is to use the simulation model to generate the input parameters for the scheduling system and test the way the instances are generated (Fowler et al. 2006). Biskup and Feldmann (2001) gave an example in which simulation was used to determine objective function values.

The third option is to use simulation for the emulation and evaluation of scheduling approaches (Fowler et al. 2006). Horiguchi et al. (2001) and Pritsker et al. (1997) examined the performance of the algorithm using the AweSim DES language. Mönch et al. (2003) evaluated different scheduling approaches for the manufacturing system using simulation-based environment.
CHAPTER 3: RESEARCH METHODOLOGY

The aim of the chapter is to provide the reader with a clear picture of the study process and the components of the system under study and to provide practical information related to the study. The chapter clarifies how the study was performed.

3.1 Literature review

This research started with an extensive review of the literature on various topics related to underground transportation systems, scheduling systems in mining and discrete event simulation. The literature review included primary and secondary sources. The primary sources were books, journals, research and technical reports; the secondary sources were conference proceedings, correspondence, Master’s theses, doctoral theses and Web documents. The review was split into a wide range of subject areas and covered the following topics:

- **Underground transportation systems**
  - Challenges related to underground transportation systems
  - LHDs
  - Power and fuel consumption
  - Gas emissions and ventilation
  - Loading and hauling practices
  - Trucks
  - Ore pass structures

- **Scheduling in mining**
  - Production scheduling
  - Simulation and scheduling
  - Decision making in production scheduling
  - Rescheduling, strategies, policies and methods

- **Discrete event simulation**
  - Selection of DES software
  - Modelling and analysis of complex dynamic systems
  - Simulation in mining
- Statistical analysis in simulation
- Verification and validation tests

Underground transportation systems were studied in detail, with the inclusion of trucking and ore passes, as loading is connected to these unit operations. Scheduling was included, as it strongly depends on the performance of the loaders. Discrete event simulation was studied in order to identify a suitable tool for analysis.

3.2 Selection of DES

DES is recognised as a suitable method for analysing rock transportation systems in the mining industry. The rock transportation system is a dynamic complex system consisting of components, such as trucks, loaders, ore passes, crushers, etc. Depending on the objectives, the interactions between the components of the system can be complex and hard to understand outside the system. However, simulations can help to explain these interactions. A simulation provides insight into operations without affecting the system. Simulation models allow various alternative scenarios to be tested in a dynamic environment. This helps to answer what-if questions beforehand and to decide whether to invest more time in further experimental analysis or to directly integrate work into the actual system.

The simulation models applied in this study were based on real systems, using operational data and information collected from the studied sites. The simulation models provided solutions to LHD operations with respect to:

- Selecting and optimising the LHD machines;
- Testing automatic scheduling systems with the use of DES;
- Assessing, managing and evaluating the studied systems;
- Testing alternative approaches/scenarios/input generation;
- Studying the effects of disturbances in mine operations such as an ore pass loss.
3.2.1 Simulation software selection

Initially, in this study, several different tools were evaluated based on the purpose, the modelling environment, model documentation and structure, verification and validation support, experimental facilities, statistical facilities, user support and financial and technical features. Many tools seemed appropriate. Therefore, the selection process was narrowed down to options that allow internal modification of the software and for which external support is available at a relatively low cost. In the end, two tools were selected: SimMine and AutoMod. These tools were used to develop the simulation models and test various scenarios. The results showed that they can model rock transportation systems with the necessary level of detail and satisfy the characteristics necessary for the study.

AutoMod is a manufacturing-oriented software with two graphic modes: static and dynamic. The modifications of the parameters can be made directly in the syntax (.m logic) or in a built-in environment (Banks, 2004; Yuriy and Runciman, 2013). Movement systems are modelled in specially designed built-in templates: vehicle path mover systems, conveyors, automated storage and retrieval systems, bridge cranes, power and free conveyors and kinematic (robotic) systems. Standard reports are available, but users can make their own custom reports. Advanced debugging and trace facilities, together with 3-D animation, enable easy tracking of errors and flaws and help to verify and validate the models (Banks et al. 2010). Add-ons to AutoMod are AutoView, AutoStat, a model zip archiver, OPC utility, a process server, and Web tools’ utilities, such as SimController or ACE graphics editor (Banks, 2004). SimController is used to make multiple runs, compare reports, view Gantt charts and view model information. ACE graphics are used to create, edit, delete or modify the graphic elements animated and used in the AutoMod model. AutoStat is used for the purpose of designing experiments by running multiple simulations under different statistical constraints, performing sensitivity analysis or optimising the model (Banks et al. 2010). AutoView is used to create video clips from the AutoMod models.
SimMine is mine-oriented software with built-in structures that do not require the user to create code. This tool has an interactive 3D layout and is developed specifically for the simulation of mining operations. In built-in interfaces, the user can set and specify underground mine related properties, such as rock properties, face profiles, activity cycles, etc. Equipment interface and delays can be visually analysed for better evaluation of the statistical results generated by the model (Yuriy and Runciman, 2013). The advantage of this software is that the user does not need to spend a lot of time coding. Its simplicity allows it to be quickly used, even in the mine, and many parameters are already implemented in it.

3.3 Description of the studied cases

The studied cases are based on the operations in three mine sites, two owned by LKAB (Kiirunavaara mine, Malmberget mine) and one owned by New Boliden (Kristineberg mine). All are located in the northern part of Sweden. The mining companies participated in the projects, providing data and information. Several study visits and interviews were conducted to understand the mining operations and to collect necessary information and data from the mine sites. The data collection process was lengthy and required several mine visits to feed the simulation models to the appropriate level of detail. There were often different sources for the same data, and this enabled comparison and triangulation. Interviews took place at Kiirunavaara, Malmberget and Kristineberg mines and included a wide spectrum of topics; only part of the information was necessary for these particular studies. During the mine visits, each unit operation was physically observed and discussed with the operators. Internal reports and documentation of the mining operations were obtained during the mine visits, as well as data from the mining operations and data from equipment manufacturers. Time studies were performed to ensure the appropriate level of detail. Alternative approaches/scenarios/input generation were tested, and models were verified and validated during and after model development. This helped develop simulation models which would represent the behaviour of the
studied system to the level of detail necessary to answer the formulated problems.

In these studies, the problems were related to selecting tools to develop simulation models, selecting the equipment to study (electric versus diesel LHDs), studying the effects of ore pass unavailability on loading operations and testing an automatic scheduling system using DES. The results from the simulation runs were used as input data for further statistical and analytical analyses. For a summary of the studied cases, see Table 3.

Table 3. Summary table of the studied cases.

<table>
<thead>
<tr>
<th>Studied case</th>
<th>Mine sites</th>
<th>Tools</th>
<th>No. of simulation models</th>
<th>Paper</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tool selection</td>
<td>Kiirunavaara mine</td>
<td>AutoMod and SimMine</td>
<td>2</td>
<td>Paper A</td>
</tr>
<tr>
<td>Equipment selection (electric vs diesel)</td>
<td>Kiirunavaara mine</td>
<td>AutoMod and SimMine</td>
<td>2 (the same model as in tool selection)</td>
<td>Paper B</td>
</tr>
<tr>
<td>LHD operations affected by a disturbance in the mining system</td>
<td>Malmberget mine</td>
<td></td>
<td>3</td>
<td>Paper C-D</td>
</tr>
<tr>
<td>Integrating DES with a scheduling system</td>
<td>Kristineberg mine</td>
<td>SimMine</td>
<td>1</td>
<td>Paper E</td>
</tr>
</tbody>
</table>

3.4 CASES 1 & 2: Tool and Equipment selection

Depending on the problem and objectives of a study, the simulation tools must fulfill a certain range of requirements covering areas related to processing characteristics, tool capabilities or cost related factors. Many software tools already fulfil most of these requirements. However, the level to which the software fulfils the given requirement often varies. Some software has a very comprehensive structure and extra features; other software is less complex whilst still fulfilling the design requirements. Therefore, the first part of the study focuses on the software tool selection process (Paper A).

With increased mining depth, energy consumption, gas emissions and ventilation requirements will also increase, establishing new
requirements for mining equipment. Therefore, the next step is to study alternative machines, such as electric powered LHD machines (Paper B).

3.4.1 Problem studied

SimMine and AutoMod models were evaluated for their appropriateness for the study based on real mining operations. The two simulation models were used to represent operations in one of the production areas in Kiirunavaara mine. The results of the two models were compared with respect to software capabilities and factors such as simulation performance, documentation and structure, statistical facilities and user support.

For case number 1 (Paper A), the focus was on selecting the appropriate tools for analysis. For case number 2 (Paper B), the focus was on the analysis of the energy consumption and gas emissions for different LHD equipment in Kiirunavaara mine using DES.

3.4.2 Mine site: Kiirunavaara mine

LKAB’s Kiirunavaara mine, located in northern Sweden, has been in operation since 1900 (Figure 6). The mine produces around 27 Mtonnes of iron every year, with the deepest production level located 1365 m below the surface. The majority of LKAB’s deliveries to the steel industry consist of iron ore pellets, with an average iron content of 66 percent. The deposit is situated in a zone where the ore body is around 80 m wide, 4 km long and 2 km deep. Mineral reserves are estimated to be 620 Mtonnes (proven) and 94 Mtonnes (probable), and mineral resources are estimated to be 2 Mtonnes (measures), 159 Mtonnes (indicated) and 37 Mtonnes (inferred). The values are based on LKAB’s Annual and Sustainability Report from 2016.
The mine uses sublevel caving as a method of extraction. At present, there are two main haulage levels, one at 1045 m and the other at 1365 m. The mine is divided into blocks with each block consisting of several sublevels in which loading and hauling operations take place (Figure 7). After production drilling and blasting, the ore is allowed to fall down into underlying production tunnels, known as drifts. The ore is loaded at the draw point by 25-tonne electric LHDs with 10.7 m$^3$ bucket size and semi-automated 21-tonne diesel LHDs with 9 m$^3$ bucket size. From the draw point, the ore is hauled to a nearby ore pass location where it is allowed to fall down under gravity. From the bottom of the ore passes that are usually located at one of the two main haulage levels, the ore is discharged via chutes onto a train. The train transports the ore to the crushers from where it is hoisted through a series of vertical shafts to the surface. Since the construction of the drifts in the production areas represents a major cost, one of the objectives in the mine is to keep the vertical and horizontal distances between the drifts as large as possible (LKAB, Annual Report, 2016). However, these distances are often constrained by the drilling and blasting techniques.
Figure 7. One of the haulage levels in Kiirunavaara mine (Courtesy of LKAB).

The layout of the production area used in the study is shown in Figure 8. Here, LHDs are used to load and haul the iron ore from the production face to the ore pass. The number of ore passes in each production area varies from two to five.

Figure 8. Production area used in the study.
Once mining has begun in a block, continuous production should be maintained until all available ore is excavated. The mining sequence used by LKAB for their sublevel caving production areas follows a V-shape pattern (Figure 9).

**Figure 9. Mining sequence (Courtesy of LKAB).**

An alternative layout (Figure 10) of the sequence was recently proposed by Quinteiro (2018). This type of design has been adjusted to cope with high horizontal stresses and higher production capacities than the existing layout (Quinteiro, 2018).

**Figure 10. Proposed fork layout for LKAB underground mines (modified after Quinteiro, 2018).**
3.4.3 Data collection

Data and information collected for Kiirunavaara mine are summarised in Table 4. Data collection began in January 2013 and ended in September 2016. There were several mine visits and interviews at the mine site. Internal reports and documentation of the mining operations were obtained during the mine visits, as well as data from mining operations and data from equipment manufacturers.

Table 4. Summary table

<table>
<thead>
<tr>
<th>Simulation models</th>
<th>Data collected</th>
<th>Data sources</th>
<th>Amount of data</th>
<th>Collection period</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Mine description and layouts</strong></td>
<td>Interviews and reports</td>
<td>Documents and layouts of the production area</td>
<td>From: Jan 13’ To: Sept 16’</td>
<td></td>
</tr>
<tr>
<td><strong>Production data</strong></td>
<td>Expert opinions and internal documentation</td>
<td>Mining level locations</td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>LHD performance data</strong></td>
<td>Video recordings (loading)</td>
<td>Several cycles</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>WOLIS* (cycles, bucket weights, locations, and equipment used)</td>
<td>85458 cycles from two production areas for 13’ and 14’ year</td>
<td>15 April and 17 Nov 14’</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Expert opinions and interviews (loading, dumping, turning, availability, utilisation)</td>
<td>Point estimate</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

*WOLIS - Wireless Loader Information System (WOLIS) data gathered from Kiirunavaara mine

3.4.4 Simulation models

The LHD operations used in the simulation models are shown in Figure 11a-b. The simulation was run using the information and
the input data obtained from the mines. The output results of the simulations included time to mine out the production area, energy and fuel costs and consumption, and gas emissions.

Figure 11a-b. TOP: AutoMod simulation model. BOTTOM: SimMine simulation model.
3.4.4.1. Model settings

Simulation (Figure 12) starts according to the shift schedule sending the LHD/s to work locations to load material from the draw points. In all cases, machines are ordered to travel to one of the production drifts. Once loaded, the LHD is assigned to the closest ore pass, but if this ore pass is not available, the machine travels to the closest of the remaining ore passes. If no ore pass is available, the LHD waits until one becomes available. Once the material is dumped into the ore pass, a check is performed on whether there is more work, and the work status is updated. Note that the ore pass is not available at this point because of rock breaking operations.

Figure 12. Flowchart of the LHD operations used in the study.
3.4.4.2 AutoMod cycle settings

The source code (.m syntax) shown in Figure 13 is used to emulate the movement cycle of the LHD machines in the AutoMod environment. The code emulates the cycle procedure by first setting the function that finds the next work location to which the LHD machine should travel. Next, the operational parameters are set. The operational parameters are the bucket capacities, acceleration, deceleration, turning time and empty/loaded vehicle travel speeds. The LHD machines are controlled by a set of instructions which update the machine and production area information. The cycles repeat until all the material has been extracted. At the end of the simulation, the results are presented in the default report and also sent to an excel file (user-customised).

```plaintext
begin the P_vehicle arriving procedure
  call F_FindNextWorkLoc(VvehPtr(Ai_ID))
  call F_SetOperParam(VvehPtr(Ai_ID))
  dispatch VvehPtr(Ai_ID) to A_dest
  while A_Done = 0 begin
    if V_locData(A_Drift, A_PointIndex) > 0 begin
      set VvehPtr(Ai_ID).A_destIndex = A_destIndex
      dispatch VvehPtr(Ai_ID) to A_WorkLoc
      wait to be ordered on OL_vehicle(procindex)
      wait for u Vload_mean, Vload_STD sec
      if V_locData(A_Drift, A_PointIndex) < VvehPtr(Ai_ID).A_Capacity then set VvehPtr(Ai_ID)
      A_current_amount to V_locData(A_Drift, A_PointIndex)
      dec V_locData(A_Drift, A_PointIndex) by VvehPtr(Ai_ID).A_current_amount
      if VvehPtr(Ai_ID) defined velocity <> 0 then
        call F_SetSpeed(VvehPtr(Ai_ID), Vr_Machine_speed(Ai_ID, Loaded), Vr_Machine_acc(Ai_ID, Loaded))
        dispatch VvehPtr(Ai_ID) to A_dest
        wait to be ordered on OL_vehicle(procindex)
        if V_crecpass = 2 then begin
          wait for u Vdump_mean, Vdump_STD sec
        end
        else
          call F_boulder(V_boulder(A_destIndex), VvehPtr(Ai_ID).A_destIndex)
          wait for u Vdump_mean, Vdump_STD sec
      end
      inc V_received(A_destIndex) by VvehPtr(Ai_ID).A_current_amount
      increment C_tonnage by VvehPtr(Ai_ID).A_current_amount
      inc C_cycles by 1
      call F_SetSpeed(VvehPtr(Ai_ID), Vr_Machine_speed(Ai_ID, Empty), Vr_Machine_acc(Ai_ID, Empty))
    end
  end
  call F_FindNextWorkLoc(VvehPtr(Ai_ID))
  call F_update(VvehPtr(procindex))
end
```

Figure 13. AutoMod cycle settings.
3.4.4.3 SimMine cycle settings

In SimMine, point-and-click menus are used to emulate the loading and transportation of the LHD machines. The software is mining-related, and no coding is required. Additional properties can be changed and adjusted depending on the user’s needs. The ‘Loading and transport activity’ tab is used to control the movement of the LHD machines. The fleet properties and boulder frequency can be changed in the ‘Fleet’ tab submenu. At the end of the simulation, the results are presented under several tabs; each has information related to fleet and location of the studied area.

3.5 CASE 3: LHD operations affected by a disturbance in the mining system

Disturbances in the mining system negatively affect the LHD operation and thus the production. An example of a disturbance is the failure of the closest unit structure to the LHD operation, such as an ore pass loss, leading to the ore pass’s unavailability. If an ore pass becomes unavailable, changes to the current loading strategy might be required. Loss of ore pass was identified as one of the most critical risks for underground mine operations (Greberg and Salama, 2016). However, because of the dynamic environment of mining operations, the appropriate changes are often difficult to make and can lead to incorrect management of the process and the equipment, especially if the current infrastructure does not support the flexibility to reallocate LHD machines to the right locations. Therefore, the next step was to study how the LHD operations are affected when one or more ore passes become unavailable (Paper C-D).

3.5.1 Problem studied

For case number 3 (Paper C-D) the simulation models represented the operations in three production areas in Malmberget mine. The Malmberget models were used to study the effect of ore pass loss on loading operations in a mine using sub-level caving method. The performance measures included time to mine out the production area, LHD waiting times caused by an ore pass loss, production
rate, LHD travelling distances and ventilation requirements and costs.

3.5.2 Mine site: Malmberget mine

Malmberget mine, located in Gällivare in the northern part of Sweden, is owned by LKAB and has been in operation since 1892. Malmberget mine produces around 16 Mtonnes of iron every year, with the deepest production level located 1390 m below the surface. The deposit consists of 20 separate ore bodies which vary in size and are divided into an eastern and western part. The three largest ore bodies are Fabian, Printzsköld and Alliansen (Figure 14). The deposits are situated in a 5 km long, 2.5 wide long wide zone. Mineral reserves are estimated to be 335 Mtonnes (proven) and 34 Mtonnes (probable), and mineral resources are estimated to be 6 Mtonnes (measures), 112 Mtonnes (indicated) and 180 Mtonnes (inferred). The values are based on the LKAB Annual report produced in 2016. The mine is using sublevel caving as a method of extraction. At present, there are two main haulage levels, one at 1000 m and the other at 1250 m level. As in Kiirunavaara mine, the ore is allowed to fall down into underlying production tunnels.

![Malmberget mine](Figure 14. Malmberget mine (Courtesy of Malmberget).)

The ore is loaded at the draw point by 21-tonnes diesel LHDs with 9 m$^3$ bucket size and either hauled to the nearby ore pass location
or loaded directly onto a 35-tonne truck. The number of ore passes in each production areas varies from one to four. The material is discharged via chutes at one of the two main haulage levels onto 90-tonne trucks and transported to the crusher. From the crusher locations, the ore is hoisted to the surface. The three production areas used in the study are shown in Figure 15, Figure 16 and Figure 17 with their ore pass locations and ore body outlines.

**Figure 15. Production area used in the study.**

**Figure 16. Production area used in the study.**
As in Kiirunavaara mine, once mining has begun in a block, continuous production should be maintained until all available ore is excavated. The mining sequence used by LKAB for their sublevel caving production areas follows a V-shape pattern (Figure 9).

3.5.3 Data collection

Data and information collected for Malmberget mine are summarised in Table 5. There were often different sources of the same data, and this enabled comparison and triangulation. The data collection period began in January 2013 and ended in December 2017. Interviews took place at the mine site. In addition, there were several mine visits. During these visits, each unit operation was physically observed and discussed with the operators. Internal reports and documentation of the mining operations were obtained during the mine visits, as well as data from the mining operations and data from equipment.
manufacturers. Time studies were done to ensure the appropriate level of detail on input data. Data were collected from video camera recordings for March-April 2016, and a time study was performed in September 2016.

Table 5. Summary table

<table>
<thead>
<tr>
<th>Simulation models</th>
<th>Data collected</th>
<th>Data sources</th>
<th>Amount of data</th>
<th>Collection period</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine description and layouts</td>
<td>Interviews, direct observations and internal reports</td>
<td>Documents and layout of the production areas</td>
<td></td>
<td>From: 13' January To: 16' September</td>
</tr>
<tr>
<td>Production data</td>
<td>Internal documentation</td>
<td>Transportation level locations and the total amount of material moved for year 14' and part of the year 15'</td>
<td></td>
<td>From: 14' January To: 14' October and From: 15' January To: 15' May</td>
</tr>
<tr>
<td>LHD performance data</td>
<td>Video recordings (loading, traveling, dumping, bucket weights)</td>
<td>946 cycles for part of the 16' year</td>
<td>From: 16' March 01 To: 2016 April 30</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Time studies (loading, dumping and turning)</td>
<td>20 cycles</td>
<td>16' September</td>
<td></td>
</tr>
<tr>
<td></td>
<td>WOLIS (cycles, bucket weights, locations, equipment used)</td>
<td>306 461 cycles for years 09' to 14'</td>
<td>17 Nov 14’</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Expert opinions and interviews (availability, utilisation)</td>
<td>Point estimates</td>
<td>15 April 14’</td>
<td></td>
</tr>
</tbody>
</table>

Observations from video recordings and time study visits were collected and used to identify the appropriate probability distribution for the simulation model.

3.5.4 Malmberget simulation model

The simulation model for one of the production areas in Malmberget mine is shown in Figure 18. The simulation was run
using the information obtained from the mine. The output results collected from the simulation included time to mine out the production area, energy and fuel costs and consumption, and gas emissions.

3.5.4.1 Model settings

As in the Kiirunavara mine simulation, the simulation start is based on the shift schedule (Figure 12) and cycle settings (Figure 13). In all cases, machines are ordered to travel to one of the production drifts and then to one of the ore passes. The machine is assigned to the closest ore pass, but if this ore pass is not available, it travels to the closest of the remaining ore passes. The ore pass availability depends on whether the ore pass is lost for the duration of the whole simulation run.

3.6 CASE 4: Integrating DES with a scheduling system

Unexpected events cause disturbances in operations, and readjusted schedules are often required in a very short period of time. Since manual rescheduling is time-consuming, production planners often randomly allocate the machines and activities, but
this usually leads to suboptimal results. Automated scheduling systems are used to help production planners reduce the time required to produce new schedules. Because of the dynamic nature of the mining environment and the uniqueness of the mine sites, the scheduling system should be adapted to the specific situation and will benefit by being tested in an artificial environment. Whilst not disturbing the mining operations, this will take the complex and changing restrictions of the system into account, evaluate the optimal production plans generated by the scheduler and search for new production plans when unexpected errors occur. Therefore, the next study was on the integration of DES with an automatic scheduling system (Paper E).

### 3.6.1 Problem studied

This study was performed using a simulation model of Kristineberg mine. The simulation model was used as a testbed platform, acting as the real mine system. The study showed the results of the integration of and the platform interaction between the SimMine simulation model and the ABB automated scheduling system (Paper E).

### 3.6.2 Mine site: Kristineberg mine

Kristineberg mine, located in the Boliden area in the Skellefteå field in northern Sweden, is owned by New Boliden and has been in operation since 1940. Currently, the Boliden area comprises four mines: Renström, Kristineberg and Kankberg which are underground mines and Maurliden which is an open-pit mine. The layout of the mine used in the study is shown in Figure 19. Every year, the mine produces around 700 ktonnes of ore containing zinc, lead, copper, gold, and silver, with the deepest production level located 1400 m below the surface. Mineral reserves are estimated to be 20 ktonnes (proven) and 4,880 ktonnes (probable), and mineral resources are estimated to be 50 ktonnes (measures), 5,040 ktonnes (indicated) and 7,400 ktonnes (inferred). The values are based on the Boliden Annual Report from 2017. The deposit is situated in a wide zone and consists of many ore lenses that are spread out and vary in size, metal types, and grade.
Figure 19. The Kristineberg mine simulation layout.

The mine is using cut-and-fill as a main method of extraction (Figure 20). The operations at the headings are performed using conventional drill and blast method. The broken ore is removed from the stope after each blast and transported to either loading bays or the crusher. Loading in the loading bays is performed by a diesel front-end-loader (FEL) with 14-tonne bucket capacity. Loading at the draw points is performed by diesel LHDs with 16-tonne bucket capacity. From the loading bays or directly from the headings, the material is transported to the crusher via 28-tonne trucks. From the crusher, the material is discharged onto the conveyor belt and into the hoisting skips for further transportation upstream. Extracted stopes are then backfilled with hydraulic fill. The stope is backfilled with deslimed sand tailings from the dressing plant and waste rock carried in from development drives.
3.6.3 Data collection

Data and information collected for the studied mine are summarised in Table 6. The data collection period began in January 2013 and ended December 2017. Interviews took place at Kristineberg mine. There were also several mine visits. During these visits, each unit operation was physically observed and discussed with the operators. Internal reports and documentation of the mining operations were obtained during the mine visits, as well as operational data from the mining operations. The time studies used here were performed by personnel in 2011.
### Table 6. Summary table

<table>
<thead>
<tr>
<th>Simulation models</th>
<th>Data collected</th>
<th>Data sources</th>
<th>Amount of data</th>
<th>Collection period</th>
</tr>
</thead>
<tbody>
<tr>
<td>Kristineberg mine (used in Paper E)</td>
<td><strong>Mine description and layouts</strong></td>
<td>Interviews and internal reports, documents and layout of the mine</td>
<td>Documents and layout of the mine</td>
<td>From: 13' To: 17'</td>
</tr>
<tr>
<td><strong>Production data</strong></td>
<td>Internal documentation (maintenance and operational data.)</td>
<td>Transportation levels and blasting plans for years 11' and 13'</td>
<td>Time studies performed in the mine by mining engineers for 11'</td>
<td>13'</td>
</tr>
<tr>
<td></td>
<td>Time studies (maintenance data of the equipment and operational data)</td>
<td></td>
<td></td>
<td>13'</td>
</tr>
</tbody>
</table>

#### 3.6.4 Kristineberg simulation model

A simulation model using SimMine was developed (Figure 21) and used to interact with the ABB scheduler. The transportation process used in the simulation model was run using the mine plan, machine fleet type, and size, shift schedule, layout and cycles obtained from the mine.

![Figure 21. SimMine simulation model of Kristineberg mine.](image-url)
3.6.4.1 Model settings

In this study, the scheduler setups and processing times are assumed to be deterministic. It is assumed that the operators work equally efficiently. By default, resources are assumed to be available when required. At the moment, the platform interaction between the simulator and the scheduler excludes the operational availability and location priorities. At the start of the simulation, the simulator sends the location dependencies, order list and tasks to the scheduler (Figure 22).

![Figure 22. Platform interaction.]

During the simulation run, SimMine requests work from the ABB scheduler in the form of work orders. The scheduler system is responsible for planning the work orders and the maintenance, whereas the SimMine model is responsible for simulating different states of the machines.

3.7 Verification and validation tests

In this study, the simulation model was verified and validated by performing a number of tests.

**Verification tests**

- Using debugging features
- Running the model under various conditions including degenerate tests, extreme conditions test, and event validity test
- Making logic flow diagrams
- Using and building in the diagnostics functions
- Model and animation inspections

**Validation tests**
• Internal validity
• Comparison of the output from the model with the real system
• Comparing historical data
• Face validity

The verification tests involved systematically scanning the logic, testing the operational behaviour of the studied system and confronting the model with a studied system. The vehicle’s input parameters were set to different values and changes observed in the average number of cycles (degenerate test). The number of blasts performed weekly in the given mine was compared to the number of blasts performed in the simulation (event validity test). The bucket capacity of the loaders was set to zero to see whether the production results would also equal zero (extreme condition test). The specialists and experts involved in the study were asked if the model behaved reasonably (face validity test). Verification tests included ensuring that after changing the random set number and comparing the results, the model’s results did not stand out too much from those to which they were compared (internal validity test). Internal validity was also ensured by using the following equation to find the desired precision minimum number of replications (Banks et al. 2013, p. 429):

\[
R \geq \left( \frac{Z_{\alpha/2}S_D}{\varepsilon} \right)^2
\]

where:

\[
R \quad \text{Minimum replication number,}
\]
\[
Z_{\alpha/2} \quad \text{Student t distribution,}
\]
\[
S_D^2 \quad \text{Sample variance,}
\]
\[
\varepsilon \quad \text{Practical significance difference.}
\]

The behaviour of the operational activities was tested, and the graphical user interface (GUI) was observed to see if there were any unacceptable changes in the system. Some of the results were calculated manually and compared to the run results (predictive validation test). The specified entity was traced to determine the exactness of the model’s logic (debugging technique) using an internal run controller (IRC).
CHAPTER 4: RESULTS AND DISCUSSION

The aim of this chapter is to provide the reader with the results of the study and to discuss the results.

The chapter begins with a short overview of the sub-studies and goes on to develop them at greater length. The sub-studies were the following:

4.1 Tool selection (related to paper A) – This study compared and evaluated AutoMod and SimMine tools;

4.2 Equipment selection (related to paper B) – This study analysed the alternative (electric) fleet selection and consequences of operating too many LHDs in the production area;

4.3 LHD operations affected by disturbances in the mine production system (related to papers C-D) – This study analysed flexibility and the consequences of maintaining the production in the area affected by an ore pass loss;

4.4 Integrating DES with a scheduling system (related to paper E) – This study presented the concept of using a mine simulation model as a testbed platform for mine scheduling.

Alternative approaches/scenarios/input generation were tested for each studied system (Kiirunavaara mine, Malmberget mine and Kristineberg mine). Each studied system was assessed, managed and evaluated during and after the simulation runs (related to papers A-E). For the study on tool selection (4.1) the parameters compared for AutoMod and SimMine software were production rates, software features and software performance. For the study on equipment selection (4.2), the parameters analysed and compared were hourly production rates, hourly energy consumption and CO₂ emissions. For the study on disturbance related to an ore pass loss (4.3), the parameters were production, travelling distances, waiting times and LHD productivity. The study on the automatic scheduling system via DES (4.4) considered platform integration.
4.1 Tool selection

The AutoMod and SimMine software were tested using a simulation model of one of Kiirunavaara mine’s production area, as described in section 3.4. Several combinations of machines were analysed for 15 different scenarios (Table 7). Different types of LHDs and bucket sizes were simulated to build a base for further studies and to compare the AutoMod and SimMine software performance.

**Table 7. Scenarios.**

<table>
<thead>
<tr>
<th>Scenario number</th>
<th>Bucket capacity (tonnes)</th>
<th>6:00 a.m. to 00:30 a.m. Scheduled time: 18.5 hrs</th>
<th>2:30 a.m. to 6:00 a.m. Scheduled time: 3.5 hrs</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>LHD type No. of LHDs</td>
<td>LHD type No. of LHDs</td>
</tr>
<tr>
<td>1</td>
<td>25</td>
<td>Electric 1</td>
<td>Electric 1</td>
</tr>
<tr>
<td>2</td>
<td>25</td>
<td>Electric 2</td>
<td>Electric 2</td>
</tr>
<tr>
<td>3</td>
<td>21</td>
<td>Electric 1</td>
<td>Electric 1</td>
</tr>
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<td>4</td>
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<td>Electric 2</td>
<td>Electric 2</td>
</tr>
<tr>
<td>5</td>
<td>14</td>
<td>Electric 1</td>
<td>Electric 1</td>
</tr>
<tr>
<td>6</td>
<td>14</td>
<td>Electric 2</td>
<td>Electric 2</td>
</tr>
<tr>
<td>7</td>
<td>21</td>
<td>Diesel 1</td>
<td>Diesel 1</td>
</tr>
<tr>
<td>8</td>
<td>21</td>
<td>Diesel 2</td>
<td>Diesel 2</td>
</tr>
<tr>
<td>9</td>
<td>25</td>
<td>Electric 2</td>
<td>Electric 1</td>
</tr>
<tr>
<td>10</td>
<td>21</td>
<td>Electric 2</td>
<td>Electric 1</td>
</tr>
<tr>
<td>11</td>
<td>14</td>
<td>Electric 2</td>
<td>Electric 1</td>
</tr>
<tr>
<td>12</td>
<td>21</td>
<td>Diesel 2</td>
<td>Diesel 1</td>
</tr>
<tr>
<td>13</td>
<td>25</td>
<td>Electric 2</td>
<td></td>
</tr>
<tr>
<td>14</td>
<td>21</td>
<td>Diesel 2</td>
<td></td>
</tr>
<tr>
<td>15</td>
<td>21</td>
<td>Electric 2</td>
<td></td>
</tr>
</tbody>
</table>

4.1.1 Results from the simulation runs

The results of the production rates achieved by LHDs for both AutoMod and SimMine are shown in Figure 23. Production rate is defined as tonnes loaded divided by the total number of days required to finish production. The output from running both simulation models results in similar production rates with a significance level of P-value (0.984), well above 0.05, thus indicating statistically equivalent results. The P-value is the probability of getting the acceptable results given that the hypothesis is accepted. The lowest variation is observed in scenario...
number 1. The highest variation in the production rates is observed in scenario number 10, with a 432 tonne difference. Hence, the results for the SimMine and AutoMod models show no significant variation in production rate. They provide similar output results and both can be used to study LHD loading operations.

![Figure 23. Comparison of production rates for AutoMod and SimMine tools.]

4.1.2 Software performance

The average simulation runtimes for AutoMod and SimMine are shown in Figure 24. The runtimes range from 9 to 21 seconds for AutoMod and 16 to 42 seconds for SimMine. The runtimes are higher for SimMine than for AutoMod, with the longest lasting over 40 seconds of the central processing unit (CPU) clock time (scenario 11). The CPU time reaches high values in scenario numbers 6 and 11 when simulating with SimMine and scenario numbers 5 and 11 when simulating with AutoMod. This extra time is required to process the simulation run in the scenarios with 14-tonne electric LHDs. Transporting the same amount of tonnes with a 14-tonne bucket LHD results in more cycles than with any other LHD with a bigger bucket load; this requires extra processing.
Figure 24. Comparison of simulation runtime for AutoMod and SimMine.

Depending how the AutoMod model is constructed and run, the time to simulate the scenarios can increase/decrease. For example, running the AutoMod in windowless mode or temporarily disabling the logic that updates the graphical user interface (GUI) will speed up the processing time. If there is a possibility of switching off some of the functions in SimMine that are not used, the runtime might decrease. Runtime is not a problem for small models but with an increased level of detail and information, the runtimes will significantly increase, especially for AutoMod models.

4.1.3 Software evaluation and recommendations

AutoMod enables the user to access the model code, with access to tracing and the internal run controller (IRC). It has other experimental and statistical facilities as well (such as the possibility of influencing the random stream numbers and implementing user-defined distributions), making it a favoured option. However, it takes time to generate the model, and knowledge of the system is required. Depending on the size and complexity of the model, the user (even experienced) will usually spend more time to build and verify the AutoMod model than the SimMine model. The latter has a base model already encased in a structure of the software.
SimMine software should be used in situations where easy and fast modelling is required and in cases where the given or specified system can be modelled by SimMine with the required level of detail specified in the objectives. AutoMod software should be used in situations where more enhanced and detailed simulations are needed, taking into account that it requires more time and user experience, as well as knowledge of how the mining systems are operating.

4.2 Equipment selection

The LHD type was determined by analysing 14 different types of LHDs, seven 1D-7D diesel LHDs and seven 1E-7E electric LHDs, where 1D-7D and 1E-7E refer to bucket sizes for diesel and electric LHDs respectively; the smallest are 1D and 1E and the largest are 7D and 7E. The production rates, energy consumption and CO₂ emissions of the different types and bucket sizes were compared. The LHD production rate was determined by dividing the total number of tonnes by the hours the LHD spent in operation. The energy and fuel consumption was determined based on the following factors: loading rate, vehicle efficiency, road gradient and surface features, load factors, and LHD operating time. Vehicle efficiency was defined as the percentage of total time that a machine actually operated. The load factor was estimated based on the ratio of the machine input power to the machine drive power. This was only possible for the LHD 7D, since only this machine was operating in the mine. Thus, its load factor (78%) was used for other LHDs.

4.2.1 LHD production rate

The results of the LHD production rates (Figure 25) obtained from the simulation show that the diesel LHDs achieve higher production rates than electric LHDs with similar bucket sizes. The diesel LHDs achieved faster travelling speed in the simulation, resulting in shorter cycle times.
4.2.2 Energy and fuel costs

The results for energy and fuel costs are presented in Table 8, and energy consumption is shown in Figure 17. Prices from the International Energy Agency (fuel price of US$1.79/L and electricity price of US$0.12/kWh) were used to calculate the electricity and fuel costs. Energy costs per hour were obtained by multiplying the hourly consumption (Table 4) by the fuel/electricity price. Energy costs per tonne were calculated by dividing the energy costs per hour (Table 4) by the LHD production rate (Figure 16). The costs per hour and costs per tonne do not cover the costs related to worn parts, tires, lubrication and maintenance and overhaul costs.

Table 8. Comparison of energy consumption and cost for diesel and electric LHDs.

<table>
<thead>
<tr>
<th>LHD type</th>
<th>Consumption</th>
<th>Costs per hour</th>
<th>Costs per tonne</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Diesel (L/h)</td>
<td>Electric kWh/h</td>
<td>Diesel (US$)</td>
</tr>
<tr>
<td>1D</td>
<td>9.89</td>
<td>25.31</td>
<td>17.70</td>
</tr>
<tr>
<td>2D</td>
<td>10.43</td>
<td>23.30</td>
<td>18.70</td>
</tr>
<tr>
<td>3D</td>
<td>20.30</td>
<td>38.67</td>
<td>36.30</td>
</tr>
<tr>
<td>4D</td>
<td>28.90</td>
<td>75.67</td>
<td>51.80</td>
</tr>
<tr>
<td>5D</td>
<td>32.10</td>
<td>126.52</td>
<td>57.40</td>
</tr>
<tr>
<td>6D</td>
<td>34.70</td>
<td>217.87</td>
<td>62.20</td>
</tr>
<tr>
<td>7D</td>
<td>42.30</td>
<td>306.59</td>
<td>75.80</td>
</tr>
</tbody>
</table>
The results show that the calculated hourly costs (Table 8) are significantly higher for diesel fuel than electricity. The fuel cost for the diesel machines (1D-7D) is, on average, US$0.24/tonne, and for the electric machines (1E-7E) it is, on average, US$0.07/tonne. Therefore, in this scenario, the energy cost is US$0.17/tonne more for a diesel LHD than for an electric LHD.

With increased bucket size, there is an increase in energy cost per hour because the heavier machines require more drive power. In this study, the machine with an operating weight of 8.7 tonnes has a drive power of 71 kW, and the machine with an operating weight of 56.9 tonnes has a drive power of 350 kW. The energy cost per tonne for diesel machines decrease with increasing machine size, but for electric machines they increase. However, using diesel LHDs increases the ventilation costs, and this must be also accounted for when selecting the fleet for underground mining operations.

4.2.3 Energy consumption

Comparison of the hourly energy consumption of diesel and electric LHDs is shown in Figure 26. To compare the two types of LHDs, diesel fuel consumed was converted to kWh using equation 12. By calculating the ratio of the LHD productivity to the hourly fuel consumption, the amount of energy consumed per tonne by the electric LHD 7E is determined to be 0.75 kWh/tonne. When similar bucket sizes are compared, the hourly energy consumption is higher for diesel LHDs than for electric LHDs. For example, the diesel LHD 7D consumes 455 kWh of energy, whereas the electric LHD 7E consumes 306 kWh (Figure 26). The diesel LHD 7D travels faster than the electric LHD 7E, but more drive power is used and more energy is consumed. This generates more production, but consumes more energy indicating that electric LHDs are more efficient in energy consumption.
Figure 26. Comparison of energy consumption by diesel and electric LHDs.

In conclusion, the electric LHDs are an advantageous alternative for underground mines because of the way they conserve energy. Minimising the use of diesel machines and increasing the use of electric machines would reduce energy consumption.

4.2.4 Fuel consumption and CO₂ emissions

In this study, the CO₂ emission was estimated based on equation 15. An estimation was used because measuring CO₂ emission is time-consuming and requires extensive field measurements. The results in Figure 27 show the hourly CO₂ emission and hourly fuel consumption from seven different diesel LHDs (1D-7D). The difference in fuel consumption is 32.41 litre/hour and CO₂ emission is 87 kg/hour for LHD 1D and LHD 7D respectively. With increased bucket size and increased number of diesel LHDs in operation, more CO₂ emissions are generated and more ventilation is required. Electric LHDs have no CO₂ emissions, and less energy is consumed, making them an advantageous alternative for underground mines. However, electric LHDs require additional infrastructure (Jacobs et al. 2015). Additional infrastructure refers to supplying electric LHDs with electric power. The ways to deliver
electric power vary; power can be delivered by electric cable, battery or pantograph, but all restrict the LHD’s flexibility of movement. Calculating the ratio of the LHD fuel consumption and CO₂ emissions shows that the diesel LHD 7D consumes 0.1 L/tonne and emits 0.27 kg of CO₂ per tonne. In contrast, the electric LHD 7E has no CO₂ emissions.

![Figure 27. Comparison of diesel LHDs’ CO₂ emissions and fuel consumption.](image)

4.3 LHD operations affected by disturbances in the mine production system

As ore passes are critical structures and ore pass unavailability/loss was identified as one of the critical risks for underground mines (Greberg and Salama, 2016), ore pass unavailability was chosen in the study to represent the reason for a major disturbance in the mine. In this study, the effects of ore pass loss on the LHDs’ operations were evaluated and analysed using DES. The scenarios in the study represents situations with different number of ore passes available and different number of LHDs in operation. A maximum of six LHDs in operation was used in this study; operating with more LHDs in the studied production area would result in dramatically longer waiting times and queuing, and would be highly unpractical. The simulation was based on 15 scenarios (Table 9) for each of the three production
areas in Malmberget mine: production area A, production area B and production area C. Since production area A and production area B had a similar pattern, production area C is used in what follows. For more information on the other production areas, see the related publication (paper C).

**Table 9. Simulated scenarios.**

<table>
<thead>
<tr>
<th>Scenario no.</th>
<th>No. of operational ore passes</th>
<th>Ore passes operational state</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>FL</td>
<td>CL</td>
</tr>
<tr>
<td>1</td>
<td>4</td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>3</td>
<td>1</td>
</tr>
<tr>
<td>3</td>
<td>3</td>
<td>1</td>
</tr>
<tr>
<td>4</td>
<td>3</td>
<td>1</td>
</tr>
<tr>
<td>5</td>
<td>3</td>
<td>0</td>
</tr>
<tr>
<td>6</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>7</td>
<td>2</td>
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<tr>
<td>8</td>
<td>2</td>
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<td>9</td>
<td>2</td>
<td>1</td>
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<tr>
<td>10</td>
<td>2</td>
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<tr>
<td>11</td>
<td>2</td>
<td>0</td>
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<tr>
<td>12</td>
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<tr>
<td>13</td>
<td>1</td>
<td>0</td>
</tr>
<tr>
<td>14</td>
<td>1</td>
<td>0</td>
</tr>
<tr>
<td>15</td>
<td>1</td>
<td>0</td>
</tr>
</tbody>
</table>

In Table 9, ‘FL’ stands for the ore passes located on the far left side of the studied production areas; ‘CL’ stands for the ore passes located closer to the middle on the left side; ‘CR’ stands for the ore passes located closer to the middle on the right side; ‘FR’ stands for the ore passes on the far right side.

4.3.1 Time to mine out the production area

Figure 28 shows the time required by LHDs to finish mining the whole production area for given scenarios, based on a maximum number of LHDs and operational ore passes. The time to finish mining the production area in ore body C (Figure 28) ranges from 856 days to 7,644 days, depending on the scenario and the number of LHDs used. When the number of LHDs is increased, the time to finish mining the production area decreases. When the number of
operational ore passes decreases, the time to finish mining the production area increases. There is a possibility of increasing the number of LHDs in the area where ore pass loss occurs. This would reduce the time to finish the production area but may increase the queuing time, depending on the number of LHDs used. If the number of operating LHDs is high, the production does not greatly improve; relocating the fleet and using an additional number of LHDs in another area to achieve a higher production rate might be a viable alternative.

![Figure 28. Time to finish mining production area C.](image)

The results show how important it is to have a well-developed strategy for managing a major disturbance to the mine production system, such as an ore pass loss. Operating with a too high number of LHDs in a production area affected by an ore pass loss causes severe LHD production disturbances and lowers the LHDs’ efficiency. The time to finish mining the production area is significantly reduced when the number of LHDs increases from one to three; however, increasing the number of LHDs from three to six does not significantly reduce the time to finish mining the production area.
4.3.2 LHD waiting times

The LHD waiting times for production area C are shown in Figure 29. The LHD waiting time is determined by the percentage of the total operating time that LHD operators have to wait in the production drifts for an ore pass to become available. With a decreasing number of available ore passes, the waiting time increases. The sum of waiting times for each scenario ranges from 0% to 61% of the total time, increasing from 0% to just under 5% when three ore passes are available, going as high as 20% when two ore passes are available and jumping to 61% when only one ore pass is available. Extra waiting time makes the use of LHDs inefficient. Operating with a high number of LHDs in areas affected by an ore pass loss will lead to longer waiting times, decreasing the LHDs’ efficiency and disturbing the LHD production rate.

![Figure 29. LHD waiting times for the ore pass to become available (production area C).](image)

4.3.3 LHD production rate

The production rates for the different areas and scenarios are shown in Figure 30 and Figure 31. The production rates are here
determined by dividing the total tonnes loaded by the total number of days required to finish mining a production area. The production rates vary depending on the scenario number and production area. Due to ore pass unavailability, the difference in the production rate when deploying three LHDs (Figure 30) reaches 6,920 tonnes/day, whilst the difference in production rate when deploying six LHDs (Figure 31) is 14,235 tonnes/day. The highest production rates are achieved in production area B, and the lowest are in production area C.

![Figure 30. Scenario vs. production rate for different areas based on the use of three LHDs.](image)

![Figure 31. Scenario vs. production rate for different areas based on the use of six LHDs.](image)
The differences in production rates depend on the average distance the LHD needs to travel from the draw point to the ore pass. Figure 32 and Figure 33 correlate the LHDs’ production rate and travelling distance, based on production areas where one LHD (Figure 32) and six LHDs (Figure 33) are in operation. When the number of available ore passes is low, the LHDs have to travel farther. The travel distances are the longest when the only operational ore passes are located to the far-left and far-right side of the production area. This can be seen by comparing scenario numbers 13 and 14 with scenario numbers 12 and 15 (Figure 32).

**Figure 32.** Correlation of LHDs’ production rate and travelled distance in production area C based on the use of one LHD.

In similar manner (Figure 33), a low number of operational ore passes results in increased distance travelled and decreased production rates. The effects of travelling longer distances are further observed by comparing scenario numbers 13 and 14 (one ore pass available closer to the center of the production area) with scenario numbers 6, 8 and 9; the results for scenario numbers 8 and 13 are close in their production rate and total distance travelled. Thus, increasing the number of LHDs in the area affected by an ore pass loss reduces the LHDs’ efficiency and should be avoided.
Figure 33. Correlations between production rate and distance in production area C based on the use of six LHDs.

4.3.4 Ventilation costs and production rates related to LHD operations

The results for the ventilation costs and production rates in production area C are shown in Figure 34 for one, three and six LHDs. The energy and ventilation costs are for the 21-tonne diesel LHD with a drive power of 350 kW and vehicle weight of 56.8 tonnes. The ventilation costs were calculated based on the technical and operating parameters of the 21-tonne LHD (motor power, operating times, production rates), ventilation system of the studied production area (flow rate capacity, ventilation requirements) and electricity price of 0.12 US$/kWh (EIA, 2017).
Figure 34. Performance of LHDs in production area C.

With an increasing number of LHDs in operation, the production rate decreases by as much as 13,642 tonnes/day, and fuel consumption increases by 0.2 liters/tonne. The highest production rate (20,350 tonnes/day) is achieved in scenarios where six LHDs are in operation, and the lowest production rate (2,280 tonnes/day) is achieved when one LHD is in operation. Thus, with fewer operational ore passes, the fuel consumption increases because of increased LHD travel distances. A decision to increase the number of LHD machines in operation affected by an ore pass loss results in a potential loss of production. For example, using three LHDs in three separate working areas with conditions similar to scenario 1 will result in a production rate of 13,614 tonnes/day, but using the same three LHDs in the same production area not affected by an ore pass loss results in a production rate of 12,499 tonnes/day. The difference increases to 6,879 tonnes/day with an increasing number of LHDs. This means that if six LHDs are used in six separate working areas, 27,228 tonnes/day are produced compared to 20,349 tonnes/day. If the disturbances related to ore pass loss are added, as for example, considering scenario number 15 where only one ore pass is available, the difference reaches 8,035 tonnes/day for three LHDs and 20,521 tonnes/day for six LHDs.
A similar pattern appears for the ventilation costs. For three and six LHDs working in three separate production areas, the ventilation costs are 0.27 US$/ktonne and 0.54 US$/ktonne for three and six LHDs respectively, but if the same LHDs are operating in one production area, the ventilation costs are 0.29 US$/ktonne and 0.71 US$/ktonne respectively. In the areas affected by an ore pass loss, the greatest difference is observed in scenario number 15, where ventilation costs reach 0.39 US$/ktonne for three and 1.64 US$/ktonne for six LHDs in operation.

4.3.5 Utilisation of the fleet and ore passes

Table 10 shows the variations in production rates and ventilation costs for the different scenarios. The results are presented in percentages and were determined by dividing the production rate obtained in scenario number 1 by the production rate achieved in scenario numbers 2-15. The ventilation variations were determined by dividing ventilation costs obtained in scenario number 1 by the ventilation costs achieved in scenario numbers 2-15.

Table 10. Variation in the percentage of production rates and ventilation costs.

<table>
<thead>
<tr>
<th>Scenario number</th>
<th>1 LHD</th>
<th>3 LHDs</th>
<th>6 LHDs</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Production rate (%)</td>
<td>Ventilation (%)</td>
<td>Production rate (%)</td>
</tr>
<tr>
<td>4 ore passes in operation</td>
<td>100%</td>
<td>100%</td>
<td>100%</td>
</tr>
<tr>
<td>3 ore passes in operation</td>
<td>93%</td>
<td>108%</td>
<td>93%</td>
</tr>
<tr>
<td>2 ore passes in operation</td>
<td>91%</td>
<td>109%</td>
<td>93%</td>
</tr>
<tr>
<td>5</td>
<td>92%</td>
<td>109%</td>
<td>92%</td>
</tr>
<tr>
<td>6</td>
<td>85%</td>
<td>117%</td>
<td>82%</td>
</tr>
<tr>
<td>7</td>
<td>68%</td>
<td>147%</td>
<td>66%</td>
</tr>
<tr>
<td>8</td>
<td>75%</td>
<td>133%</td>
<td>67%</td>
</tr>
</tbody>
</table>
Based on the calculated results (Table 6), the production rate decreases from scenario 1 (four ore passes in operation) by as much as 50%, 45% and 33% when using one, three and six LHDs respectively (one ore pass in operation). When using one, three and six LHDs, the ventilation costs increase from scenario 1 (four ore passes in operation) by as much as 200%, 224% and 306% when using one, three and six LHDs respectively (one ore pass in operation). This suggests that with an increasing number of LHDs in the area affected by disturbances related to ore pass loss, the production rate decreases, and the ventilation costs increase. These results are related to a specific study, but they highlight the need for a well-developed strategy in order to mitigate the disturbances that negatively affect the system.

4.4 Integrating DES with a scheduling system

In this study, a simulation model of an underground mine system was connected to an automated mine closed-loop scheduling system to act as a test bed and decision support system. A closed loop scheduling system is regulated automatically without human interaction, whereas an open loop control system requires manual inputs. ABB’s automated Ability Operations Management System (OMS) is responsible for generating schedules and assigning tasks. The SimMine simulation acts as a real mine, simulating the real mining operations, such as loading, and sending the events to the scheduler. The simulation and production scheduling discussed here are based on the Kristineberg mine operations.

4.4.1 Platform integration

The integration of the SimMine simulator and the ABB Ability OMS is presented in Figure 35. The layout (Figure 35) includes
information on production/development plans, maintenance plans, system settings, localisation, and additional mine services that can affect scheduling.

**Figure 35. Layout of the integration of the scheduler and the simulator.**

The automatic scheduling feature of the ABB Ability OMS enables task integration and supports manual scheduling and intervention. The system can be continually updated with the latest information. Based on the information received from the mine, SimMine acts as the mine operation centre (MOC), coordinating activities. The results are presented in form of Gantt charts (see Figure 36) from two perspectives: machine-centric and location-centric. Sending feedback on work progress, together with reasons for machine stops, from each operation to the fleet management system improves the decision making for the automatic scheduling of the production cycle. Users can also interact (in the Gantt chart) with the current schedule by reassigning orders to different equipment or editing the properties of the resources.
Testing different algorithms, procedures and rules in the artificial environment prevents potential time and cost losses related to on-site testing and allows the results to be viewed before they are implemented in the real system. The flexibility to select alternative scenarios and to examine any effects on the process control system within a short period of time in a safe environment reduces operator stress, provides instant task progress reports, reduces administrative workload, coordinates mine systems with fewer human errors and unifies scheduling.
CHAPTER 5: CONCLUSIONS

The aim of this chapter is to provide the reader with conclusions and answers to the research questions.

The research presented in this thesis addresses the following issues:

- Challenges related to LHD operations in underground mines;
- Tool selection and equipment selection (electric versus diesel);
- Disturbances in LHD operations related to ore pass losses;
- DES model connected to automatic production scheduling system.

**RQ1** What are the major challenges related to LHD operations in underground mining?

The literature review shows that with increasing mining depth, efficient mining processes are extremely important because distances, stresses, seismicity, energy consumption, ventilation requirements and travelling distances will increase (based on the I²Mine study). Also, production disturbances are critical for deep underground mines and affects the mining operations. With increased ventilation requirements and deeper mines, larger and more powerful ventilation systems are required, resulting in high energy costs and high ventilation requirements. To cope with the numerous challenges of the future, more attention must be paid to automation, to flexibility and to the transition from diesel to electric equipment.

**RQ2** How can the use of DES improve the LHD system?

- The study presented in Paper A shows that the software available on the market provides a wide range of capabilities for modelling. Depending on the choice, the parameters and functions can be controlled and preinstalled, providing fast and easy development of a simulation model for mining operations.
• The study presented in Paper B shows the possibility of using DES to study the equipment selection process, relating it not just to production but also to energy consumption and gas emissions.

• The study presented in Paper C shows that DES can be used to study and test various scenarios of LHD operations when subject to mine production system disturbances. This paper studies the possible consequences of incorrect management of LHD operations when one or more ore passes become unavailable, highlights the need for a well-developed strategy for LHD operations. It makes suggestions for devising a strategy to mitigate the production disturbances related to unavailability of ore passes. DES helps users see which alternatives/options that maintain or even improve production.

• The study presented in Paper D shows how ventilation costs related to LHD operations can be studied using DES. The system can be analysed before selecting a fleet. This is important because production requirements, ventilation costs and work environment are crucial aspects of fleet selection.

• In Paper E, a concept of using a mine simulation model as a testbed platform for a mine scheduling system including LHDs, and how DES can be integrated with a scheduler is presented. Using DES to study the results of other models lets the user test various types of input generation, logic and behavior of the scheduling system. It provides a platform for testing the system in a controlled environment and adds alternatives beforehand.

RQ3 How can the use of electric LHDs change the loading operations?

• The study in Paper B shows that based on current energy prices, the energy costs for diesel and electric LHDs are US$0.24/t and US$0.07/t, respectively. Diesel LHDs emit 2.68 kg of CO2 gas per litre of fuel. These emissions require
more ventilation, generating additional costs. Electric LHDs have zero CO\textsubscript{2} emissions and produce less heat.

- The study presented in Paper D shows that in the area affected by an ore pass loss, the energy cost per tonne and CO\textsubscript{2} emissions increase. When four operational ore passes are compared to one operational pass, the difference in energy costs is 0.36 US$/tonne, and the difference in CO\textsubscript{2} emissions is 2.1 kg/tonne. The ventilation costs are 2.18 $/ktonne/unit when six LHDs are in operation compared to 0.18 $/ktonne/unit or 0.66 $/ktonne/unit when one and three LHDs are operated respectively. When there is a loss of ore passes and more LHDs are added, there are additional ventilation costs of 2US$/ktonnes/unit. For the studied cases, minimising the use of diesel LHDs and maximising the use of electric LHDs will reduce energy costs and the need for ventilation to mitigate CO\textsubscript{2} emissions.

**RQ4** How can automated scheduling influence the loading operations?

- The study presented in Paper C shows that operating with a high number LHDs in the production area affected by an ore pass loss results in higher variations in production rate than when operating with a low number of LHDs. To avoid increased waiting times and traffic congestion by using too many machines in the working areas, the suggestion is to evaluate whether the system can accommodate an extra machine in the production area by using an automatic scheduling system that will optimally redirect the machines to the alternative locations.

- The study presented in Paper E shows that an automated scheduling system connected to the simulator can be used to test different algorithms, procedures/rules and scenarios. This prevents potential time and cost losses related to tests being performed on the site and allows the results to be viewed before they are implemented in the real system. The automatic scheduling system reduces
operator stress, provides instant task progress reports, reduces administrative workload, coordinates mine systems with fewer human errors and unifies scheduling. Schedules produced in real-time give the flexibility to select alternative scenarios and the ability to examine any effects on the process control system, such as loading operations, especially in cases of disturbances when new alternative schedules have to be provided.
CHAPTER 6: FUTURE STUDIES

The aim of the chapter is to provide the reader with a brief overview of suggested future studies.

The simulation models create a baseline for future studies of possible alternative underground transportation systems and processes. Based on the results from this research, future studies should be conducted in the following areas.

6.1 Electrification

Operating at greater depths will increase energy costs and ventilation requirements. With increased operational costs, the choice of LHDs is an essential consideration. With increasing ventilation problems and more regulations at greater depths, it is important to study the selection process and the performance of electric LHDs, along with alternative ways of powering them, safety issues and required infrastructure.

6.2 Alternative strategies and flexible systems

When major disturbances occur in the mine system, for example, when an ore pass is long term unavailable, mine operators are forced to seek alternative methods and strategies to mitigate the disturbance. Further studies are required, including work on better mitigation strategies, mine infrastructure and also improved ore pass design. One option is to use an automatic scheduling system to provide the planner with the best solutions and mitigation strategies. Other new options should aim at creating contingency plans in the form of, for example, guidelines that would eliminate the risk related to production loss.

6.3 Automation and flexibility

The possibility of connecting the scheduler to the mine control system, together with the possibility of the attached simulator acting as a digital twin, opens up new ways of looking at mining operations. With further development of the platform, scheduling could become more flexible and automated, with the ability to handle major disturbances and improve task execution.
REFERENCES

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LIST OF APPENDED PAPERS
Selection of discrete event simulation software for simulating mining operations

Selection of discrete event simulation software for simulating mining operations

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Abstract: Simulation is increasingly gaining attention as one of the tools that can be used to predict and evaluate the performance of mining systems. It has been used for various applications such as fleet optimization in underground mining, comparison of timing and efficiency between drills, and mine to mill production systems. Due to the availability of a large number of simulation tools, careful selection should be made depending on the type of problem to be simulated. The study presented in this paper aims to compare two different simulation tools, AutoMod and SimMine, by comparing the two underground loading models created. This is achieved by analysing different equipment alternatives for possible future conditions when the mine depth increases. Both tools produce statistically equivalent results for simulated production. The paper presents a discussion regarding the choice of software and based on the study, SimMine is recommended for easy, fast modelling, whereas AutoMod, with its wider palette of software features and facilities provided, is recommended for more enhanced and detailed simulations. The study is based on the mining operation in the Kiirunavaara underground mine, and the simulation is conducted based on a fixed production target.

Keywords: Loading system, discrete event simulation, software selection

1 Introduction

There are many techniques available for estimating and maintaining the number of pieces of equipment in mining, such as theoretical approaches and simulation. The theoretical estimate of the number of loading units is generally a quick method, but has the drawback of not including some important factors like variability of tramming distances, variability of vehicle performances, queuing, and traffic congestion when more than one LHD is utilized (Atkinson, 1992; Raj et al., 2009). The most accurate way to estimate the number of loading units is to utilize simulation (Sturgul, 1999; Raj et al., 2009; Greberg et al., 2016). The simulation model will estimate the required number of LHDs considering the variability of the data, thus resembling the real-life scenario in a more accurate way than when using analytical methods. The use of computer simulation will allow for a much quicker evaluation of different loading units and better control of the processes of the currently running system. The drawback with using simulation is that generating the model and required output can be a time-consuming process. Common features of these tools such as animation and graphical interface offer a direct approach to the increased understanding of a specified mining environment (Raj et al., 2009; Banks, 2004). These tools enable better interaction between the variables and the system performance in applying different rules and procedures in form of functions, probability distributions, processes, and algorithms. They also help to solve the issues related to fleet requirements or mine planning optimization problems. However, the method requires the collection of a large amount of data and often requires the data to be fitted to statistical distributions. Failure to do so may result in inaccurate values and wrong conclusions. This means that the data from different processes and the model itself should be frequently validated (Sturgul, 1999). Simulation is well-established in operations research and industrial engineering. However, much less literature exists on the results of comparison between two simulation models of the same mining system, where not only the software but also the built-in models are compared with each other. In this study, the simulation models were developed using AutoMod and SimMine tools.

2 Selecting simulation software

To select the appropriate software, Nance (1995) proposed six main characteristics. The software should have a capability to generate the random numbers and permit the use of other variations of random distributions. The software should also have the capability of listing the processing in such way that the object can be created,
manipulated and deleted. Additionally, the software should include features such as statistical analysis, report generation, and time flow capability. Many software programs meet these requirements, thus further evaluation is necessary. An example of such an evaluation was presented by Albrecht (2010), who identifies 87 discrete event simulation and modelling tools which include simulation programming languages. Albrecht (2010) then combines the guidelines for selecting the simulation software from Ahmed, et al. (2003), Banks and Gibson (1997), Hlupic, et al. (1999), and Page (1994). When selecting the simulation the procedure covers seven major areas to look at (Albrecht, 2010):

- Modelling Environment – varying degrees of details/levels of description, full accessibility to the underlying model code, and good readability of the code and model animation with virtual reality as a bonus.
- Model documentation and structure – varying levels of model description from a very high level to a very low level and independence of the system and computer architecture.
- Verification and Validation – tools to support verification and validation of the simulation models.
- Experimentation facilities – automatic batch runs including insertion of the warm-up period and independent replications with re-initialization between the runs.
- Statistical facilities – alternative statistical distributions together with the possibility of handling large volumes of random numbers. Additionally, the statistical facilities that enable standard experiment analysis functions such output data analysis.
- User support – package maintenance, technical and promotional information including a training course, tutorials, and demonstration models.
- Financial and technical features – cost of the package, ease of use and ease of installation together with hardware requirements.

In this study, the simulation tools AutoMod (manufacturing-oriented software) and SimMine (mining-oriented software) were used to model the Kirunavaara mine. They were selected, as they can model the rock transportation systems of the selected mines with the necessary level of detail, and they are also commonly used simulation language environments capable of satisfying the characteristics necessary for the study. Two models were built using the two different software programs which provided opportunities to explore the variations and limitations of them both.

**AutoMod**

AutoMod software consists of a material movement system with built-in and simulation environment (Rohrer, 1997). The graphics offer two modes: static and dynamic where the moving objects can be observed during the simulation run. The flexibility comes from AutoMod syntax and built-in environments. There is a possibility to modify various parameters via AutoMod syntax or directly in the movement system such as speed, turning speed, acceleration or deceleration (Banks, 2004; Yuriy and Runciman, 2013). In addition to the AutoMod syntax, the software includes built-in templates: vehicle path mover system, conveyors, automated storage and retrieval system, bridge cranes, power and free conveyors and kinematic (robotic) systems. These features, combined with panel control in form of drag-and-drop and _m logic creates powerful simulation software that is able to communicate with control systems and other simulation models. Performance reports with statistics and 3-D animation are created automatically, providing a realistic and statistically accurate view of the system, helping to verify and validate the models. Advanced debugging and trace facilities enable easy tracking of errors and flaws (Banks et al., 2010). In the AutoMod the loads move through processes and compete for resources (Banks et al., 2010). The AutoMod also includes AutoView, AutoStat, the model zip archiver, the OPC utility, the process server and the web tools utilities such as SimController or ACE graphics editor (Banks, 2004). In SimController it is possible to view the results of the model and redirect useful output files for multiple runs. SimController also enables comparison of the reports, viewing Gantt charts and viewing the model information. The ACE is the graphics editor that offers direct control of the graphics represented in a model. With ACE graphics you can create, edit and delete the model elements. AutoStat is an extension to AutoMod created for the purpose of designing experiments by running multiple simulations under different statistical constraints, performing sensitivity analysis or optimizing the model (Banks et al., 2010). AutoView is an extension to AutoMod where video clips from the AutoMod models can be made. While making the movie there is set of customization tools that enable the user to follow the vehicles with the camera in the recorded simulation run or set the camera to certain views for the specified time period. The possible movie formats are AVI and MPEG.

**SimMine**

SimMine is a discrete event simulation software that uses a full graphical interface and does not require coding. The logic and behaviour of different entities is already built into the software and can be controlled through their properties. To incorporate the randomness/variance in the processes the tool utilizes statistical distributions and uses a point-and-click interactive spreadsheet for entering the data. An extra feature is the interactive 3D layout which allows the user to point-and-click and set properties directly from the 3D
layout but also to visually inspect and analyse the sequence of development in the animation of the development process. Since it is developed specifically for the simulation of mining operations, it is made of interfaces where the user can set and specify underground mine-related properties such as rock properties, face profiles, activity cycles, etc. Equipment interface and delays can be visually analysed for better evaluation of the statistical results generated by the model (Yuriy and Runciman, 2013). SimMine software offers direct import of the mine layouts (CAD layouts), comprehensive model output statistics, the capability to evaluate the design of the production facilities and the selection of production equipment such as trucks and loaders. Results come from simulating the real-life underground data that incorporates all the equipment performance data, operational data and other relevant data such as rock properties.

3 Case study

The Kiirunavaara mine is an underground iron-ore mine located in northern part of Sweden that uses the sublevel caving mining method (Figure 1). The mine consists of a high-grade magnetite deposit approximately four kilometres long, with a dip of about 65 degrees from the horizontal plane and with an average thickness between 80 and 100 meters in the northeast direction.

A three-meter blasting round on each production drift starts in sequence at the hanging wall, commonly using an upward raise to provide a free face, and then retreats toward the footwall. Mucking out by LHD continues until the waste dilution reaches the set limit. The amount of ore produced is approximately 10,000 tonnes from one blasted ring, which makes the total ore to be mined from this block approximately 6.17 Mtonnes. It is assumed that loading starts from the first drift on the left side of the block and continues to the next drift until the last drift is finished. The same procedure is repeated until the whole production area is mined.

As seen in Figure 2, two ‘parking areas’ were defined, which were used as waiting places for LHD machines during breaks. When dumping, a machine working on the left-hand side of the mine block will use the first two ore passes, while when working on the other side of the block, the machine will dump material to the remaining ore passes. The usage of the two ore passes on each side of the working area depends on the buffer capacity and boulder frequency. If the buffer capacity limit is reached, then the LHD operator will move to the next ore pass. If a boulder enters the ore pass, the LHD operator will wait for up to five minutes; when waiting time exceeds five minutes, the LHD operator will move to the next ore pass. If the rock breaker is working to clear the boulders, and at the same time the next ore pass buffer capacity limit is reached, the LHD operator will wait for the rock breaker to finish removing the boulders. The simulation of the base case model includes one 25-tonne electric LHD machine working from 6 a.m. until 10 p.m. before blasting, and one 21-tonne diesel LHD machine working from 10 p.m. until 6 a.m. The results from the base case scenario are compared with the results from the analysed scenarios.
The procedure of building SimMine and AutoMod models begins with importing the planned layout into the graphical environment. The next step is to define all the necessary parameters such as performance characteristics of the equipment, rock properties (ore/waste), face profile, mining cycles, shift schedules, blasting times and/or costs. Further actions are to implement necessary procedures and interactions between the operating machines. In SimMine this is done by choosing appropriate options in point-and-click menus. In case of AutoMod, it is necessary to incorporate the programming code. The base case model is developed to benchmark against the planned development/production rate or already existing system. Further steps involve running alternative scenarios and verification and validation of the models by means of the physical and statistical tests. During the process of model development, the reporting and documentation on various optimization scenarios including variations such as additional equipment, shift or different performance characteristics are arranged. The final stage involves making the recommendations in order to improve or optimize the studied processes in the system.

**AutoMod cycle settings**

To emulate the movement of LHD machines the AutoMod uses a process-interaction method. The LHD machine would move until it arrives at the particular location, gets delayed or leaves the process. The process is created and controlled in the AutoMod by use of source codes (.m syntax) or through point-and-click control panels. An example of such a process used in this study for diesel and electric LHD machines is shown in Figure 3. The code emulates the cycle procedure. The process begins by calling the function that finds the next work location to which the LHD machine should travel. Next is to set operational parameters. The operational parameters, in this case, are the bucket capacities, acceleration, deceleration, turning time and empty/loaded vehicle speeds. Subsequent statements ensure that the work is continued until all of the material has been drawn from the production drifts and if necessary appropriate changes have been made to the specified destinations. This is done by sending the LHD to the assigned working locations. Upon arrival, if necessary, the machine is sent to order list to await further instructions. Further down the list, the loading information is updated. After updating the machine and production area information, the function that sets speed is invoked to send the LHD machine to the closest dumping location. If there is a boulder in the ore pass, the LHD operator waits for the ore pass to be cleared and then dumps the material into the ore pass. However, if the time to dump the material into the ore pass is predicted to take longer than five minutes the LHD operator drives on and dumps the material in the adjacent ore pass. Afterwards, the information such as the amount of transported material is updated. Next, the cycle repeats until all the material has been extracted.

**SimMine cycle settings**

In SimMine, to emulate the loading and transportation of the LHD machines the user should use the point-and-click menus. There is no need to use coding and, since the software is mining-related, many additional features can be implemented into the model by checking the right option from the drop-down menu list. For each activity, a set of additional properties can be changed depending on the requirements. Further adjustments can be made in the mine layout. To control the movement of the LHD machines, the transport method (LHD, Truck) is set from the loading location to the chosen destination in the ‘Loading and transport activity’ tab. The fleet properties can be changed by navigating to ‘Fleet’ tab submenu. The user can set the occurrences of boulder frequency based on either tonnage (mean tonnage between failures) or rounds (risk of boulder per round) but also set the time to fix the boulders based on selecting the probability distribution. At the end of the simulation, the results are presented under several tabs. Each tab consists of different information such as location and vehicle sequences but also production statistics, time statistics, vehicle statistics and costs information. The location and vehicle sequences are presented in the form of Gantt charts or tables. Other information covers a wide range of information such as a number of cycles, travelled distance, working/idle time, utilization, downtime, breaks, operating costs or investment costs.

**Input data**

Input data for the simulation consists of the mine layout (Figure 2), availability of the vehicles, availability of the production areas and operational data of the vehicles. Data was collected from the mine and from the equipment manufacturer. Before model formulation, data analysis from different sources was required to ensure valid simulation.
results. The techniques used for data analysis vary depending on the amount of available data, when available data is too small or data is missing, and when there is no data available. In cases when more data is available, such data can be fitted to probability distributions that characterize the uncertainty and randomness of the operation. In some cases, when only a small amount of data is available, an attempt to fit into distributions may become inappropriate, and then the empirical distribution (actual data values), point estimate, uniform or triangular distribution can be applied. When no data is available due to, for example, the fact that the system being modelled does not yet exist, a subjective estimate can be made on the basis of guesses and assumptions. However, since the input data provide the driving force for the simulation, a careful check of the physical characteristics, limits or nature of the process should be performed by experts.

The input data (Table 1) shows the operational parameters for all analysed machines. The diesel LH 621D is the 21-tonne engine machine, while LH 625E, LH 514E, and LH 621E are the electric LHDs with 25-tonne, 14-tonne and 21-tonne bucket capacities respectively. The electric LH 621E is not yet released onto the market, and thus the data for this machine is not yet available. The simulations were then statistically analysed. As depicted in Table 2, scenarios one to twelve were selected to analyse the performance when only one type of the machine is used in all available time, and the remaining three scenarios were analysed when machines work only before blasting.

### Table 1. Operational parameters

<table>
<thead>
<tr>
<th>Machine type</th>
<th>Forward &amp; Reverse speed (km/h)</th>
<th>Acceleration Deceleration (ms²)</th>
<th>Bucket capacity (tonnes)</th>
<th>Drive power (kW)</th>
</tr>
</thead>
<tbody>
<tr>
<td>LH514E</td>
<td>23.5/20.5</td>
<td>0.1/0.05 -0.6</td>
<td>25</td>
<td>315</td>
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<tr>
<td>LH625E</td>
<td>18/16.0</td>
<td>0.1/0.05 -0.6</td>
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<td>283</td>
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<tr>
<td>LH621D</td>
<td>33.9/28</td>
<td>0.1/0.08 -0.6</td>
<td>14</td>
<td>132</td>
</tr>
<tr>
<td>LH621E</td>
<td>21/18</td>
<td>0.1/0.08 -0.6</td>
<td>21</td>
<td>345</td>
</tr>
</tbody>
</table>

### Scenarios

Several combinations of machines were analysed for fifteen different scenarios (Table 2). The machines were scheduled to work from 6:00 a.m. in the morning until 00:30 a.m. before blasting which comprises of 18.5 hours. After the blasting, the LHD was scheduled to work for 3.5 hours from 2:30 a.m. to 6:00 a.m. In the simulation, all breaks and delays were excluded to obtain the actual, effective machine working time. The simulation ended when the LHD operators finished removing all the available ore from the studied production areas. The results obtained from the simulation were then statistically analysed. As depicted in Table 2, scenarios one to twelve were selected to analyse the performance when only one type of the machine is used in all available time, and the remaining three scenarios were analysed when machines work only before blasting.

### Table 2. Scenarios

<table>
<thead>
<tr>
<th>Scenario number</th>
<th>Bucket capacity (tonnes)</th>
<th>LHD type</th>
<th>Scheduled time (hours)</th>
<th>No. of LHDs</th>
<th>LHD type</th>
<th>Scheduled time (hours)</th>
<th>No. of LHDs</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>25</td>
<td>Electric</td>
<td>18.5</td>
<td>1</td>
<td>Electric</td>
<td>3.5</td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>25</td>
<td>Electric</td>
<td>18.5</td>
<td>2</td>
<td>Electric</td>
<td>3.5</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>21</td>
<td>Electric</td>
<td>18.5</td>
<td>1</td>
<td>Electric</td>
<td>3.5</td>
<td>1</td>
</tr>
<tr>
<td>4</td>
<td>21</td>
<td>Electric</td>
<td>18.5</td>
<td>2</td>
<td>Electric</td>
<td>3.5</td>
<td>2</td>
</tr>
<tr>
<td>5</td>
<td>14</td>
<td>Electric</td>
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<td>1</td>
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<td>3.5</td>
<td>1</td>
</tr>
<tr>
<td>6</td>
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<td>2</td>
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<tr>
<td>7</td>
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<td>Diesel</td>
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<tr>
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<tr>
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</tr>
<tr>
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<td>3.5</td>
<td>1</td>
</tr>
<tr>
<td>11</td>
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<td>2</td>
<td>Electric</td>
<td>3.5</td>
<td>1</td>
</tr>
<tr>
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<td>3.5</td>
<td>2</td>
</tr>
<tr>
<td>13</td>
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<td>Electric</td>
<td>18.5</td>
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<td>Electric</td>
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<td>2</td>
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<tr>
<td>14</td>
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<td>Diesel</td>
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<td>Diesel</td>
<td>3.5</td>
<td>1</td>
</tr>
<tr>
<td>15</td>
<td>21</td>
<td>Electric</td>
<td>18.5</td>
<td>2</td>
<td>Electric</td>
<td>3.5</td>
<td>2</td>
</tr>
</tbody>
</table>

The first eight scenarios involved the analysis of the same type (electric or diesel) and the same number (one or two) of LHDs of all capacities operating for the duration of the whole scheduled time (6:00 a.m. to 00:30 a.m. and 2:30 a.m. to 6:00 a.m.). In all of the analysed scenarios, when two LHDs are employed, each machine will work on one side and use separate ore passes, meaning that there is no interaction of the LHD machines during production.

The 9th to 12th scenarios involved two electric or diesel LHDs working from 6 a.m. until 0:30 a.m. before blasting, and only one electric or diesel LHD working after blasting. The aim is to check the possibility of using two LHDs before blasting and one LHD after the blast. This is because of the short available time after the blast until the new shift machine schedule begins. Since after the blast the same LHD, which was working in a manual mode, will be prepared to work in semi-automated mode. The arrangement of these changes will delay the loading operation and reduce the LHD’s effective working time.
The last three scenarios were selected to check if it is worth loading from morning until the time before blasting and to include no loading activities after blasting. The scenarios involved two electric 25-tonne LHD machines, and two diesel 21-tonne LHD machines. The evaluation of these scenarios was based on the production rate, energy consumption and gas emissions. Summary of the scenario settings:

- The first setting involves one or two LHD machines operating from morning until before the blast, and one or two after the blast.
- The second setting involves two electric or diesel LHD machines operating from morning until before the blast, and only one electric or diesel LHD machine after the blast.
- The last setting involves two electric or diesel LHD machines from morning until before the blast, and no LHD machine after the blast.

Model validation and verification

Verification is the process of ensuring that the conceptual model design has been transformed into a computer model with sufficient accuracy. Validation is the process of ensuring that the model is sufficiently accurate for a certain purpose (Muller, 2014). The aim of verification and validation of the model is to ensure that the created model is accurate and represents the real system. There are several techniques that can be used for verification, such as testing the model logic, using debugging techniques, running the model under varying conditions, making logic flow diagrams, or building diagnostics into the model. The validation can be performed by using a degenerate test, testing internal validity, using an extreme condition test, comparing historical data, testing face validity, comparing output results with an actual system, or through a Turing test (Banks et al., 2010). A verified and validated simulation model should provide results that are very close to those seen in the actual operating system. In this study, verification was done using debugging techniques, animations check, model inspection, and by running the model under varying conditions. The debugging features were used to make sure that everything was running correctly before resuming execution. Simulation runs were initially conducted with the conceptual estimated size of equipment, storage facilities, and haulage systems structures. Initial results allow these parameters to be redefined and radically changed; changes involve extra programming but enhance the versatility of the program to conform with proposed mine logistics. The validation was done by using internal validity and by comparing the output from the model with the output from the real system.

5 Results and discussion

Results from the simulation runs

The simulation was first conducted for the base case scenario, which involves using one 25-tonne electric LHD machine and one 21-tonne semi-automated diesel LHD machine. The current loading and hauling operations modelled in AutoMod and SimMine simulation software for the base case scenario produce daily averages of 6,035 tonnes and 6,120 tonnes, respectively. This was seen to be close to the real-life production in the mine. Next, the scenarios shown in Table 2 were run to compare the output difference between AutoMod and SimMine models. The results are shown in in Figure 4.

When comparing AutoMod and SimMine simulation results (Figure 4), the highest variation in the production rates are observed in the 10th scenario (432 tonnes) whereas the lowest variation in the production rates is observed in the 1st scenario (similar results). The 1st and 5th scenario can be considered to achieve the smallest variation. These scenarios involve the use of one electric LHD machine from 6 a.m. until 0:30 a.m. before blasting, and one electric LHD machine after the blast. The 2nd, 6th, 10th, and 14th scenario can be considered to achieve the highest difference variation. These scenarios involve the use of two electric or diesel LHD machines from 6 a.m. until 0:30 a.m. before blasting, and one, two or none of the electric LHD machines after the blast.

The result shows that simulation using these two different simulation software results in similar production rates, thus suggesting that both software tools are capable of capturing similar results when simulating an underground loading environment. Different results (Figure 4) are observed due to the fact that the loading and dumping operations are continuous in nature and were simulated with randomly generated input parameters (based on the specified probability distributions).

Statistical and practical relevance for the AutoMod and SimMine models

The simulation analysis in this research was conducted using both AutoMod and SimMine simulation software. To increase the credibility and validate the model assumptions, output results and various settings were discussed and
improved with the help of experts who are knowledgeable about the studied system and processes. From experience and from observations the SimMine model and AutoMod model in this study provide similar results. Despite the similarity observed in these tools, it is still necessary to investigate the statistical relevance of the two separate simulations. In other words, to compare the relative performance of two or more system designs (Banks et al., 2010). For this the statistical technique correlated sampling, also known as Common Random Number (CRN), was used. This means that the same random numbers are used to simulate both alternative system designs. However, the same is not true when comparing between the relative performance of the AutoMod model and the relative performance of the SimMine model. In this case, independent sampling (different random numbers) is used. In order to evaluate the tools under the same conditions, a statistical hypothetical comparison using t-test was performed. Statistical significance tests tell us how likely it is that there will be differences between sample groups. The t-test was chosen because the variances of the two models were not known, and the sample size was less than 25 (Kanji, 1999; Cohen et al., 2013). The sample size of 16 from both models was used and then tested with a five percent level of significance. The purpose is to test the null hypothesis (HO) of the two population means, which is: HO: μ1 = μ2, where μ1 and μ2 stand for the mean daily production rate, energy consumption, gas emissions for diesel units, and time to mine the entire block from both models was used and then tested with a five percent level of significance. The purpose is to test the null hypothesis (HO) of the two population means, which is: HO: μ1 = μ2, where μ1 and μ2 stand for the mean daily production rate, energy consumption, gas emissions for diesel units, and time to mine the entire block from AutoMod and SimMine simulation models, respectively. The hypothesis is accepted if true; otherwise, it is rejected based on obtained P-Value. The P-value is the probability of getting the results (or more extreme results) given that the null hypothesis is true. The result of this test is shown in Table 3.

Table 3. Results of t-test for two population means: AutoMod and SimMine

<table>
<thead>
<tr>
<th>Parameter</th>
<th>AutoMod</th>
<th>SimMine</th>
<th>Statistical relevance</th>
<th>P-Value</th>
<th>Mean diff.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Production rate (tonnes/day)</td>
<td>8,927</td>
<td>8,907</td>
<td>8,907</td>
<td>2.772</td>
<td>8,907</td>
</tr>
<tr>
<td>Time to mine block (days)</td>
<td>757</td>
<td>756</td>
<td>756</td>
<td>302</td>
<td>300</td>
</tr>
</tbody>
</table>

To accept or reject the null hypothesis, the obtained P-values were compared to a 0.05 significance level. The condition is to reject the null hypothesis if the P-value is less than or equal to 0.05; otherwise, the null hypothesis is accepted. As can be seen in Table 3, the P-value is greater than 0.05, which means the null hypothesis is accepted for all analysed parameters. This means that there is no significant difference between the means of the daily production rate and time to mine for the entire mine block from the AutoMod and SimMine simulation models. This indicates that both simulation models produce statistically equivalent results.

Software performance

Currently, when making an evaluation the user tends to focus not only on the product design arena but also on the high software performance that is both short and reliable; therefore the results of the runtimes for AutoMod and SimMine models are presented in Figure 5. Each tool had a varying number of tools available for processing. There was also the possibility to run the model in real time in the built-in environment or run the model in windowless mode. In case of SimMine, the time was clocked at the end of each run and in case of the AutoMod, the time was read straight from the output report. To reduce the external influence of bias and make sure that the AutoMod and SimMine models both were run in a similar environment, the same computer settings were used. This meant disabling Wi-Fi connection and reducing the number of programs running in the background. Additionally, the simulation models were run multiple numbers of times to report differences. When changing the seed number, the difference in simulation runtimes ranged within tenths of seconds and the difference between multiple simulation runtimes did not vary by more than few seconds, thus an average runtime value was used.

Figure 5. Comparison of average simulation runtimes for AutoMod and SimMine models

The results in Figure 5 range between 9 to 21 seconds and 16 to 42 seconds for the AutoMod and SimMine respectively. An observation was made that using more than one LHD of the same type in the SimMine model created complexity that increased the simulation runtime. Even when one LHD was used it still took longer to finish loading operations. Examples of these occurrences are between 1st and 2nd scenario, 3rd and 4th scenario, 5th and 6th scenario and 7th and 8th scenario. However, it does not mean that bucket size and speed of the vehicle had not had an influence on the simulation runtime. For example, the 5th, 6th, and 11th scenario when running 14-tonne electric LHD produced the longest runtimes. In this case, the runtimes are the longest with reaching values of over 40 seconds of CPU time clock for scenario number 11. Overall the differences between the SimMine and the AutoMod runtimes do not significantly vary, however, the AutoMod might take a longer time to simulate the scenarios depending on the way the model is constructed/coded. Observations were made that running the AutoMod in windowless mode with no switched functions improved the simulation runtime without affecting the output results. What this means is that there is a room for further improvement of the coding. For example,
using the logic that invokes reporting the loaded material in
the graphical user interface (GUI), each time the vehicle
performs a cycle, consumes more time than using the logic
that invokes reporting at the end of the simulation run. This
line of code is not necessary for the windowless run and
therefore can be excluded from the windowless simulation
run. However, if necessary the line of the removed code can
be reconnected again later.

Simulation software evaluation
The AutoMod and SimMine software evaluation (Table 4)
is based on the modelling package evaluation table from
Albrecht, (2010). The software features presented in Table 4
differ extensively favouring the AutoMod due to its wide
range of additional facilities that are provided for modelling.
There are also certain categories that have to be further
elaborated such as the time to build the model, structure of
the system and system capabilities.

Table 4. AutoMod and SimMine package evaluation

<table>
<thead>
<tr>
<th>Category</th>
<th>Modelling package</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>AutoMod</td>
<td>SimMine</td>
</tr>
<tr>
<td><strong>A. Modeling environment</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1. Model development environment provided</td>
<td>yes</td>
<td>yes</td>
</tr>
<tr>
<td>2. Alternative model creation methods</td>
<td>yes</td>
<td>yes</td>
</tr>
<tr>
<td>3. Visual modelling (drag-and-drop)</td>
<td>yes</td>
<td>yes</td>
</tr>
<tr>
<td>4. Accessibility of model code</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td>5. Ease of use</td>
<td>no</td>
<td>yes</td>
</tr>
<tr>
<td>6. Animation</td>
<td>yes</td>
<td>yes</td>
</tr>
<tr>
<td>6. Virtual reality</td>
<td>no</td>
<td>no</td>
</tr>
<tr>
<td><strong>B. Model documentation and structure</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1. Operating system independence</td>
<td>no</td>
<td>no</td>
</tr>
<tr>
<td>2. Model chaining (Linking outputs from different models)</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td>3. Efficient translation to executable form</td>
<td>yes</td>
<td>yes</td>
</tr>
<tr>
<td>4. Alternative worldviews</td>
<td>no</td>
<td>no</td>
</tr>
<tr>
<td><strong>C. Verification and validation</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1. Facilitates the verification of simulation models</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td>2. Provides interactive model checking</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td>3. Trace file provided</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td><strong>D. Experimentation facilities</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1. Automatic batch run</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td>2. Warm-up periods</td>
<td>yes</td>
<td>no</td>
</tr>
<tr>
<td>3. Independent replications of experiments</td>
<td>yes</td>
<td>no</td>
</tr>
</tbody>
</table>

4. Re-initialization (initializing again) | yes | no |
5. Speed adjustment | yes | no |
6. Breakpoints (intentional stopping) | yes | no |
7. Experimental design capability | yes | no |
8. Automatic determination of run length | yes | yes |

**E. Statistical facilities**
1. Alternative statistical distributions | yes | yes |
2. User-defined distributions | yes | no |
3. Random number streams | yes | yes |
4. User-specified seeds of random number streams | yes | yes |
5. Distribution fitting | no | no |
6. Goodness-of-fit tests | no | no |
7. Output data analysis | yes | yes |

**F. User support**
1. User manuals | yes | yes |
2. Technical and promotional information (e-mail, internet, discussion groups) | yes | yes |
3. Lecturer’s guide for educational licenses | yes | yes |
4. Tutorial | yes | no |
5. Training course | yes | yes |
6. Package maintenance | yes | yes |
7. Portability | no | no |
8. File conversion | no | no |

**G. Financial and technical features**
1. Package cost includes life cycle maintenance costs | yes | yes |
2. Ease of installation | yes | yes |
3. Low hardware/system requirements | yes | yes |
4. Frequent and comprehensive updates | yes | yes |

Depending on the size and complexity of the model, it
would usually take much longer for an experienced user to
build the AutoMod model than the SimMine model. This is
excluding the time needed to gather the input data and run
any other tests such as verification and validation tests. In
AutoMod, it is possible to code all the parameters and
functions however this increases the development time and
level of necessary skills required for coding. For example,
in case of the AutoMod, the degree of details controllable
within the model depends on the user (how and what is
included in the model), whereas in case of the SimMine
there are already many parameters that are useful and it does
not require extra time spent on programming the input and
output parameters. The same applies to the readability of
model code. Therefore, the degree of details controllable within the model is higher for AutoMod and readability depends on the user as opposed to SimMine tool. However, the user has an advantage when it comes to SimMine since it already has a mining infrastructure and many mining-related parameters already encoded in the software.

The SimMine software produces mining-related reports providing additional information such as development time, production outputs or cost-related information. For the AutoMod software, an extra coding is required to generate the output in the appropriate form, otherwise, the report comes in the same output format used in the AutoMod. However, in case of the SimMine, the files can be a database (.mdb extension) or AutoCAD files. In case of the AutoMod, the imported AutoCAD layout (if without an extension that transforms the AutoCAD layout to suitable package) requires creating additional paths on top of the imported layout.

The AutoMod provides interactive checking control in form of the internal run debugger (IRC) that is a perfect tracer of the bugs and functionality of the model created. This means the user can go line by line and see how the program handles and changes variables of the created code. There is a possibility to do the same in the SimMine, however, it is not available for the normal user and restricted to the development package.

When it comes to random number generation, the AutoMod uses Combined Multiple Recursive Generator (CMRG) to generate a reproducible sequence of random numbers of length up to 2^191 distributed continuously and uniformly on the open interval 0 to 1 (Banks, 2004; Banks et al., 2010). Whenever the random numbers are required the AutoMod software draws a sample from the sequence of numbers. The sequence of random numbers is split into streams of length up to 2^127 (each stream is split into 2^51 random number sets, each with a length of 2^76). These random numbers, if necessary, are then converted to the desired statistical distribution or random variate (Banks, 2004). The random variate is a variable generated from uniformly distributed numbers, for example, the random variate represents the random event in the model (Banks, 2004). Several probability distributions are available in AutoMod such as continuous distribution (user-defined based on the cumulative frequency), exponential distribution, gamma distribution, lognormal distribution, oneof distribution (user-defined based on the frequency with possibility to mix two distributions together), normal distribution, triangular distribution, uniform distribution, Weibull distribution, nextof distribution (based on round-robin manner) and constant distribution (Brook Automation Inc., 2003). Additionally, there is a possibility of encoding the user-specified distribution in the syntax. In case of the SimMine, the generation of the random numbers is performed by use of a linear congruential generator (LCG). The LCG is the most widely used technique for generating the random numbers where the initial seed for a linear congruential random-number generator is the integer value that initializes the random-number sequence (Banks et al., 2010). Whenever the random numbers are required, the SimMine software draws a sample from the sequence of numbers and converts them to the desired statistical distribution or random variate. The probability distributions available in SimMine are triangular distribution, normal distribution, uniform distribution, lognormal distribution, exponential distribution or constant distribution.

There are also differences in software capabilities. For example, let’s suppose that two trucks enter the loading area. The first truck requires two buckets to fill up, whereas the second requires only one bucket to do so. However, there is only one loader at the moment. In this case, let’s consider three alternatives. First: neither of the trucks claims the loader. Second: the first truck (two buckets to fill) claims the loader and waits for another load. Thirdly: the second truck (one load to fill up) claims the loader and continues to transport the ore further downstream, releasing the location for the first truck. Now, in case of the SimMine, the possibilities are to follow the First-In-First-Out (FIFO) selection in which the first truck would claim the loader or follow the Last-In-First-Out (LIFO) selection in which the last truck that had entered the waiting list would claim the loader. When it comes to AutoMod the flexibility is higher and offers the possibility not only to claim the loader according to FIFO rule or LIFO rule but also any other method that can be implemented as a code. For example, the selection can be adjusted according to any dispatching rule based on the processing time, due date, number of operations, machine priority, according to a particular event or any other user-defined rule.

Software recommendations

In the case of the AutoMod software, the model uses different processes in the system to interact with each other. This is achieved by focusing on the job-driven model, whereas, in case of the SimMine, the model is already there with preinstalled functionalities and options related to underground mining operations. From that perspective, even though the AutoMod software has a better functionality and flexibility in building a new mining model it is still considered as a time-consuming and complex task for the user. Furthermore, the AutoMod is not mine-related software, therefore the user has to understand and decide which parameters to use as input and output values. When it comes to SimMine software, there is room for improvement since the user may require extra functionality in the software. For additional parameters, there is a development group at SimMine that could incorporate the mine-specific changes/parameters depending on the customer needs.
However, it is important to use the software program which is able to imitate the real-world environment to an acceptable level of details depending on the requirements and objectives of the model, since otherwise the simulation results can differ and mislead in providing the information to the user. Nonetheless, the SimMine software is recommended for easy and fast modelling, whereas for more enhanced and detailed simulations the AutoMod is recommended, with its wider palette of software features and facilities provided.

6 Conclusions
This research compares AutoMod and SimMine models by simulating the Kiirunavaara mine with different LHD machines, with different sizes, operating between the production draw points and a group of ore passes. The analysis was based on the daily production and energy consumption and gas emissions using discrete event simulation. The following conclusions have been reached:

- AutoMod and SimMine models achieve adequate and statistically equivalent production results.
- Software features differ extensively favouring the AutoMod due to its wide range of additional facilities that are provided for modelling.
- In AutoMod the degree of details controllable within the model and readability depends on the user. This means that all the parameters and functions have to be coded by the user increasing the development time and level of the necessary skills required for coding.
- In SimMine the parameters and functions are preinstalled providing an easy and fast access in developing a simulation model related to mining operations.
- SimMine is recommended for easy and fast modelling, whereas for more enhanced and detailed simulations the AutoMod is recommended, with its wider palette of software features and facilities provided.

Future study may involve the detailed evaluation of the capital and operating costs of the proposed loading equipment models but also enable further enhancement towards adaptive control. Future work should incorporate the adaptive control and auto-tuning that can be easily implemented in the typical process control computer. It should have the ability to present information to the user/operator at the right time from the system including the directions and consequences of the selection being made. Implementation of process control units would help to reduce the variability, increase the efficiency and maintain the production targets.

Conflicts of Interest: The authors declare no conflict of interest.

References


Analyzing energy consumption and gas emissions of loading equipment in underground mining

INTRODUCTION

Loading equipment for underground hard-rock mining operations has historically been dominated by load-haul-dump machines (LHDs; de la Vergne, 2003) powered by diesel or electricity. Diesel LHDs have been used for material transportation in mining since the 1960s. Diesel LHD infrastructure consists of storage tanks, pumps, and pipes for fuel supply, including fueling stations throughout the mine. Diesel LHDs are versatile and can move easily from one location to another, but they have high operating costs associated with fuel consumption, consumables, regular checks, and the ventilation required to mitigate the heat and emissions they generate (Novak, Gregg, & Hartman, 1987; Chadwick, 1996; Miller & Barnes, 2002). Electric LHDs, not commonly used in underground mines (Paraszczak, Laflamme, & Fytas, 2013), have the advantages of producing less noise, less heat, and no exhaust gases (Chadwick, 1996; Paraszczak et al., 2013; Paterson & Knights, 2013; Paraszczak, Svedlund, Fytas, & Laflamme, 2014). They can be powered by batteries, overhead electric lines, or trailing cables. Batteries offer the highest flexibility, but battery-powered vehicles are heavy and must be regularly recharged. Overhead power lines might be feasible when routes remain constant for an extended time but are impractical when a high degree of maneuverability is necessary. Currently, the most viable way to power electric LHDs is with a trailing cable plugged into the mine’s electrical infrastructure. Powering with an electric cable reduces the versatility of LHDs due to limited haul range, relocation issues, and restricted movement, presenting problems with cable faults and wear. Electric LHDs, however, have obvious advantages in mass mining methods such as block and sublevel caving, where relocation delays are less critical because operations occur along a main path for an extended time period (Paterson & Knights, 2013).

In the coming decades, underground mines can be expected to operate at even greater depths, with concomitant higher energy use due to longer haul distances and ventilation to mitigate geothermal heat and exhaust gas...
emissions from working equipment. The expected increased costs make the choice of LHDs an essential factor in cost control. The demand for electric LHDs can also be expected to increase as mining companies seek to address high ventilation costs, higher fuel prices, and stricter regulations on emissions (Miller & Barnes, 2002; Salama, Nehring, & Greberg, 2014).

This study, conducted at an existing underground mine, compared the operational performance of diesel and electric LHDs. The AutoModTM discrete event simulation tool was used to obtain the equipment productivity and operating times, and the results were used to estimate energy and gas emissions. The study analyzed seven diesel and seven electric LHDs with similar bucket capacities.

Energy costs, consumption, and gas emissions

Diesel fuel is the most common energy source consumed in mining operations (Kecojevic & Komljenovic, 2010). Other sources of energy such as natural gas and gasoline are not common in mining operations. As shown in Figure 1, the price of crude oil was approximately US$108 per barrel in the world market in 2012, the year of this reference case. The case represents current judgments regarding exploration and development costs and accessibility of oil resources. Under the assumption that the Organization of the Petroleum Exporting Countries producers will maintain their market share and will schedule investments in incremental production capacity, the price of oil is projected to be US$102/barrel in 2020 and reach US$148/barrel in 2040 (United States Energy Information Administration, 2014). Continued increases in global energy demand, future increases in energy prices, increased mine depths and associated costs, and environmental issues require that mining companies investigate the implications of increased operational costs on their mine plan. Haulage methods that result in reaching desired production objectives at lower costs will be very significant (Kecojevic & Komljenovic, 2010).

Figure 1. World oil prices forecast (United States Energy Information Administration, 2014)

The energy consumption of diesel or electric LHDs can be expressed in litres of fuel or kWh of electricity per tonne hauled, respectively. Measuring the amount of energy consumed is most accurate on site; however, this is very expensive because it requires the continuous monitoring of every piece of equipment in operation. In the absence of field measurements, consumption can be estimated based on available energy models such as the energy savings measurement guide, energy efficiency opportunities, energy mass balance, and mining industry energy bandwidth (Bise, 2003). Most of these models use simplified methods to estimate diesel fuel and electricity consumption (Tatiya, 2013) and are less expensive; they are also less accurate than on-site measurement. The methods depend on load factor, engine capacity, road conditions, and the length of time the equipment is used (Caterpillar, 2009). Equation 1 can be used to estimate diesel fuel consumption during loading and dumping (Kecojevic & Komljenovic, 2010). For electric LHDs, the estimate is based on the motor input power and the length of time the equipment is used:

\[
FC = \frac{K \times GHP \times LF}{KPL} \tag{1}
\]

where \(FC\) is the fuel consumed (in L/machine hour), \(K\) is kilograms of fuel used per brake horsepower per hour, \(GHP\) is the gross engine horsepower at governed engine revolution, \(LF\) is the load factor (in %), and \(KPL\) is the weight of fuel (in kg/L).

Equation 2 is used to convert the fuel consumed by diesel LHDs into kilowatt hours to compare with electric LHDs:

\[
ED = \frac{FC \times 38.6}{3.6} \tag{2}
\]

where \(ED\) is the energy consumed by diesel equipment (in kWh), the calorific value of 38.6 MJ/L is the amount of heat released by a fuel when combusted, and 3.6 MJ is the heat used to produce 1 kWh.

Equation 3 is used to estimate the energy consumption in kWh per loading cycle, \(E\), from the loading point to the dumping point and back to the loading point, for diesel and electric LHDs:

\[
E = \frac{TR \times g \times [(V_W + B_C) \times V_L + (V_W \times V_E)]}{1000} \times t \tag{3}
\]

where \(TR\) is the total resistance (in t), \(g\) is the acceleration due to gravity (in m/s\(^2\)), \(V_W\) is the gross vehicle weight (in t), \(B_C\) is bucket capacity (in t), \(V_L\) is the vehicle speed when loaded (in m/s), \(V_E\) is the vehicle speed when empty (in m/s), and \(t\) is the time taken to transport material for dumping and return to the loading point (in h).
The CO₂ emissions are estimated based on the combustion process of fixed carbon restrained in a volume of diesel fuel. These emissions occur due to incomplete combustion in the diesel engine and impurities in the fuel (Pellegrino, Quin, Thekdi, & Justiano, 2005). The emissions are calculated based on the diesel conversion factors published by the United States Environmental Protection Agency (2005). Equation 4 is used to express the amount of CO₂ emitted (in t/h) from diesel equipment (Bogunovic & Kecojevic, 2009):

\[
\text{\( CO_2 = FC \times CC \times 10^{-6} \times 0.99 \times \left[ \frac{44}{12} \right] \)}
\]

where CC is the carbon content of the fuel (in g/L), 0.99 is the oxidation factor (i.e., 99% of fuel burns out and 1% is unoxidized), and 44/12 is the molecular weight ratio of CO₂ to carbon.

**Discrete event simulation**

Discrete event simulation was used to estimate the LHD operating times, which were used as input for the fuel consumption estimation. Discrete event simulation is a technique to model stochastic, dynamic systems in which changes in the state of the event variables occur at discrete points in time. This technique is very suitable for modelling complex systems (Law & Kelton, 1991) and has been increasingly used for systems in open pit and underground mining operations (Banks, Carson, Nelson, & Nicol, 2010). Simulation can model operations to analyze, optimize, improve, and plan systems with regard to, for instance, fleet requirements, the flow of hauling machines, and mine planning.

Among the various simulation tools available on the market, AutoMod was selected for this study. AutoMod is used to create discrete event simulation models of various applications such as assembly lines, manufacturing systems, and mining operations. It is used to analyze and optimize alternative system designs and can predict results for existing and future systems. AutoMod provides advanced debugging and trace facilities that enable errors and flaws to be easily traced. The tool is equipped with concurrent 3D graphics and a comprehensive set of templates and objects for modelling different applications (Muller, 2011). It also creates performance reports with statistics and 3D animation, providing a realistic and statistically accurate view of the system. AutoMod is also equipped with the Autostat™ feature, which reduces the time required for experimentation and analysis (Muller, 2011).

**CASE STUDY**

The study mine uses the sublevel caving mining method, in which development drifts are initially made. Next, the ore passes are drilled, extending vertically from the current mining area to the bottom of a new mining area, where transportation levels exist. Horizontal sublevels are created, and access routes over the length of the orebody within a sublevel are developed. The self-supported horizontal crosscuts are drilled through the orebody perpendicular to the access routes.

In this mine, spaces between sublevels are approximately 28.5 m and the crosscuts are 25 m apart. At the crosscuts, near-vertical rings of holes are drilled in a fan-shaped pattern. Each ring contains approximately 10,000 t of ore and waste. The ore is recovered on each sublevel, starting with overlying sublevels and proceeding downward. In each sublevel, the ore is removed from the hangwall to the forefront of the footwall. As the ore is recovered from a sublevel, the hangwall will collapse according to design and will cover the mining area with broken waste rock.

As shown in Figure 2, the mine is divided into 10 main production areas or blocks, extending from the uppermost mining level down to the current main level. Each block consists of several sublevels and is approximately 400–500 m long, with its own group of ore passes located at the centre of the production area and extending down to the main haulage level. At the time of the study (2012), the mine was producing approximately 27 Mt/y of crude ore. By 2015, it is estimated to produce 35 Mt/y from all 10 blocks. To reach this target, the mining company plans to increase the loading capacity from each block to 10,000 t/d.

![Figure 2. Mining blocks and haulage system structures](image-url)

The mine uses one 25 t capacity electric LHD that operates from 6:00 am to 10:00 pm and one 21 t capacity diesel LHD (called “7D” in this study) that operates from 10:00 pm to 6:00 am. The average daily production of these machines was 60% of the future planned production. These LHDs load the ore from draw points within each production drift and transport it to the ore passes. Large trains operating on the main level transport the ore from ore passes to a crusher, which crushes the ore for hoisting to the surface through a series of vertical shafts. Once mining has begun in a block, continuous production must be maintained until all available ore is removed (according to the mine restrictions).
For this study, the energy consumption and gas emissions for the LHD 7D and the other diesel LHDs shown in Table 1 were compared to electric LHDs of similar bucket size. The 25 t electric LHD was not considered for this analysis because there was no available diesel counterpart with similar bucket size. The LHD 7D data were collected from the mine, whereas data for all other LHDs were obtained from the manufacturers. The energy consumption was estimated based on simulation results and empirical formulas. Simulation was used to estimate LHD operating times; the simulation results then served inputs to estimate energy consumption and gas emissions. The combined use of simulation and analytical calculations enables a more accurate estimate of energy consumption and emissions than the single analytical method traditionally used. The single analytical method can be successfully applied when a system involves fewer random activities. During loading operations, randomness is present in many situations, such as variability in loading and dump times, tramming distances, and times to clear boulders at the ore passes. When a system contains uncertainty and random behaviour, the discrete event simulation approach offers the advantage of more accurate accounting for real-world uncertainty and operations diversity.

### Simulation model formulation

The simulation method for this research was chosen because the mining system includes many uncertain operational elements and random behaviour. The discrete event simulation approach was considered the most appropriate technique to study such operations and can accurately account for the real-world uncertainty and diverse variables of the interdependent components in mining operations. A detailed, accurate simulation model for large, complex systems requires a large dataset, statistical distributions of the data, and a careful choice of simulation software. In this study, the simulation model was developed using AutoMod software, which consists of material movement systems that allow users to model manual and automated equipment such as LHDs with a high degree of accuracy (Muller, 2011). When system elements such as paths and stations are known, then operating parameters such as speed, turning speed, acceleration, and deceleration can be defined in the movement system (Banks, 2004).

#### Input data

Table 1 shows the kinematics data for the diesel and electric LHDs. Fourteen types of LHDs were analyzed (seven diesel and seven electric) with similar bucket capacities. The electric LHD 7E was not yet on the market; therefore, data for this machine were estimated based on assumptions from the manufacturer’s experts. The average speed for all LHDs operating in third gear was used for the analysis.

Table 2 shows the technical parameters used for energy consumption calculations. The calculated energy was the energy required to drive the LHD. The operating weight of both types of LHD increased in proportion to machine size. The diesel LHDs had higher drive power compared to the electric LHDs with similar bucket sizes. Normally, the fuel or electricity consumption rate of a vehicle is determined based on the machine loading rate, vehicle efficiency, road conditions, and other factors.

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**Table 1. Kinematics parameters for diesel and electric LHDs (LHDs: load-haul-dump machines)**

<table>
<thead>
<tr>
<th>LHD type</th>
<th>Bucket capacity (m$^3$)</th>
<th>Speed (empty) (m/s)</th>
<th>Speed (loaded) (m/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diesel</td>
<td>Electric</td>
<td>Diesel</td>
<td>Electric</td>
</tr>
<tr>
<td>1D</td>
<td>1E</td>
<td>1.5</td>
<td>6.67</td>
</tr>
<tr>
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<td>2E</td>
<td>1.6</td>
<td>5.42</td>
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<tr>
<td>3D</td>
<td>3E</td>
<td>3.0</td>
<td>7.22</td>
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<td>4E</td>
<td>4.5</td>
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<td>5E</td>
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<tr>
<td>6D</td>
<td>6E</td>
<td>5.4</td>
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<tr>
<td>7D</td>
<td>7E</td>
<td>9.0</td>
<td>7.08</td>
</tr>
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</table>

**Table 2. Technical parameters for diesel and electric LHDs (LHDs: load-haul-dump machines)**

<table>
<thead>
<tr>
<th>LHD type</th>
<th>Vehicle operating weight (t)</th>
<th>Drive power (kW)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diesel</td>
<td>Electric</td>
<td></td>
</tr>
<tr>
<td>1D</td>
<td>1E</td>
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<td>3D</td>
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<td>19.6</td>
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<td>4D</td>
<td>4E</td>
<td>24.3</td>
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<td>5E</td>
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<td>6D</td>
<td>6E</td>
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<td>7D</td>
<td>7E</td>
<td>56.8</td>
</tr>
</tbody>
</table>
Analyzing energy consumption and gas emissions of loading equipment in underground mining

gradient and surface features, load factors, and the LHD operating time. The load factor was obtained by taking the ratio of the power the LHD was using (input power) to the power required when the LHD operated at its rated capacity. A strip chart recorder was used to measure the average input power for diesel LHD 7D in the mine, which was 274 kW. As Table 2 indicates, this LHD had a drive power capacity of 350 kW, which gives a load factor of 78%. The same value of load factor was assumed for all analyzed diesel and electric LHDs.

The resisting force was estimated based on the tire rolling resistance because the LHDs were hauling material from the production drifts to the ore passes near the main drift, with a grade resistance of zero. The simulation results were used to obtain the LHD operating times, after excluding time lost for maintenance, production area availability, meal breaks, shift changes, and interference with other mining activities, such as drilling, charging, and ore pass maintenance. The production area availability due to other mining activities was 80%, which was applied for all LHDs. For this study, data for breakdown, scheduled maintenance, and equipment downtime were available for the diesel LHD 7D. No operating characteristics data were available for the other analyzed diesel and electric LHDs, which excluded a comparison of availability and utilization for diesel and electric LHDs. In the simulation, a machine availability of 80% was used for all LHDs. The value was based on the mean time between failure of 59.9 h and mean time to repair of 14.3 h for the diesel LHD 7D (Gustafson, Schunnesson, Galar, & Kumar, 2013). Using equations 1–4, energy consumption and gas emissions were calculated.

Model logic

Figure 3 shows the section of the mine block used for the analysis. The block consists of 17 production drifts, each approximately 100 m long, and four ore passes located close to the main drift. A 3 m blasting round on each production drift begins in sequence at the hangingwall, using an upward rise to provide free face, and then retreats towards the footwall. Mucking out by LHDs continues until the waste dilution reaches the set limit. Approximately 10,000 t of ore is excavated from one blasted ring, which means the total ore to be mined from this block is approximately 6.17 million tonnes. Loading is assumed to start from the first drift on the left side of the block and to continue to the next drift until the last drift is finished. This procedure is repeated until the

Figure 3. Section of a single mine block used for analysis
entire production area is mined out. The flowchart of an LHD loading operation is shown in Figure 4.

In Figure 3, two parking areas are included as waiting areas for LHDs during breaks. In this case, only one LHD is operating at a time. The machine uses two ore passes when operating on the left side of the production area and the remaining two ore passes when operating on the right. Which ore pass is used depends on the buffer capacity and boulder frequency. If the buffer capacity limit is reached, then the LHD moves to the next ore pass. If a boulder enters the ore pass, the LHD waits for clearance. For this study, this waiting time is triangularly distributed as a minimum of 1 min, an average of 5 min, and a maximum of 30 min. If the boulder clearance is not finished during this time, the LHD moves to the next ore pass. If the rock breaker is working to clear boulders when the next ore pass buffer capacity is reached, the LHD waits for the rock breaker to finish removing the boulders. The initial simulation model consists of the diesel LHD 7D, which is currently in operation in the mine. The simulation model was adjusted and all analyzed LHDs were simulated, and then productivity, energy consumption, and gas emissions were compared.

**Model validation and verification**

Verification ensures that a conceptual model design has been transformed into a computer model with sufficient accuracy; validation ensures the model is sufficiently accurate for a certain purpose (Sargent, 2003). Both processes ensure that the model is accurate and represents the real system. Verification techniques include testing the model logic by using debugging techniques, running the model under varying conditions, making logic flow diagrams, and building diagnostics into the model (Muller, 2011). Validation techniques include using a degenerate test, testing internal validity, using an extreme condition test, comparing with historical data, testing face validity, comparing output results with the actual system, or using a Turing test (Banks et al., 2010). A verified and validated simulation model could provide results very close to the actual operating system.

For this study, verification was achieved by using debugging techniques, undergoing an animations check and a model inspection from the specialists, and running the model under varying conditions. The debugging features were used to ensure everything was running correctly before resuming execution. Simulation runs were initially conducted with a conceptual estimated size of equipment, storage facilities, and haulage systems structures. Initial results allowed these parameters to be redefined and radically changed, which involved extra programming that enhanced the program’s versatility to conform to proposed mine logistics. To achieve validation, internal validity was used and compared model and real-system outputs.

**RESULTS AND DISCUSSION**

**Productivity comparison**

The simulation was first conducted for the diesel LHD 7D, which was currently operating in the mine. After the first simulation, the model was adjusted and applied to the other LHDs to compare hourly production rates, hourly energy consumption, and CO₂ emissions.

Figure 5 shows the hourly productivity of the analyzed LHDs. The diesel LHDs were more productive than the electric LHDs of similar bucket size because, for the case analyzed, diesel

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**Figure 4. Flowchart of LHD loading operation (LHD: load-haul-dump machine)**
LHDs travel faster with higher versatility, resulting in shorter cycle times than electric LHDs. Hourly production increased in proportion to bucket capacity for both diesel and electric LHDs. Despite the greater maneuverability and thus shorter cycle times of the smaller bucket machines, the larger bucket machines were able to dump more ore.

Based on the productivity results for all analyzed machines, a single unit would not be enough to achieve the future target of 10,000 t/d; therefore, increased production capacity will demand additional electric or diesel LHDs to be in operation and, in turn, multiple drifting and stoping operations. The production area used for the analysis can accommodate up to two LHDs operating simultaneously. During dumping, the machine operating on the left side of the mine block would use the first two ore passes, and the LHD operating on the right would dump material in the ore passes on the right side. If more than two machines were used, the performance evaluations should include delays for queuing and traffic. The option to add more equipment would need to be justified financially because the cost of additional equipment might not be recuperated by extra production.

**Energy consumption and gas emissions**

Recognizing that transporting ore is one of the mining industry’s most energy-intensive activities, with energy and ventilation contributing greatly to operation costs, energy consumption and CO$_2$ emissions were analyzed to address the mitigation of underground diesel emissions for underground air quality and safety regulation compliance.

Energy consumption of the diesel LHD 7D in the mine during its operation was measured and simulation and the empirical formulas were used to estimate energy consumption of the other analyzed machines. Simulation was used to obtain each LHD’s operating time, which was then used to calculate use. In all cases, the LHD utilizations were calculated according to equation 5:

\[
\text{Utilization} = \frac{\text{Operating time}}{\text{Total simulation time}} \times 100\% 
\]  

Table 3 shows the fuel or electricity consumption for the diesel and electric LHDs. Based on 2014 prices, hourly costs were higher for diesel fuel than electricity. For example, in 1 h of operation, the diesel LHD 7D consumed 42.3 L of fuel; its electric counterpart consumed 306 kWh of electricity. Based on an assumed fuel price of US$1.79/L and electricity price of US$0.12/kWh (International Energy Agency, 2014), this consumption equates to an hourly fuel cost of US$75.80 and an electricity cost of US$36.80. This gives US$0.24/t and US$0.18/t for diesel and electric LHDs, respectively. The cost increased in proportion to bucket size for both machine types. The diesel fuel consumption was converted to kWh for comparison purposes with electric units.
Energy consumption was higher for diesel than electric LHDs. For example, in 1 h of operation, the diesel LHD 7D consumed 415 kWh of energy, whereas its electric counterpart consumed 306 kWh (Figure 6). Electric LHDs consume less energy because they are equipped with lower powered motors compared to diesel machines. Electric engines are also much more energy efficient than combustion engines of similar size. The diesel LHD with a bucket capacity of 4.6 m$^3$ had 220 kW of power, whereas the electric LHD of the same capacity had only 132 kW.

The energy required to drive an LHD is proportional to its speed. For the case analyzed, diesel machines travel faster than electric ones in the same gear because the diesel engine has a torque converter with a low offset ratio. This feature means diesel machines have higher speed and energy consumption compared to their electric counterparts. The difference in consumption rates increases in proportion to bucket size because larger buckets weigh more.

For example, a diesel machine with a 1.5 m$^3$ bucket weighs 8.7 t (with an empty bucket), and one with a 9 m$^3$ bucket weighs 56.8 t.

Figure 7 shows the hourly CO$_2$ emission and fuel consumption for each diesel LHD. The LHD 1D consumed 9.89 L of diesel and emitted 26.5 kg of CO$_2$ in 1 hr, whereas the LHD 7D consumed 42.3 L of diesel and emitted 113.5 kg of CO$_2$.

Analysis of these results demonstrated that energy costs were higher for the diesel LHDs than their electric counterparts. On average and based on 2014 energy prices for the case analyzed, the energy costs for diesel and electric LHDs are US$0.24/t and US$0.07/t, respectively. Diesel equipment also had higher heat and CO$_2$ emissions. Based on the ratio of emissions and the amount of fuel used, the diesel LHDs emitted 2.68 kg of CO$_2$ gas for every litre of diesel fuel. In comparison, electric LHDs had zero CO$_2$ emissions, consumed less energy, and produced less heat; however,
electric LHDs have higher initial costs because of the additional infrastructure required (transformer boxes and substations). This additional infrastructure also requires time to relocate when electric machines are moved from one production area to another, causing potential production delays.

These results indicate that using electric rather than diesel machines can reduce costs, especially in an environment of increasing energy prices and deep mining with sub-level caving methods. When other mining methods are used, a detailed analysis might be required for the operational performance of electric LHDs.

CONCLUSIONS
In this study conducted at an existing underground mine, the energy consumption and gas emissions of seven diesel and seven electric LHDs with similar bucket capacities were analyzed and compared. The results indicated the following:

• Diesel LHDs are mobile and versatile, and their operational flexibility makes them more productive than electric LHDs of similar bucket sizes.
• Based on current energy prices, the energy costs for diesel and electric LHDs are US$0.24/t and US$0.07/t, respectively.
• Diesel LHDs emitted 2.68 kg of CO₂ gas per litre of fuel. Emissions from diesel LHDs require large volumes of air for ventilation, which generates additional costs.
• Electric LHDs have zero CO₂ emissions and produce less heat than diesel units.
• Minimizing the use of diesel LHDs will reduce the need for ventilation to mitigate engine heat and emissions; maximizing the use of electric LHDs will reduce energy costs.
• Using simulations, valuations, and verification processes to develop and upgrade models for analyzing, planning, and operating is very beneficial for investigating and solving operational issues such as the one presented in this study.

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The effects of ore pass loss on loading, hauling and dumping operations and production rates in sublevel caving mine

The effects of orepass loss on loading, hauling, and dumping operations and production rates in a sublevel caving mine

by B. Skawina*, J. Greberg*, A. Salama†, and A. Gustafson*

Introduction

Underground mining of iron ore takes place through a number of different unit operations, such as extraction, transport, storage, or sizing of the material. In caving and sublevel stoping mines, orepass systems are commonly used to transfer broken rock between different levels (Hambley, 1987). Orepass structures are exposed to the risk of failure, leading to a need for redevelopment or rehabilitation, thus creating a major problem for the mining operations (Brummer, 1998). An orepass can be lost as a result of operational or structural failure, leading to either short-term or long-term loss. This paper focuses on the effects on the LHD operations and related production rates following a long-term loss of one or more of the orepasses in an underground mine using sublevel caving. In this study, long-term is defined as the time that it takes to mine out a whole production area. The effects of increasing the LHD fleet size in order to maintain production are also analysed. Different scenarios have been used to analyse different production areas, different numbers of vehicles, and different numbers of available orepasses.

LKAB Malmberget mine

The LKAB’s Malmberget mine, located in northern Sweden, is an iron ore mine using the sublevel caving method to extract ore. Underground operations in Malmberget began in 1920 (Quinteiro and Hedström, 2001). Sublevel caving was introduced as the main mining method in the 1970s, and since then main haulage levels have been built on levels 600, 815, 1,000, and 1,250 metres. The ore reserves are distributed over 20 large and small orebodies, 14 of which are currently in production. The orebodies are spread over an area 5 km long and 2.5 km wide.

A total of three production areas (A, B, and C) are analysed in this study (Figures 1–3). The figures depict the orebody outlines, the orepasses, and the drifts used for hauling the ore from the face to one of the orepass locations. These production areas were chosen as together they will represent a substantial share of the total mine production in the future. In each of the production areas studied, there are four orepasses. The orepasses are around 300 m long, have a diameter of around 3 m, and dip around 60 degrees. The drifts are approximately 5 m high and 6.5 m wide.

The current loading practice in Malmberget mine is based on the use of manually operated diesel LHDs with a 21 t bucket capacity. In the smaller orebodies, there is usually one LHD in operation. In larger production areas, two LHDs may be used simultaneously. The number of orepasses in each orebody varies from one to four, depending on the size of the production area. The sequencing of drifts to be extracted is mainly based on factors related to rock stress, the ancillary activities in the production drifts, distances to the orepasses (depending on the production rate requirement), and the amount of ore left in each of the production drifts. The ore is

Synopsis

Orepass failure is a well-known problem in deep mines, and the risk of losing an orepass is associated with severe production disturbances. In the near future, one possible scenario in the Loussavaara Kiirunavaara Aktiebolag (LKAB) Malmberget mine is to concentrate the mining operation in fewer, but larger, production areas. In this paper we evaluate the effects of orepass loss on loading, hauling, and dumping operations and production rates using discrete event simulation, by simulating part of the Malmberget mine loading and hauling system under different environmental and operational constraints.

Keywords

LHDs, orepass loss, production simulation, underground iron ore mining.
The effects of orepass loss on loading, hauling, and dumping operations and production rates

Problem definition
One concern related to the use of orepass structures is their ability to convey or store material without unnecessary stoppages (Beus et al., 2001) that could result in unwanted disturbances in production. These stoppages are associated with hang-ups, piping, blockages, orepass degradation, and wall stability (Beus et al., 2001; Hadjigeorgiou and Lessard, 2010). Disturbances are often the result of caving, poor fragmentation (boulder arching or cohesive arching), poor design of the orepass (wall stability or piping), or ingress of water (mud rushes or piping) (Beus et al., 2001). Even though loss of orepasses is a recognized problem and often leads to a long-term reduction in operational capacity, there is still no efficient strategy that will quickly bring the orepasses back to an operational state and maintain the

Figure 1—Production area A

Figure 2—Production area B

Figure 3—Production area C

transported from the drawpoints to the orepasses. From the orepasses, the ore is collected on the main haulage level by trucks and then hauled to the crusher, where the rock is fragmented and further transported.
The effects of orepass loss on loading, hauling, and dumping operations and production rates

Operational capacity of the area. Several simulation studies that dealt with transportation systems of rock in underground mines have been performed for different mining operations (Hoare and Willis, 1992; Hunt, 1994; Turner, 1999; Salama, Greberg, and Schunnesson, 2014; Salama et al., 2016; Usmani, Szymanski, and Apel, 2014). However, the effects of orepass loss on LHD operations have not been extensively studied and published. As the rehabilitation cost is high and the time required is usually long, in some cases extending to years, a new orepass is often developed instead. If unpredicted orepass failure occurs, alternative strategies should be applied in order to compensate for the loss. One of the possible strategies is to increase the fleet size in the areas where an orepass is available or use an alternative transportation system such as trucking. However, this might not be a feasible solution due to, for example, possible traffic congestion.

Simulation

The study presented in this paper was conducted using discrete event simulation (DES). When using DES, the goal is not to model an exact representation of the real system but to answer or solve specific problems using a simplified model of the system. In other words, the goal is to capture the important features that are of interest without losing focus on quality aspects such as building of the model, analysis, and event runs (Thesen and Travis, 1998). In this study, the simulation tool AutoMod was used to model the studied production areas. AutoMod was chosen as it has the necessary capabilities and flexibility in developing various custom-made transportation systems, together with inbuilt debugging and tracing features (Greberg and Sundqvist, 2011) that make the verification and validation process easier.

Scenarios

In this study, 15 production scenarios were simulated (Table I) for each of the three production areas. In Table I ‘FL’ stands for the orepass located in the far-left side of the studied production areas, ‘CL’ stands for the orepasses located in the centre-left position, while ‘CR’ and ‘FR’ stand for centre-right and far-right, respectively. The number of operational orepasses varies between one and four while the number of LHDs ranges between one and six in each scenario. The number of vehicles was varied for each scenario in order to check whether the current infrastructure and system can accommodate additional machines in case of orepass loss. The maximum number of six LHDs was used in this study, since larger numbers of LHDs would result in excessive waiting times for the machines, and would be highly impractical. The simulated production scenarios are not meant to predict exact numbers but are constructed for the purpose of analysing and visualizing how the loss of an orepass affects the productivity of LHDs. This means that the focus was not on optimizing the number of LHDs but to study and analyse the effects of orepass loss on loading, hauling, and dumping operations and production rates for the specified scenarios. The modelled scenarios can be summarized as follows:

- In scenarios 2 to 5, three orepasses were operational and one was not operational
- In scenarios 6 to 11, two orepasses were operational and two were not operational
- In scenarios 12 to 15, one orepass was operational and three were not operational

All simulations resulted in the following outputs: time to mine out the production area, the LHD waiting times caused by an orepass loss, the production rate, and the LHD travelling distances. Production rate is defined as total tons loaded into the orepasses located in one of the production areas divided by the total number of days required to complete production. Waiting time is defined as the percentage of the time that LHD operators had to wait in the production drifts for an orepass to become available.

Simulation model verification and validation

A number of tests were performed in order to ensure the correctness of the model (verification) and to ensure that the model reflects the real system (validation). Verification and validation tests of the model are necessary to increase the level of confidence, the credibility, and the probability of the model correctness (Kleijnen, 1995; Sargent, 2011). Thus, verification and validation tests reduce the risk of making errors and generating misleading results. When verifying, the consistency of the simulation model with the conceptual model was ensured by testing the model behaviour with the specialists and experts involved in the study. Moreover, by testing the logic with the use of an interactive run controller (debugging feature), an examination of the animation and systematic scanning of the logic was performed. When validating the model, a comparison of scenarios with the real system was made with the assistance of specialists. This was ensured by comparing the production data from the operating mine with the results from the simulation. In addition, extreme condition tests, degenerate tests, traces, and internal validity were used to test the credibility of the model. An extreme condition test was performed on the LHDs by setting their bucket capacity to zero and, after the simulation was running, observing the final production results. The degenerate test was used to see how the average number of cycles,

<table>
<thead>
<tr>
<th>Scenario no.</th>
<th>No. of operational orepasses</th>
<th>Orepass operational states</th>
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<td></td>
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<td>CL</td>
</tr>
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<td>15</td>
<td>1</td>
<td>0</td>
</tr>
</tbody>
</table>
The effects of orepass loss on loading, hauling, and dumping operations and production rates

production rates, and queuing change when different input parameters are used in the simulation model. Traces were used to trace the behaviour of the specific entities during the run of the model (possible via debugging feature). Tracing of an entity was used to ascertain any abnormal symptoms that the specific entity displayed, and further to determine the exactness of, and if necessary correct, the model logic. Internal validity was tested (five additional replications were performed for each set of different scenarios in each production area) by ensuring that, after changing the stream random numbers in the model, the model results did not differ significantly from each other. In most of the cases, the results of running the model with different stream random numbers deviated from each other by no more than 5%. However, higher output variations were detected in production area C after running scenarios 12–15, with more than two LHDs in operation. This deviation is related to the way in which the vehicles claim the orepass locations. The orepass locations are claimed by vehicles after the bucket has been filled and before the vehicle has started to travel to the location. This could create a condition, especially for larger production areas, with fewer accessible orepass locations, in which the vehicle further away from the accessible orepass claims the orepass before the vehicle closer to that orepass. Since the simulated production scenarios were not meant to predict exact numbers but were constructed for the purpose of analysing and visualizing how the loss of an orepass affects the productivity of LHDs, this was considered acceptable and no further action was required.

Model settings and logic

The simplified model logic is described in the flow chart shown in Figure 4. In all cases, machines were ordered to travel to one of the production drifts and then assigned to one of the orepasses. The machine operators chose their destination by selecting the closest orepass. If the closest orepass was unavailable, the machines travelled to another orepass close by. If all orepasses were unavailable, the machine operators waited until an orepass was available. This means that the LHD cycle time is affected by the time that the LHD has to wait in the production drift for the orepass to become available. Each simulation ended when there was no longer any material left to be loaded, indicating that the production level was mined out. It was assumed that there was enough blasted material for loading at the faces at all times. The condition of the lighting in the drifts, floor, roof, and walls was not considered as an obstacle to the loading processes, hence collisions with walls or spillage of the muck would not influence the vehicles. To simplify the operation, one type of machine, a 21 t diesel LHD, was in operation for 15 hours per day per simulated production area. However, during this time the LHD encountered delays in operation whenever production disturbances or breakdown of the vehicle took place, thus reducing the operating time to almost 10.3 hours per day. Work was performed seven days a week. During the breaks, the machine was sent to the closest parking space. Whenever breakdown of a vehicle took place, the vehicle stopped at its current location.

Input data

The model input data consists of the mine layout (Figures 1–3) with the distances from the loading location to the orepasses, availability of the vehicles, availability of the production areas, and performance data of the vehicles (Table II). The theoretical tonnage available in each production area was estimated by adding 10 000 t of ore for every 3 m blast. These resulted in totals of 15.06 Mt of ore in production area A, 14.37 Mt in production area B, and 17.43 Mt in production area C. It was assumed that LHDs will be available for operation 90% of the total time and the production disturbances were estimated to be 20% of the total time. Availability of the vehicles and production disturbances are values obtained from the studied mine. The LHD performance data (Table II) was collected from the mine on several different occasions through video recordings, documentation, and time studies. Data collected from video recordings and from time studies was used to identify the probability distributions and Kolmogorov-Smirnov, Anderson Darling, and chi-square goodness-of-fit tests were performed to evaluate the chosen distribution and make sure that the formal conditions are satisfied. Kolmogorov-Smirnov and chi-square are standard goodness-of-fit tests (Banks et al., 2013). The Anderson Darling test is similar to Kolmogorov-Smirnov but is based on a more comprehensive measure of difference (Banks et al., 2013). In Table II:

- Time spent at the face refers to the time that the LHD operator spent at the face preparing the material for
The effects of orepass loss on loading, hauling, and dumping operations and production rates

<table>
<thead>
<tr>
<th>LHD type</th>
<th>Time spent at the face (s)</th>
<th>Time spent at the orepass (s)</th>
<th>Travelling speed (km/h)</th>
<th>Bucket weights (t)</th>
<th>Turning (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>21 t diesel machine</td>
<td>Log-Pearson III distr. ($\alpha=7.6013$, $\beta=0.18222$, $\mu=2.5624$)</td>
<td>Log-Pearson III distr. ($\alpha=1.8987$, $\beta=0.0456$, $\mu=2.2533$)</td>
<td>17.48</td>
<td>Normal distr. (19.32, 3.436)</td>
<td>7</td>
</tr>
</tbody>
</table>

Table II

Performance data of the LHDs

Loading and loading the bucket. The time is measured from when the vehicle enters the production area to when it leaves. Time was recorded only if the machine was leaving the face area with a loaded bucket, otherwise the time spent at the face was considered as a disturbance and assumed to be included in the production disturbances. The input data was collected from a production area located in the same orebody as production area B and with similar production area layout as production area B. Time spent at the face varied from 13 to 429 seconds, with an average of 59.8 seconds.

- Time spent at the orepass refers to the time that the LHD operator spent at the orepass location dumping the material. The time is measured from when the vehicle enters the orepass area to when it leaves. The input data was collected from a production area located in the same orebody as production area A and having a similar production area layout. Time spent at the orepass varied from 10 seconds to 12 seconds, with an average of 10.4 seconds. This saved time when dumping and thus the machine stopped at the orepass for no longer than 2 to 3 seconds.

- Travelling speed refers to the average speed used for vehicle movement in the simulation model. An average travelling speed was calculated from the wireless online loader information system (WOLIS) (Adlerborn and Selberg, 2008) of one of the mined production areas from years 2009–2014. The input data was collected from a production area located in the same orebody as production area B and having a similar production area layout. The average speed was based on the distances and travelling cycle times. In the real system, the travelling speed of the LHD could be different depending on whether the bucket is empty or loaded, the skills of the operators, infrastructure, or machine reliability. However, with regard to the purpose of the simulation, there is a need for some degree of simplification to avoid unnecessary complexity of the model. Therefore, the simulation model travelling speed was calculated based on average cycle times. This simplification does not affect the study and the simulation since the aim is not to simulate the exact behaviour of the LHD movement but rather to ascertain the effects of orepass loss on loading, hauling, and dumping operations.

- Bucket weights represent the tons drawn from the production face. The bucket weights are based on the WOLIS data gathered from the operating mine. Bucket weight data shown in Table II is from one year of production (Gustafson et al., 2013). A normal distribution was selected to represent this data as it is usually used to model the events with limited variability (Banks, 2004).

Turning represents the time that the LHD operator spent turning the vehicle before and after entering the orepass location. The time is measured from when the LHD operator starts to make a turn to when the vehicle starts to make its way to or away from the orepass location. The data was collected from a production area located in the same orebody as production area A and having a similar layout. Time spent turning the vehicle varied from 6 seconds to 8 seconds, with an average of 7 seconds.

Results and discussion

The simulation was conducted for 15 scenarios for each of the three production areas in three different orebodies: A, B, and C respectively. Each scenario consists of six runs where the number of LHDs ranged from one to six, and the number of operational orepasses ranged from one to four. Having many vehicles in only one production area creates a higher risk of production disturbances, especially when an orepass fails. Furthermore, the ventilation of the drifts to remove the exhaust gases would have to increase, increasing both the cost for ventilation and the traffic congestion. Therefore having more than six LHDs in operation would not be a feasible solution and thus was not simulated in this study. The time to mine out the production area varied depending on the number of LHDs and orepasses in operation (Figure 5–7).

Time to mine out the production area

In Figures 5–7, the horizontal axis shows the scenario number clustered with the maximum number of operational orepasses for a given range of scenarios. The vertical axis shows the time that the LHDs required to finish mining the whole production area. The time to finish mining the production area in orebodies A, B, and C (Figure 5–7) ranges from 733 days to 5,902 days, 628 days to 4,098 days, and 856 days to 7,644 days respectively, depending on the scenario and number of LHDs used. The highest variation in the time to finish a mining production area is observed for orebody C, whereas the lowest variation is observed in orebody B.

Variation in number of operational orepasses with the same number of LHDs in operation

When there are four or three operational orepasses (Figure 5–
The effects of orepass loss on loading, hauling, and dumping operations and production rates

7), the time to mine out the production area is similar, with a difference of less than 653 days. When three or two orepasses are operational, the time to finish a mining production area varies little compared to when all four orepasses are operational. This suggests that in the case of scheduled maintenance or operational or structural failure of an orepass, there are enough orepasses available to take up the extra capacity. Hence, there is a possibility of increasing the number of LHDs operating, as long as the orepasses are not over-utilized and as long as the LHD waiting time for the orepass to become available is not excessive. When only two orepasses are operational, the differences become larger, and vary up to 131 days. If only one orepass is operational, the production rate drops considerably, extending the time to mine out the production area by up to 3,804 days.

Variation in number of LHDs in operation

In all scenarios, the addition of one LHD is sufficient to maintain or even improve production. However, the waiting time for the orepass to become available slowly starts to take effect, reducing the vehicle working time, since when the number of LHDs increases from one to four, the difference in the time to finish mining the production area is greater than when increasing the number from four to five or to six. In scenarios 12–15, with more than two LHDs in operation and only one operational orepass, the difference in time to finish mining the production area is much lower. As a consequence, this leads to longer waiting times for the LHDs since an extra LHD increases the waiting time for the vehicles operating in that area.

Effect of an orepass loss on LHD waiting times

The LHD waiting times are related to waiting for an orepass to become available for dumping the ore, and are presented in Figures 8–10. Waiting time can comprise up to 70% of the total time that the vehicle was scheduled to work. The total sum of waiting times for each scenario confirms that the lower the number of operational orepasses, the higher the total waiting time, ranging between 0% and 70% of the total time. The locations of the orepasses in the production area are not as critical as the orepass loss in each set of scenarios (2–5, 6–11, 12–15). For example, in orebody A, when six LHDs are used, the differences in waiting times in scenarios 6–11 are not higher than 3.3%, whereas the difference in waiting time between scenarios 5 and 6 is 17.4%.

Production rates

Figure 11 and Figure 12 show comparisons of production rates between the production areas. Depending on the scenario number and production area, the production rates vary, but tend to decrease with decreasing number of available orepasses, especially when six LHDs are in operation. Only the results from scenarios with three and six

![Figure 5—Time to finish mining production area A](image)

![Figure 6—Time to finish mining production area B](image)

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LHDs are shown since the results from scenarios with one, two, four, and five LHDs follow a similar pattern. In production area B the production rate is higher than in production areas A and C as there are shorter distances to cover. Differences between the production areas depend mainly on the average distance that the LHD needs to travel to the orepasses. The longest average distance that the LHD must travel to the orepasses is in production area C. The highest variation in the production rate, when using six LHDs (Figure 12) was 14 235 t/d in orebody A, whereas the highest variation when using three LHDs (Figure 11) was 6 920 t/d in orebody C. Thus, the higher the number of the LHDs operating in the production area affected by an orepass loss, the higher the production rate variation.

Figure 13 shows the relationship between the production rate and distance travelled in production area C. The results are based on the use of one LHD. A similar pattern followed in production areas A and B. From Figure 13 it can be seen that the lower the number of operational orepasses, the higher the average LHDs travelling distance and the lower the production rate. In scenarios number 12 and 15 the machines had to travel further than in scenarios number 13 and 14, resulting in lower production rates. This is because the location of the operational orepass in scenario 12 and scenario 15 is to the far left and far right side of the studied production area. The most severe scenarios are those in which the orepasses closer to the centre of the studied production areas are lost. Similarly, in scenarios number 6, 8, and 9, the LHDs had to travel further than in scenarios number 7, 10, and 11.

Likewise, in Figure 14, the relationship between the production rate and total distance travelled in production area C is shown. These results are based on having six LHDs in operation. In scenarios with more operational orepasses (scenarios number 6, 8, 9, 10, and 11) the travelling distances were further than in the scenarios with one operational orepass (scenarios number 13 and 14). This suggests that the location of the orepasses and sequencing of the vehicles affects the LHD distance travelled, but not necessarily the production rate.

Additionally, the different locations of the orepasses affect the production. This can be shown when three LHDs are working on the right-hand side of the production area. The distance to the far right orepass is much shorter than to the other orepasses, but only one LHD can be accommodated at a time in the FR orepass, meaning that the other two LHDs would have to travel to another orepass or wait for the first LHD to finish. Therefore, an appropriate strategy for fleet management is necessary in order to maintain the production rate. One possible solution is to decrease time losses by
The effects of orepass loss on loading, hauling, and dumping operations and production rates

Figure 9—LHD waiting times for the orepass to become available (production area B)

Figure 10—LHD waiting times for the orepass to become available (production area C)

Figure 11—Scenario vs. production rate for different areas based on the use of three LHDs
managing the movement of the LHDs. For example, whenever one LHD is dumping, the second is loading, and the third is travelling to or from the orepass. Another option is to not allow the use of more than one LHD in each production area, by redirecting part of the fleet into another production area.

Conclusions
Several scenarios were studied with the purpose of analysing the effects of the loss of one or more orepasses on the LHD operations and the production rate. The main objectives were to study how the current LHD operations are affected when one or several of the orepasses cannot be used, and how many of the orepasses can be lost while still maintaining production rates. The following conclusions can be drawn.

- If one or two of the four orepasses are lost, production can be maintained, or even improved, by the addition of one LHD.
- In the scenario when only one orepass remains in operation, the production rate of the vehicles drops due to increased waiting times, especially when extra LHDs are being used.
- Operating with a high number LHDs in the production area affected by an orepass loss results in greater variations in production rate than when operating with a low number of LHDs. Thus, instead of using more LHDs in the area where an orepass failed, transferring the LHDs to another production area (if possible) to achieve higher production rates could be a short-term option.
- When two adjacent drifts are used for loading, the LHDs may end up having to travel to the orepass located further away from the production drift as the closer one may be occupied by another LHD. Consequently, this would lead to a decrease in production rate and should be avoided.
- When two orepasses are operational, increasing the number of LHDs from five to six would result in a higher production rate, whereas if only one orepass remains in operation, using six LHDs would result in almost the same production rate as using five LHDs. The remaining orepass in operation will be at maximum utilization, with the LHDs continuously claiming that orepass. This restricts the increased number of LHDs from dumping more material into the

Figure 12—Scenario vs. production rate for different areas based on the use of six LHDs

Figure 13—Relationship between LHDs production rate and travelled distances in production area C based on the use of one LHD
orepass and highlights the need for analysis of the system in order to avoid increasing waiting times and traffic problems by using too many machines in the area.

- In the studied production areas, the most severe scenarios involve losing the orepasses that are located closer to the centre of the production area.

- Mine management should have a strategy for orepass loss situations in order to be able to mitigate the possible production disturbances and try to avoid high variations in the production rates. Otherwise, loss of an orepass would likely result in additional fleet requirements or in production disturbances while the orepass is being restored.

- In order to avoid increased waiting times and traffic congestion due to using too many machines in the working areas, an evaluation of whether the system can accommodate an extra machine in the production area could be conducted. Alternatively, the machine should be redirected into another production area, or additional orepass inlets could be constructed to allow entry for the incoming fleet.

The simulation created a baseline for further studies of possible alternatives and improvements that can be made for future underground transportation systems. As most underground mines operate with small economic margins, strategies should be developed to mitigate the production disturbances related to the loss of one or more orepasses. The results of this investigation show clearly the effect of the number of additional LHDs in operation on production rate, stressing the importance of a well-developed strategy.

**References**


Simulating the effect of LHD operations on production rates and ventilation costs in a sublevel cave underground mine

Simulating the effect of LHD operations on production rates and ventilation costs in a sublevel cave underground mine

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ABSTRACT

Load-haul-dump machines (LHDs) are typically used in underground metal mining operations. Delays or inappropriate use of LHDs can result in production loss. Optimized LHD use is especially crucial in larger mines because longer travel distances increase heat, dust, and gas emissions, which in turn increase ventilation costs. This study, conducted at an existing Swedish sublevel cave underground mine, used discrete event simulation and the AutoMod™ DES tool to determine the ventilation costs related to using too many diesel LHDs in a production area with reduced ore pass availability. Ventilation costs related to operating LHDs in the production area were found to increase by as much as 200, 224, and 306% for one, three, and six LHDs in operation, respectively, when fewer ore passes are available.

KEYWORDS

Discrete event simulation (DES), Loading operations, Production rate, Ventilation costs

INTRODUCTION

Transporting material from the mine face to the dumping location requires flexible loading and hauling equipment that is able to cope with sharp curves and a complex mining environment. Load-haul-dump machines (LHDs) are designed for such an environment. Models are available to suit specific features of the working environment such as small, large, or narrow open stopes (Atlas Copco Rock Drills AB, 2007). When carrying a full load, haulage speed is higher for LHDs than for front-end loaders with the same engine size. Also, payloads are approximately 50% higher for LHDs than for front-end loaders (Atlas Copco Rock Drills AB, 2007). LHDs are capable of climbing steep grades: effectively at up to 20% and ideally at up to 14% (Bloss, Harvey, Grant, & Routley, 2011). The haulage distance between stope and ore pass is usually between 50 m and 400 m. LHDs are efficient for tramming distances up to 200 m (Bloss et al., 2011), whereas when loading trucks work efficiently when the distance is usually less than 100 m (Atlas Copco Rock Drills AB, 2007). The largest diesel-powered LHDs, with drive motors of up to 350kW, can handle a payload of 21 t (Wennmohs, 2014). Most mining equipment runs on diesel fuel, which provides flexibility, high durability, and high efficiency (Pronk, Coble, & Stewart, 2009); however,
other alternatives such as electric, hybrid diesel/electric, or hydrogen fuel cell machines are increasingly being developed and used. This change is due to diesel-powered machines accounting for a significant portion of the ventilation costs in the mine (Chadwick, 2008; Paraszczzak, Laflamme, & Fytas, 2013). Currently, as part of the European sustainable intelligent mining systems (SIMS) project, Epiroc AB is planning to demonstrate technology for a diesel-free underground mine at Agnico Eagle’s Kittlää mine in Finland using battery-powered mobile equipment, with the goal of achieving the same production as could be realized using a fleet of similar diesel-powered equipment in the same environmental conditions (SIMS, 2018).

The purpose of this study, based on operations at an existing Swedish mine, is to determine the changes in production rate and ventilation costs related to the use of additional diesel LHDs when an ore pass is not available. The AutoMod™ discrete event simulation (DES) tool is used to simulate the underground loading operation at one of the production levels. The ventilation costs related to operating the LHDs in the studied production area are calculated based on the technical and operating parameters of a 21 t LHD (motor power, operating times, production rates), the ventilation system of the studied production area (flow rate capacity, ventilation requirements), and electricity costs.

Ventilation in underground mines

Ventilation accounts for a large part of the power costs for a mine. According to de la Vergne (2003), ventilation is responsible for one-third of the power costs for a mine; Paraszczzak et al. (2013) report that the costs are higher (up to 40%). The ventilation requirements (diesel exhaust and noxious gases) in Sweden include regulations for nitrogen dioxide (NO₂) and carbon monoxide (CO) content (Workplace exposure to dusts and aerosols – diesel exhaust, 2017). In this study, the ventilation requirements and costs related to LHDs are analyzed. When many diesel machines are in operation, high amounts of diesel exhaust are generated (Prönk et al., 2009). If the working areas are not ventilated properly, workers will be at risk of exposure to noxious gases and carcinogenic agents (Prönk et al., 2009). There is a need to provide oxygen to personnel and for combustion, but also to provide the necessary volume of air to reduce fumes from diesel engines, remove diesel particle matter (DPM) and blasting fumes from the air, regulate heat, and improve visibility (Tuck, 2011). One solution is to replace older mining equipment with newer equipment that emits less noxious gases (thus improving health and safety for the personnel) and produces less DPM, dust, and heat, while at the same time reducing energy consumption. An alternative is to use electric vehicles in the system or creating an entry zone where personnel access is restricted to limit human exposure to poor quality air.

Discrete event simulation (DES)

DES is usually applied to dynamic, complex systems, and many recent studies of mining operations have used this technique. Yuriy, Runciman and Vayenas (2013) presented and discussed the Vale Canada experience using the WITNESS, Quest, SimMine®, ProModel®, and AutoMod simulation tools in terms of functionality, capability, and user-friendliness when applied to underground mining operations. Bailey, Olsson, and Glassock (2012) used Arena® software to model the face handling system at Olympic Dam, Australia. Greberg and Sundqvist (2011) simulated the mine planning process at the Newcrest Cadia East underground operation in Australia to optimize the development of panels. Usmani, Szymanski, and Apel (2014) used a simulation to optimize the extraction level of a block cave mine. Greberg, Salama, Gustafson, and Skawina (2016) used the AutoMod simulation model to analyze the costs of using haul trucks to transport ore and waste up the ramp to the existing crusher.

A detailed, accurate simulation model for large, complex systems requires a large dataset, statistical distributions of the data, and a careful choice of simulation software. In this study, the simulation tool AutoMod (Applied Materials, 2019) was used to model the studied production area. The AutoMod tool is based on DES and was used because it has the necessary capabilities and flexibility to model various custom-made transportation systems and has built-in debugging and tracing features that make the
verification and validation process easier. A verified and validated simulation model provides results similar to those seen in an actual operating system.

CASE STUDY

This study is based on data and information obtained from an underground mine located in Sweden. The mine uses sublevel caving methods to extract iron ore. The ore reserves are distributed over 20 large and small orebodies, with 14 orebodies currently in production. Depending on the type of orebody, the interval between the sublevels varies from 20 to 30 m, with drift spacing of 22.5 m. The orebodies are divided into blocks and then further divided into production areas spread over a 5 km long and 2.5 km wide area, each with its own group of ore passes. The ore passes are approximately 300 m long, have a diameter of approximately 3 m and dip approximately 60°. The drifts are approximately 5 m high and 6.5 m wide. The sequencing of drifts to be extracted is based on rock stress–related factors, the ancillary activities of the production drifts, distances to the ore passes (depending on the production rate requirement), and the amount of ore left in each of the production drifts. The number of ore passes in the production areas varies between one and four depending on the size of the production area. Figure 1 presents the orebody outlines, ore passes, and production drifts used for hauling the ore from the face to one of the ore pass locations.

![Figure 1. Studied production area at an underground mine in Sweden (plan view)](image)

The current loading practice is based on the use of manually operated diesel LHDs with a 21 t bucket capacity. In the smaller production areas, there is usually one LHD in operation. In larger production areas, two LHDs can be used simultaneously. The ore is transported from the draw points to the ore passes. From the ore passes, the ore is collected on the main haulage level by trucks and then hauled to the crusher, where the rock is fragmented and further transported via conveyor to the hoisting system and to the surface.

The ventilation system in the mine is automated and consists of 4 inflow and 12 outflow ventilation shafts. The studied production area is located in the eastern part of the mine at level 996 (Figure 2). The inflow ventilation shafts cover different regions and the evacuation lifts are located behind these shafts. Outflow shafts cover the old mine workings and some of them have been sealed with concrete and rock to prevent airflow leakage. In the winter, the air is heated with an oil heater to a temperature of approximately +1°C. The first set of fans pushes the air through the ventilation shaft to level 800. From level 800, a second set of fans push the air down to level 1,250. In the central region on level 800, two 2 MW fans operating at a pressure of 1,500 Pa, distribute the air to other parts of the mine.
Simulation settings for the predefined production area

The production area shown in Figure 1 was modelled using AutoMod simulation software. In the simulation, the LHDs were ordered to travel to one of the production drifts and then assigned to the closest available ore pass. If all ore passes were unavailable, the LHD operators waited in the production drift until an ore pass was available, which affected the LHD cycle time. Each simulation ended when there was no longer any material left to be loaded, that is, when the production level was mined out. It was assumed that there was enough blasted material at the face to load the LHDs at all times. The LHDs were scheduled to operate in the simulated production area for 15 h/day. During operations, the LHDs encountered delays, thus operating time was reduced, whenever production disturbances or vehicle breakdowns took place. Whenever a vehicle broke down, it stopped at the location of the breakdown. Work was performed seven days a week and during breaks, the LHDs were sent to the closest parking space.

Input data

For the model, the total theoretical volume of ore available in the studied production area was 17.43 Mt. The LHD performance data were collected from the mine on several occasions via video recordings, documentation, and time studies. These data consisted of time spent by the LHD at the face (13–429 s), time spent by the LHD at the ore pass (10–12 s), travelling speed (17.4 km/h), bucket weights (19.32 ± 3.4 t), and turning times (6–8 s). The data variations were fitted to probability distributions using the statistical tool EasyFit (MathWave, 2019) to characterize the uncertainty and randomness of the operation. It was assumed that LHDs would be available to operate 90% of the total time and production disturbances were estimated to account for 20% of the total time. The 21 t diesel LHD used in the study has a drive power of 350 kW and a vehicle weight of 56.8 t. Based on the information obtained from the mine, the ventilation in the studied production area has flow-rate capacity of 250 m$^3$/s, consuming 2,661 kW power at a typical resistance of 0.1 Ns$^2$/m$^2$. The ventilation requirements of the studied...
production area are set at 0.05 m³/s/kW. The assumed electricity cost was set to US$0.12/kWh (United States Energy Information Administration, 2017).

**Scenarios**

Fifteen combinations of ore pass availability and location were analyzed and are termed scenarios in this study (Table 1). For each scenario, the number of operational ore passes varied between one and four. In Table 1, FL stands for the ore pass located in the far-left side of the studied production area shown in Figure 1, CL stands for the ore passes located in the centre-left position, and CR and FR stand for centre-right and far-right, respectively. Each scenario was tested with one, three, and six operating LHDs with the aim of determining the incurred ventilation costs due to longer haul times resulting from ore pass unavailability when additional LHDs are used. Varying the number of LHDs was done to observe the variations in energy and ventilation costs in the current infrastructure in case of ore pass unavailability. A maximum of six LHDs was modelled because a larger number of machines would result in extensive waiting times and would be highly unpractical. For all scenarios, LHD production rates and operating times were obtained from simulation analysis. The typical resistance was calculated based on the ventilation capacity of the studied production area. The ventilation costs related to operating the LHDs in the studied production area were obtained by calculating the airflow rate for loaders, then converting the flow rate into an air power and further multiplying the air power by operating times and electricity costs.

**Table 1. Number and location of operational ore passes for 15 simulated operation scenarios in the studied production area; FL: far left ore pass, CL: centre left ore pass, CR: centre right ore pass, FR: far right ore pass**

<table>
<thead>
<tr>
<th>Scenario</th>
<th>No. of operational ore passes</th>
<th>Ore pass operational state (1: operating, 0: not operating)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>4</td>
<td>1 1 1 1</td>
</tr>
<tr>
<td>2</td>
<td>3</td>
<td>1 1 1 0</td>
</tr>
<tr>
<td>3</td>
<td>3</td>
<td>1 1 0 1</td>
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<td>3</td>
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<td>3</td>
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<tr>
<td>6</td>
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<td>7</td>
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<td>8</td>
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</tr>
<tr>
<td>14</td>
<td>1</td>
<td>0 0 1 0</td>
</tr>
<tr>
<td>15</td>
<td>1</td>
<td>0 0 0 1</td>
</tr>
</tbody>
</table>

**Model verification and validation**

Verification and validation tests of the model are necessary to increase the level of confidence, the credibility, and the probability of model correctness (Sargent, 2011). Verification tests ensure that a conceptual model is accurately translated into a simulation model, whereas the validation tests ensure that the model reflects the real system to a given/specifed level of detail (Sargent, 2011).
The results of the simulations run in this study were tested for correctness and credibility. Simulation model consistency with the conceptual model (verification) was ensured by having the specialists and experts involved in the study test the model behaviour and use a debugging feature called an internal run controller. Specialists also assisted with the comparison of the scenarios to the real system (validation). Validation was ensured by comparing the production data from the operating mine with the results obtained from the simulation. In addition to validation, extreme condition tests, degenerate tests, traces, and internal validity were used to test the credibility of the model. Internal validity was tested by running an additional five replications for each set of scenarios in each production area to ensure that the model’s results did not significantly differ.

RESULTS AND DISCUSSION

The simulation was conducted for 15 scenarios, each with a different number (1–4) and arrangement of operational ore passes. For each scenario, runs were completed with one, three, and six LHDs in operation (one run for each number of LHDs for a total of three runs per scenario). The production rate and ventilation costs for each run are presented in Figure 3. The ventilation costs calculated in this study are intended for comparative use only and do not represent realistic cost values. The production rate is defined as total tonnes loaded divided by the total number of days required to finish production.

![Figure 3. Production rate (PR) and ventilation costs (VC) for the studied production area during scenarios with varying ore pass availability and location and one, three, or six operating load-haul-dump (LHD) machines](image)

The production rate varied depending on the scenario, with a general trend towards a lower rate as the number of available ore passes decreased. For each scenario, an increase in the number of LHDs in operation resulted in an increase in the production rate. The highest production rate, which occurred when six LHDs were in operation and all ore passes were available (scenario 1), was 20,350 t/day, whereas the lowest production rate, 2,280 t/day occurred when only one LHD was in operation (scenario 15). In areas affected by an ore pass availability issue, the production rate decreased by as much as 2,259, 6,920, and 13,642 t/day for one, three, and six LHDs in operation, respectively. The ventilation cost increased as the
number of LHDs in operation increased. For one and three LHDs in operation, the ventilation cost averaged US$0.12/kt/LHD and US$0.43/kt/LHD, respectively. The difference between the highest and lowest ventilation costs across all 15 scenarios was US$0.09/kt/LHD with one LHD operating and US$0.36/kt/LHD with three LHDs operating. In cases where six LHDs were used, the ventilation cost increased to an average US$1.21/kt/LHD, and the range across scenarios was US$1.47/kt/LHD. Overall, when less ore passes are operational and more LHDs are in use, ventilation costs rise by an average of US$1.09/kt/LHD.

In Table 2, the variation in the production rate and ventilation costs are shown as percentages, where a production rate of 100% and a ventilation cost of 100% are defined as the values recorded for scenario 1 (four ore passes in operation). All other percentages are calculated as (scenario X / scenario 1) × 100, where scenario X is scenario 2–15.

<table>
<thead>
<tr>
<th>Scenario</th>
<th>1 LHD</th>
<th>3 LHDs</th>
<th>6 LHDs</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Production rate (%)</td>
<td>Ventilation cost (%)</td>
<td>Production rate (%)</td>
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<td>Four ore passes in operation</td>
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<td>Two ore passes in operation</td>
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<td>15</td>
<td>50</td>
<td>200</td>
<td>45</td>
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<tr>
<td>One ore pass in operation</td>
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Based on the results in Table 2, the production rate decreases by as much as 50, 45, and 33% for one, three, and six LHDs in operation, respectively, when ore pass availability is reduced. The results also show that the ventilation cost increases by up to 200, 224, and 306% for one, three, and six LHDs in operation, respectively, when fewer ore passes are available.

In the event that an ore pass is not available, buying new LHDs is not a likely scenario, and often LHDs from other production areas are relocated to the affected production area to compensate for the longer travelling distances and maintain production targets. Increasing the number of LHDs in a production area with reduced ore pass availability decreases the production rate and increases ventilation costs. This can be illustrated using the production rates for one, three, and six LHDs in scenario 1. The production rates for the separate production areas are based on the production rates achieved in scenario 1 and multiplied by the number of production areas used. Comparing three LHDs operating in three separate production areas (13,614 t/day) with the same three LHDs operating in a single production area (12,499 t/day) results in a difference of 1,115 t/day. Similarly, six LHDs operating in six separate production areas yields 27,228 t/day, whereas the same six LHDs operating in a single production area...
yields 20,349 t/day, a difference of 6,879 t/day. If the same comparison is made using the results of scenario 15, in which only one ore pass is operational, the differences are even higher: 8,035 t/day for three LHDs and 20,521 t/day for six LHDs. Similarly, the ventilation cost for three LHDs working in three separate production areas with similar conditions to scenario 1 is US$0.29/kt, whereas the ventilation cost for three LHDs working in a single production area is US$0.54/kt. The ventilation cost for six LHDs working in six separate production areas with similar conditions to scenario 1 is US$0.54/kt, whereas if six LHDs are in operation in a single area, the cost is US$0.71/kt. If a similar comparison is made using scenario 15, in which only one ore pass is operational, the difference in cost is even higher, US$0.39/kt for three LHDs in operation and US$1.64/kt for six LHDs in operation.

CONCLUSIONS

In this study, scenarios representing variations in ore pass availability with different numbers of operating LHDs were analyzed to determine the additional ventilation costs related to LHD operation when one or more ore passes are not available. The results indicated the following:

- Increasing the number of LHDs in the areas affected by disturbance (e.g., ore pass availability) causes LHD production rates to decrease and ventilation costs to increase.
- The production rate decreased by as much as 50% (2,259 t/day), 45% (6,920 t/day), and 33% (13,642 t/day) for one, three, and six operating LHDs, respectively, when the number of ore passes was reduced from four to one.
- The ventilation cost increased by as much as 200% (US$0.09/kt/LHD), 224% (US$0.36/kt/LHD), and 306% (US$1.47/kt/LHD) for one, three, and six operating LHDs, respectively, when ore pass availability was reduced from four to one.
- Increasing the number of LHDs operating in an area affected by a disturbance (e.g., ore pass availability) generated higher ventilation costs and decreased the potential production rate. One, three, and six LHDs operating in separate production areas produced an additional 1,115, 8,035, and 20,521 tonnes/day, respectively, than did the same number of LHDs operating within a single production area.

In the event that an ore pass is not available, buying new LHDs is not a likely scenario, and often LHDs from other production areas are relocated to the affected production area to compensate for the longer travelling distances and maintain production targets. This study shows that increasing the number of LHDs in a production area with reduced ore pass availability decreases the production rate and increases ventilation costs.

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REFERENCES


Automatic closed-loop scheduling in underground mining using DES

Automatic closed-loop scheduling in underground mining using
discrete event simulation

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Synopsis: Today’s mining operations require fast reporting and rapid rescheduling based on
updated information. An automatic mine scheduling system could not only quickly reschedule but
also propose alternative solutions. In reality, the scheduling system is connected to a fleet
manager system in the mine. To avoid the financial and physical risks associated with testing
such a system in real time, it could be first evaluated via discrete event simulation models. This
would produce a safe environment for the system before final implementation of the algorithms
and tools. In this study, this is achieved by integrating an automatic ABB scheduling system and a
SimMine simulation model.

Keywords: discrete event simulation; production planning; scheduling system; underground
mining.

Introduction

The unexpected events can cause disturbances in operations, creating problems that production
planner must solve on a daily basis. In the underground mining operations, the manual
rescheduling is time consuming and production planners often find themselves in a situation where
new or readjusted schedules are required in a very short period of time. This often leads to
randomly manned machines and seldom represents an optimal solution. It is also difficult to
maintain a high quality of reporting and may even lead to a loss of important information. An
automated decision support system could help planners initiate the necessary changes within a
short period of time when unexpected events occur or when task execution can be improved. To
avoid the financial and/or physical risks of testing the system in a real environment, automatic closed-loop scheduling can be connected to a simulation system. Tests inside the real mine will still be required, but with simulation, many early issues may be resolved before changes are implemented in the mine operations. The challenge is threefold: to evaluate and accommodate complex and changing restrictions, to generate optimal production plans that meet production targets, and to promptly activate new production plans when unexpected errors occur. Introducing such an approach will help to optimize short-term production scheduling, reduce overall production costs and time, increase equipment efficiency, solve complex scheduling problems, and improve decision making by increasing the efficiency and agility of planning tasks that are subject to change. This, in turn, will reduce operator stress, provide instant task progress reports, reduce the administrative workload, coordinate mine systems with fewer human errors, and unify scheduling.

This paper discusses the integration of the ABB scheduler and the SimMine simulation model to create a possible digital twin that could be used in the future to test various mine scenarios. This study was part of the Optimized Medium Range Mine Scheduling (OMMS) project and was carried out by Luleå University of Technology (LTU), ABB, Boliden, and SimMine, coordinated by RockTech Centre, and founded by the partners and SIP STRIM. Further studies are continuing as part of a Sustainable Intelligent Mining Systems (SIMS) project, founded by the partners and the H2020 EU Research and Innovation Programme.

Production scheduling in mining

Traditionally, production planning in mining operations has been divided into four levels of planning (Figure 1). In long-term planning, the focus is on formulating rough plans for mine production over a long period. These plans change over time to reflect the effects of a changing economy and market demand for minerals (Hartmann and Mutmansky 2002). Long-term plans can be used to establish financial forecasts relevant for the purchase of new equipment, changes in the workforce, and variations in operating costs (Hartmann and Mutmansky 2002). Long-term planning allows mine planners to first set mining targets and then test scenarios to find the best way to achieve the targets over multiple working periods. Medium-term and short-term plans have a level of detail which reflects the immediate needs of the mine. These plans provide the information necessary for
future production forecasts and predictions of market demands; they usually include daily production rates, material handling plans, equipment, and resource allocation. Production scheduling, creates detailed schedules for a short period to ensure control of the ore grade, consistent daily operations (following a monthly plan), effective use of available resources, and maximum utilization of resources within the constraints of parameters (Hartmann and Mutmansky 2002).

Scheduling is defined by Pinedo (2002), Brucker (2004) and Dugardin et al. (2007) as a process of allocating the resources (people, equipment, or production lines) to tasks/jobs over a period of time. The goal is to optimize (maximize or minimize) certain objectives, such as processing times or quantities, by making appropriate decisions (Dugardin et al. 2007; Fowler et al. 2006). When assigning the resources, decisions must be based on the production targets and other factors, such as priorities, sequencing, beginning and end of the job/task, but also including unexpected stops and rescheduling (Herrmann 2006). With the increasing complexity of systems and changing restrictions, scheduling should undergo a continuous cycle of improvement. However, the manual process of generating schedules consumes a lot of time, is prone to errors, and makes it difficult to incorporate real-time information. Thus, while production planners currently
use their own experience and personal judgment (Song et al. 2015) in scheduling, they could be assisted by production scheduling systems.

**Discrete event simulation**

Discrete event simulation (DES) a widely used and suitable method for analysing dynamic systems in mining. In DES, variables change at discrete points when events occur (Banks 1998). An event can be an arrival or departure of a machine, for example. Simulation models are typically used for tactical and strategic decision support to solve problems that cannot be tackled by analytical techniques (Fowler and Rose 2004). They offer a solution to a problem in a controlled environment. In fact, one of the key advantages of simulation is that it does not disturb the existing system (Dugardin et al. 2007). This study considers the use of discrete event simulation (DES) to emulate and evaluate the scheduler, with a focus on system interaction, the scale of operation, and interaction between the SimMine underground mine model and the ABB automatic production scheduling system. The simulator and scheduler act as a test bed and decision support system. This enables the user to evaluate the scheduling platform and tune configurations in the model and the scheduling algorithm in the scheduling platform to match the mine conditions without interacting with the existing system.

**Scheduling and simulation**

There are various options for simulation and scheduling. The first option involves using the simulation as a base schedule for generating the initial schedules, improving the existing schedules, or optimizing the existing schedules from within the simulation (Fowler et al. 2006). This means the simulation model is basically a production control system used to derive production control instructions (Fowler et al. 2006). In an example given by Potradi et al. (2002), a DES model was used to generate the schedule for machines and to plan the start of operations. A second option for simulation is to set parameters or test the instance generation required for scheduling. In this case, simulation is used to evaluate certain parameter setting strategies for a given production control scheme (Fowler et al. 2006). A third option is to use simulation to emulate and evaluate scheduling approaches. This allows the behaviour of a system to be evaluated. However, the
simulation software has to have the functionality to model specific characteristics of the system (Mönch et al. 2003). Horiguchi et al. (2001) and Pritsker et al. (1997) examined the performance of the algorithm by developing a simulation model in the AweSim DES language, while Mönch et al. (2003) evaluated scheduling approaches in a simulation-based environment for manufacturing systems. In this study, we establish a connection between the ABB scheduler and SimMine to create a possible digital twin of the mine that could be used in the future to test various scenarios.

Integration of ABB scheduling system and SimMine simulation model

The following section describes the integration of the ABB Ability Operations Management System (OMS) and the SimMine simulator. The focus is on closing the loop in short-term mine scheduling by using simulation as a test bed and decision support system. ABB Ability OMS is a system for scheduling processes that creates detailed production schedules based on information collected from various sources. The system supports manual scheduling and intervention to enable such techniques as short interval control. Additional ability of ABB Ability OMS is the ability to create automatic short-term schedules. Scheduling is typically considered a Manufacturing Execution System (MES), the intermediate layer between Enterprise Resource Planning (ERP systems) and production control. This means any industrial application of a MES must be tightly integrated with the (possibly complex) information system on site. With this in mind, ABB OMS was developed to support processes that can be modelled using the ISA-95 standard (ANSI/ISA-95.00.03 2005). The adherence to ISA-95 makes the integration with various information sources easier, as a common data model is used. Further, in the mining industry, the standard is recognized by many companies as an enabler for future operation optimization (Roberts 2016). Lastly, the automatic scheduling feature in ABB Ability OMS enables closed-loop scheduling by integrating task progress from either mobile clients in the production fleet or the shift foreman. Automatic scheduling is thus used to continuously reschedule based on the latest information available. Information is presented in form of Gantt charts (see Figure 2) in which information is continuously updated and presented fromA Gantt chart is a graphical representation of the planned and actual performance (Cox et al. 1992). In this study, machine location, machine activity, machine alarms, and delays are shown.
Users can interact (in the Gantt chart) with the current schedule by reassigning orders to different equipment or editing the properties of the resources.

![Gantt chart](image)

**Figure 2** - Views of the scheduling system. The top presents a machine-centric Gantt chart; the bottom presents the same information from a location-centric view.

SimMine is a mining-oriented, discrete event simulation tool built on a full graphical interface. Since SimMine was developed specifically to simulate mining operations, it is made of interfaces where the user can set and specify underground mine-related parameters, such as fleet type and properties, rock properties, location sequences, shifts, and processing times. SimMine software accommodates various mine layouts and produces comprehensive output statistics. In this work, we develop a number of new modules to represent the surrounding information system, including extraction plans and maintenance systems. Based on the information obtained from the studied mine, SimMine can not only simulate the process but also act as the mine operation centre (MOC) to coordinate activities. The SimMine model of the simulated mine is shown in Figure 3. The figure includes the main interface and the graphical user interface (GUI). The main interface is split into main menu toolbars and tabs. Each tab consists of different input/output data for the current simulation. Input data include fleet data (shift properties, activity cycles, fleet properties, operator costs, and materials), and layout data (face profiles, location type, dependencies, priorities, rock...
properties and capacities). Output data include the statistics related to sequence, development, production, location, fleet performance measures, activity and loading, and transport. The GUI represents the layout of the 3D modelled scenario; it includes the tools and objects used for selecting, creating, or modifying the layout and view two perspectives: machine-centric and location-centric.

![Simulation model](image)

**Figure 3 – Simulation model**

*Platform integration*

The automatic ABB Ability OMS scheduling system is responsible for generating schedules and assigning resources (or equipment) to tasks. The SimMine simulation is responsible for simulating the real mining system and sending events to the scheduler (Figure 4). The layout of the integration of the SimMine simulator and the ABB OMS scheduling system includes information related to the production/development plan, maintenance plan, system settings, localization, and additional mine services that can affect the scheduling in the mine.
Production control begins at the start of the simulation. More specifically, at the start of the simulation, the simulator sends the location dependencies (available faces), order list (blast orders), and tasks (full set of Drill and Blast activities that should be performed in a particular setup) to the scheduler (Figure 5).

During the simulation, vacant vehicles request work from the scheduler. Work comes in the form of work orders. When machines perform work, their work status is transferred to the scheduler (start/stop of activities and expected end time). To communicate and exchange information, the ABB scheduler and SimMine tool use web services. The information is presented to users in form of a Gantt chart (Figure 2). In the chart, activities such as the current status of work, finished or future mining activities are grouped depending on the orders, type of equipment, type of material,
and location. This improves the decision making for the automatic scheduling of the production cycle by sending feedback on work progress, together with reasons for machine stoppages, from each operation to the fleet management system.

**Assumptions and limitations**

Formulating mathematical models to represent machine environments is very complex and subject to complex policies and rules related to sequence-dependent set-ups (Sivakumar 2001). The level of detail is an important part of the design, as it may increase the credibility of the model but also increase the complexity and execution time (Fowler and Rose 2004). The scheduler’s primary function should be to reduce the time and effort necessary to produce feasible production schedules and to guide production management in the decision-making process. In the scheduling system, multiple tasks/activities are glued together, creating different recipes for different sets of tasks.

In the present design, the physical transportation of jobs and tools between machines is not modelled, and transportation related to moving the machines is not yet incorporated in the scheduler. It is also assumed that the operators work equally efficiently. By default, resources are assumed to be available when required. At the moment, the interaction between the simulator and the scheduler excludes breakdowns, operational availability, and location priorities. The designed system was tested on a small-scale operation (8 production faces). The next stage is to add new features and test the system in a larger environment (whole mine). Additionally, SimMine does not adhere to any communication standards, whereas ABB Ability OMS adheres to the ISA-95 standard; thus synchronization of the model representation is not a unified process. In principle, SimMine holds the necessary information to generate a unified process model in ISA-95, including, for instance, the processing times, machine parking locations, and production cycles. However, the way the information is stored in SimMine is not the same as specified in ISA-95, since some of the details may be cumbersome to express in ISA-95. Therefore, a solution might be to use the Open Mining Format (Cuddihy 2017) to express detailed data, such as 3D block model coordinates and properties.
Concluding remarks

This study integrates a closed-loop ABB Ability OMS scheduling system with a SimMine simulation model. The proposed system can act as a test bed and a decision support system. This enables the user to evaluate the schedules produced by the scheduling platform without interacting with existing systems or processes. When the scheduler is connected to the simulator, the mine operation centre can continue to update and use the simulator to test different algorithms and procedures/rules applied to the schedule or to see the results of applying a specified production schedule in the simulator beforehand. In this way, the process control system can be tested in situations that could occur in a real environment, rather than in simple predefined scenarios. This opens the possibility to examine any effects on the process control system compared with a case that is not driven by the scheduler (i.e. the current situations in mines, where the planning of activities is highly manual).

Future development

An ongoing challenge in this project is to create a synchronized model representation that adheres to standard communication protocol. A future consideration might be to create a standardization protocol that could be used in the wider spectrum. Additional factors to consider are the rock mechanics, geological mapping, media, preventive maintenance, and material buffer capacity in different locations.

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