APPLICATION OF SIMULATION TECHNIQUES IN DEVELOPMENT PLANNING FOR CAVING METHODS

by

Zhen Li

B.Eng., Kunming University of Science and Technology, 2008

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF
THE REQUIREMENTS FOR THE DEGREE OF

MASTER OF APPLIED SCIENCE

in

The Faculty of Graduate Studies

(Mining Engineering)

THE UNIVERSITY OF BRITISH COLUMBIA
(Vancouver)

July 2012

©Zhen Li, 2012

Abstract

The mining industry globally is moving towards exploiting more mineral deposits by underground methods for several reasons. Large scale underground block/panel caving mining methods are becoming more popular due to the low operating costs associated with economies of scale. However, the planning for a caving mine is very challenging.

Simulation techniques have been used successfully by many industries for a long time. They have proven to be valuable in assisting the mine planning process, forecasting the performance of modeled systems, and testing alternatives at very low cost. In this research, simulation techniques were applied in the planning phase of a panel caving mine. These techniques were based on the existing experience as well as new software technology development. A state-of-art mine development simulation software package, SimMine®, was used as a tool for this study.

Oyu Tolgoi is a large copper-gold complex located in southern Mongolia. It contains the Hugo North deposit which will be extracted using the panel caving method. Pre-production development (PPD) will involve over 40 km of lateral development and 70,000 m^3 of massive excavations. So the PPD time and cost will be significant. The global mining industry has only limited experience to ensure effectively the design and planning for such complex, large scale projects. A case study of the Hugo North Lift 1 PPD is the focus for the simulation outlined in this research thesis.

A simulation model was developed for the PPD planning. This was found to more accurately predict long term lateral development and mass excavation rates and scheduled ventilation requirements. The process of simulation was significant in enabling the optimization of development planning and equipment selection. There appear to be considerable opportunities for simulation of such planning aspects in mining. This research aims to contribute to future software development that delivers more reliable and functional simulation tools for mining engineers. These should realize significant safety, financial and environmental advances through improved scheduling for PPD in the next generation of large, complex underground mines.

Table of Contents

List of Tables

List of Figures

Acknowledgements

Primary thanks must go to my supervisor, Dr. Malcolm Scoble for his supervision of this research project. Also, I am most grateful to Ms. Jo-Anne Dudley, principal advisor of the Oyu Tolgoi (OT) Project, for her supervision and advice on this project, and in particular for giving me the opportunity to work with the OT study team based in Vancouver. Without their support this research would never have become a success.

In addition, I greatly appreciate the help of the other professors in the Norman B Keevil Mining Department who provided me with excellent guidance and advice, namely Dr. Scott Dunbar, Dr. Dirk van Zyl, Dr. Robert Hall and Dr. Davide Elmo.

I am also indebted to Ivanhoe Mines, Rio Tinto PLC and Mitacs for their generous financial support through a collaborative research funding program which covered a substantial portion of my living and tuition expenses during the term of my graduate studies.

Secondly, I want to thank everybody in the OT study team in Vancouver. I really appreciate the opportunity to be involved in this world-class project and the chance to work with such a high quality team in my early engineering career. I would like to take the opportunity to thank Mr. Troy Newman, study manager; Mr. Jeremy Wolgram, planning engineer; and Mr. Amri Sinuhaji, senior mining engineer, for their help in improving my understanding of this complex project and their valuable comments on the simulation model. Thanks are also due to Mr. Mungunshagai Byambaja and Mr. Batkhuu Batjargal, junior mining engineers for collecting activity time data from the Oyu Tolgoi site.

Finally, I would like to dedicate this thesis to my parents and thank them for their constant encouragement while I was completing my postgraduate studies.

1. Introduction

This chapter reviews the background to a research project which has addressed the development of new simulation techniques and their application to planning the construction of the next generation of very large and complex underground block and panel caving mines. The Hugo North Lift 1 Cave, planned for extraction in the Oyu Tolgoi Mine, Mongolia, was used as a case study.

The chapter begins with a definition of the simulation technique and its application areas, focused particularly in underground mining. It then goes on to discuss the global mining context and the caving mining method. After then introducing the nature of the Oyu Tolgoi Mine case study it presents the nature of the research and objectives of this thesis.

The chapter concludes by outlining the thesis content and identifying the potential value of this research.

1.1 Background

1.1.1 Simulation Techniques

(1) Definitions of Simulation

Simulation is the imitation of some real thing, state of affairs, or process. The act of simulating something generally entails representing certain key characteristics or behaviours of a selected physical or abstract system. (Banks, 1998) Schmidt and Taylor (1970) proposed that “A *system* is defined to be a collection of entities, e.g. people or machines that act and interact together toward the accomplishment of some logical end.”

Simulation is the process of creating a model (i.e., an abstract representation or facsimile) of an existing or proposed system (e.g. a business, mine, watershed, forest, or organs in one’s body) in order to identify and understand those factors which control the system and/or to predict (forecast) the system's future behavior (Banks, Carson II, & Nelson, 1996).

A model is a representation of an actual system. The limits or boundaries of the model that supposedly represent the system need to be clearly understood (Banks, Carson II, & Nelson, 1996). The model should be complex enough to answer the questions raised, but not too complex.

In this research the modeling system is discrete (i.e. variables change instantaneously at separated points in time), dynamic (the system evolves over time), and stochastic (involving probability inputs). That is, a discrete-event simulation model "is the modeling of a system as it evolves over time by a representation in which the state variables change instantaneously at separate points in time" (Law & Kelton, 2000).

(2) Application Areas of Simulation

Simulation techniques have long been used successfully by multiple industries. but have been mostly utilized in manufacturing systems design and in improvements of plant operation (Law & McComas, 1992). Simulation is the most widely used operational research and management science technique and its application areas are numerous and diverse. The typical application areas can be divided into manufacturing and material handling, public systems, and service systems according to Banks et al. (1998). Examples of the major application areas are listed below (Law & Kelton, 2000):

- Physics, chemistry and biology
- Medical and health
- Economics and finance
- Social science
- The military
- Engineering
- Natural resources

Despite the history of such diverse application there has been comparatively little development of simulation as a means to assist in underground mine planning.

1.1.2 The Global Mining Context

Future mineral resources in a global context are tending to become deeper and the demand for metals is expanding. This will lead to a significant increase in massive underground mining. Block/panel caving methods can facilitate such mass production with a comparatively low cost.

Therefore, this mining underground mining method has become more popular in recent years for extracting large mineral deposits. It is mainly used for mining underground copper and gold, but other minerals such as molybdenum, diamond, asbestos or coal can also be mined by the caving method if the orebody's geotechnical condition allows the employment of this mining method.

Block/panel cave mining refers to all mining operations where the orebody has caved naturally after undercutting; the caved material is then recovered through drawpoints. This includes block caving, panel caving, inclined drawpoint caving and front caving (retreating brow cave) (D. Laubscher, 1997-2000). In the past, the block caving method was only used in deposits with very low quality rockmass conditions, since these can be easily caved and the method provides very good fragmentation. Recently, however, block/panel caving can be used in competent deposits due to an improved understanding of caveability and fragmentation, well managed drawing strategies, and improved technologies in underground mobile equipment such as with the larger load-haul-dump (LHD) machines and enhanced drill rigs for secondary blasting (D. H. Laubscher, 1994).

Figure 1-1 Anticipated Increased Percentage of Ore to Concentrator by Mining Method (Moss, 2010)

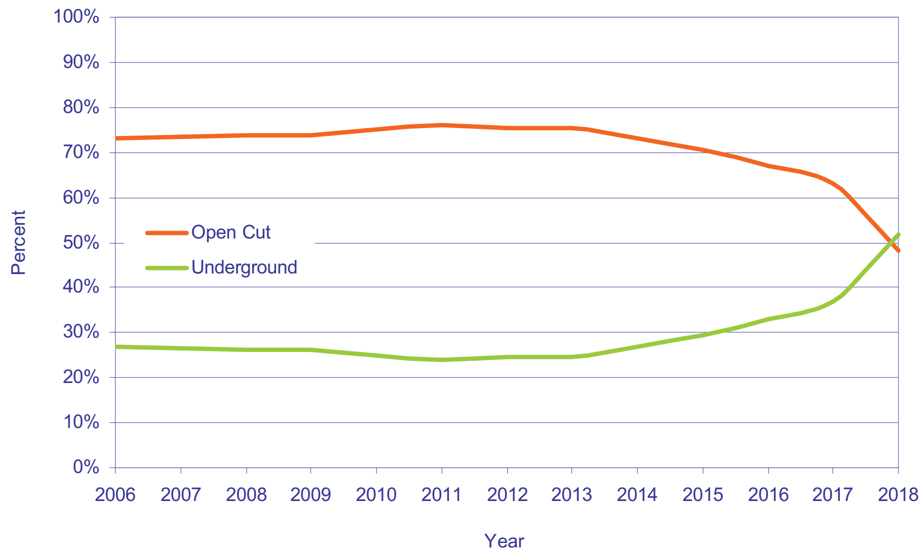
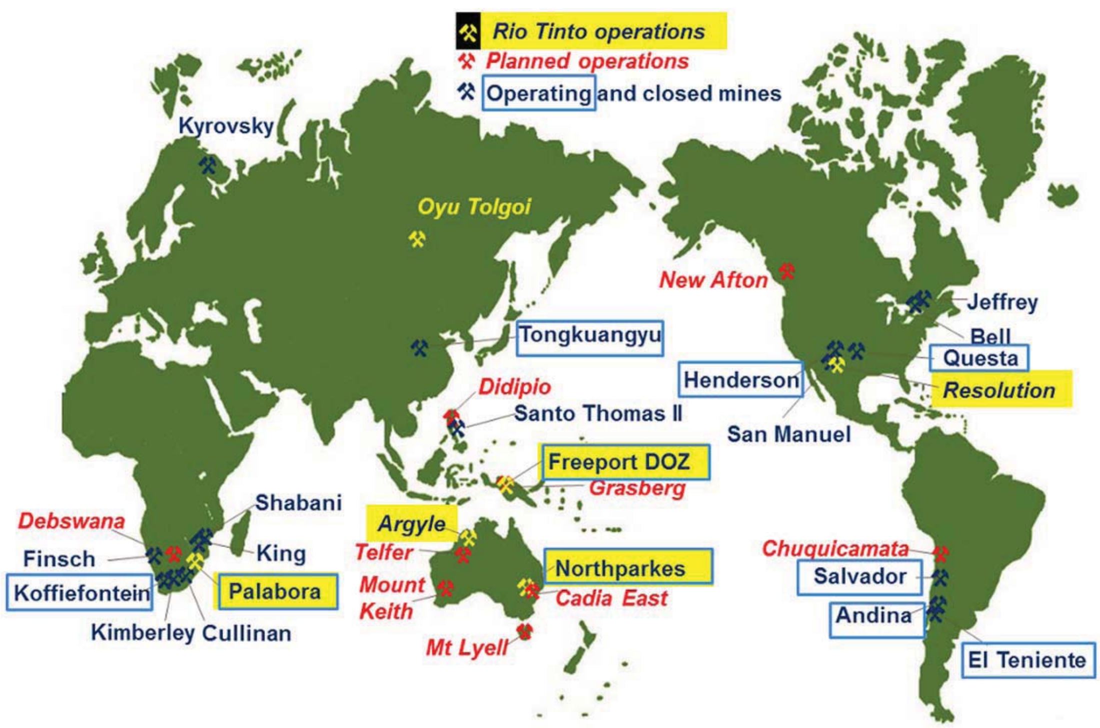


Figure 1-1 presents the anticipated trend of copper ores produced by open pit and underground mines by percentage. This chart was based on Brook Hunt data which was prepared for Rio Tinto Minerals (Moss, 2010). The data predicted that there will be a decrease in open pit production compared to that mined underground. Trends indicate that by 2018 to 2020 more than half of global copper production will be from underground. There are several reasons for this change: a) More reserves are situated at a greater depth and these need to be extracted underground; and b) Open pit mines are going to reach their pit limits, after which they are then closed or transition to underground mines (D. H. Laubscher, 1994). However, to achieve this outcome, underground mining must become more productive and achieve a competitive economy of scale. Because caving is the lowest cost underground mining method, several large-scale low grade open pit deposits need to be examined to determine the feasibility of transitioning to this method (D. H. Laubscher, 1994).

The size of underground mines has increased significantly over the past years. A typical large open pit mine will produce 50,000 to 200,000 tonnes per day (tpd). But large underground block/panel caving mines will eventually rival or exceed the production of open pit mines. For example, the Grasberg mine is planned to produce copper and gold all from underground deposits including Grasberg, MLZ, DMLZ, Kucing Liar and DOZ Block Caves after 2016. The peak production will reach 250,000 tpd (Freeport, 2012). The Hugo North Lift one at Oyu Tolgoi in Mongolia will produce 90,000 tpd, and with the open-pit mine the Oyu Tolgoi mining complex may achieve 170,000 tpd (Ivanhoe Mines Ltd, 2012).

The design and planning of such a large underground mine, however, will be technically very challenging. The mining industry lacks adequate experience in using simulation for ensuring effective design and planning for such complex, large scale projects.

Figure 1-2 World's Major Block and Panel Cave Mines (Moss, 2011)



There are around 30 block caving mines throughout the world, among which ten are under study and construction, nine are currently operating and about eleven have already been closed. Figure 1-2 shows the major block caving and panel caving mines around the world.

1.1.3 Block Caving Mine Planning

(1) Design and planning process of block/panel cave mines

After a large underground deposit has been discovered that is conducive to caving extraction method then Figure 1-3 shows the ensuing design and planning process of block/panel caving mines. The first step is to understand the geological and geotechnical characteristics of the deposit. After that the mine design and planning process continues. The design phase includes:

- Assessing caveability and fragmentation
- Designing the undercut and extraction levels
- Designing the ore handling system
- Designing the ventilation system
- Designing infrastructure layout
- Defining a draw strategy
- Identifying hazards and risk management

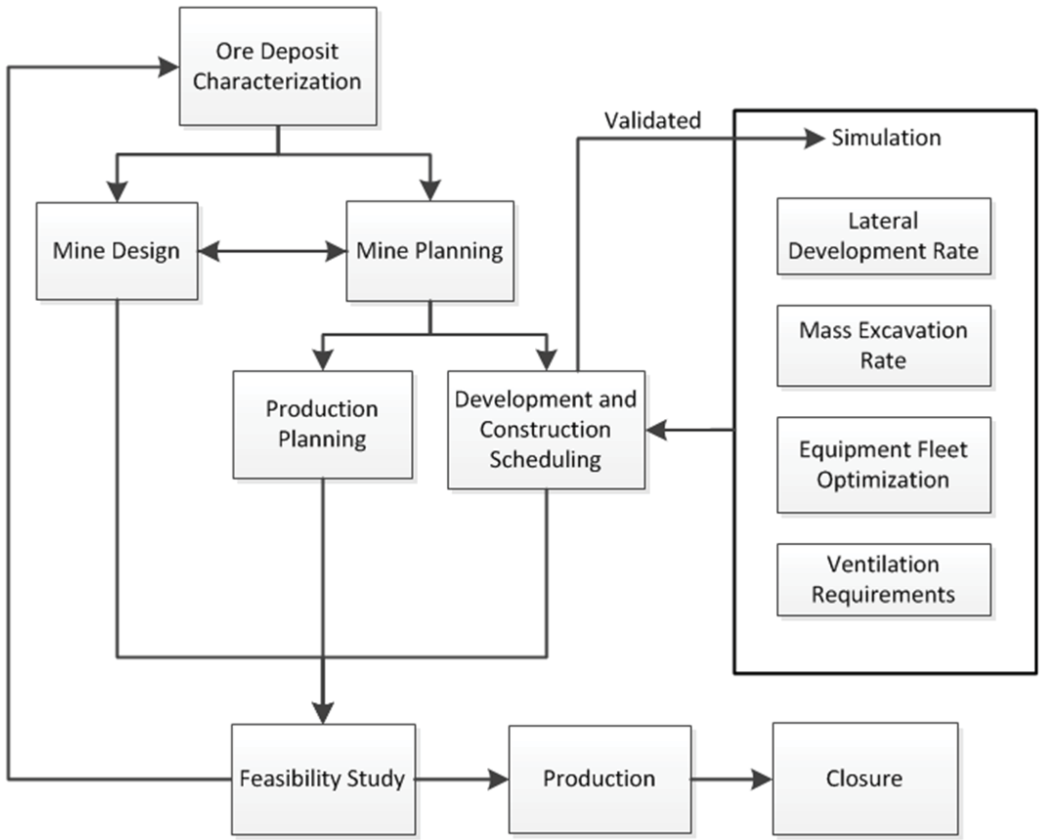
At the same time, the proposed mine planning and design phases are frequently interactive, relating to two major aspects:

- Development and construction scheduling
- Production planning

(2) Simulation application in the mine design and planning process

In this thesis, simulation was implemented in the development and construction scheduling phase of a large and complex underground mine in Mongolia. The information that arises from simulations, such as development and construction rates (DCR), optimization of development systems, and ventilation requirements are a critical foundation needed to support the mine development and construction scheduling. After the development and construction schedules have been completed it is possible to go back over the process and validate it with the simulation model, for example, through checking the critical path, crew build-ups, equipment purchase (start) dates, and ventilation capacity. When both the mine design and planning have been completed then the Feasibility Study of the proposed mine can be completed.

Figure 1-3 Design and Planning Process of Block Caving Mines



(3) Block/Panel cave mine development planning

There are two phases in underground mining: mine development and production. Several important things need to be considered in planning a block cave mine. A well-established development plan will ensure the later success of mine production and reduce future risks. Therefore, significant time and money warrant investment in this phase. Some of the typical questions of underground development planning are listed as follows, but there are many more questions related to the development planning process that still need to be answered.

- How can the deposit be accessed?

- How can the ore be transported to the surface?
- How can the working face and production areas be ventilated?
- Where should infrastructure be laid out?
- When can the ore handling system be commissioned?

Three aspects are very important for underground mine planning: infrastructure development and construction, as follows:

Mine development: comprises tunnels and excavations to access the orebody from the surface, allowing both equipment and personnel to travel through the structures; the space that infrastructures require to be located. Permanent mine development includes (Luxford, 2000):

- Decline or ramp development
- Hoisting shafts or conveyor drifts
- Level development (undercut, extraction, haulage, and ventilation)
- Ventilation shafts, raises, and drifts
- Ore passes
- Shaft stations
- Cutouts, muck bays, and sumps
- Major infrastructure excavations for:
    a) Workshops and storage
    b) Crushers
    c) Pump stations
    d) Refugee stations
- Boreholes for drainage and power reticulation

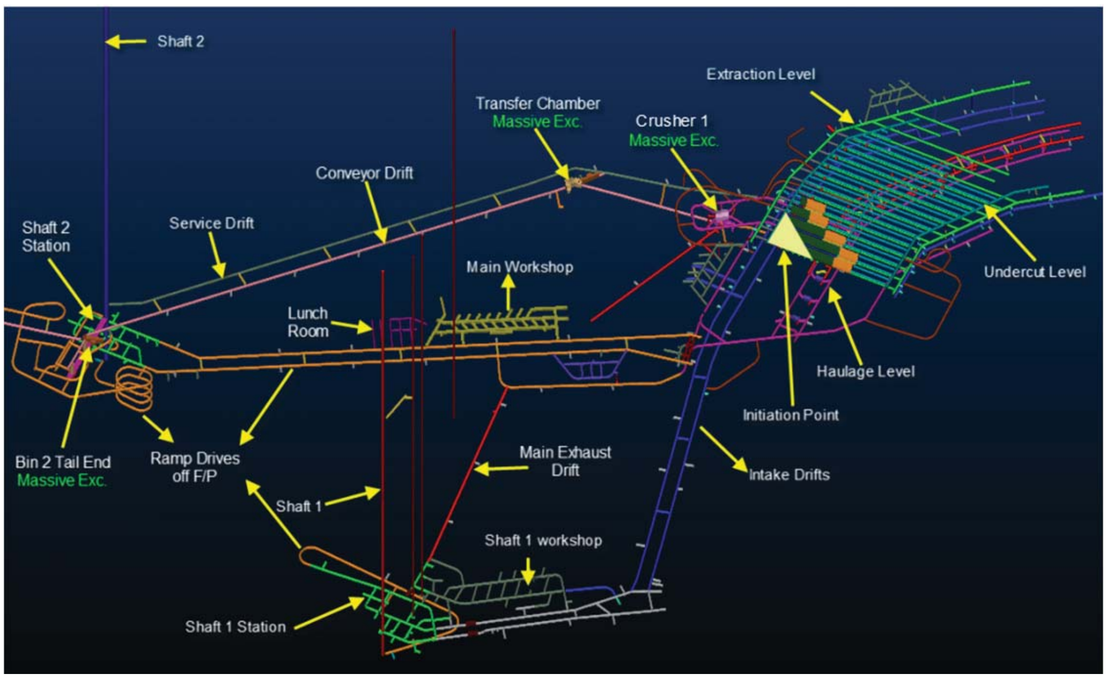
Construction: Development of the lateral and vertical tunnels and large excavations; building and installing equipment in these tunnels and chambers to make all these systems functional, e.g. sinking shafts, reaming raises, installing ventilation fans, installing conveyor belts, installing crushers, etc.

Mine Infrastructure: Facilities to allow the mine to operate:

- Ore handling systems

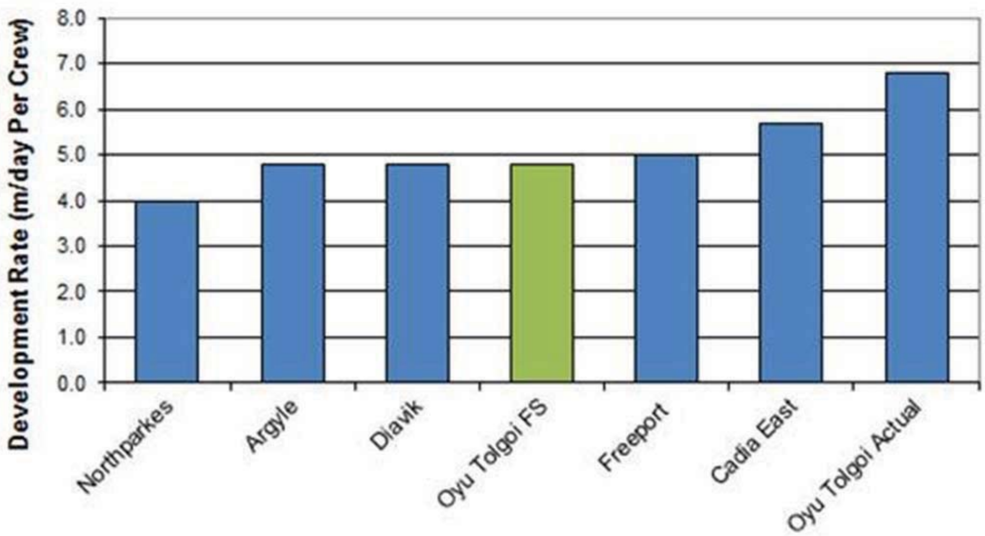
- Dewatering systems
- Power and water reticulation
- Communication and control systems
- Ventilation systems
- Workshops, lunch rooms, magazines, crushers, conveyers, wash bays, truck chutes, lube stations, etc.

Figure 1-4 An Example of the Major Development and Infrastructure of Hugo North Lift (after Wolgram, Li, & Scoble, 2012)



Development and construction rates (DCR) are critical in planning underground mines because these will ensure the timely achievement of certain scheduled milestones. Figure 1-5 summarizes examples of the recorded lateral development rate of some of Rio Tinto's underground operations. The rates were based on the average meters one crew could advance in a multiple headings work environment. An Australian example of experience in development rates is: a jumbo (drilling 3.2 m long blast holes) with at least three faces available in reasonable conditions, achieved an average development rate 10 m/d (Luxford, 2000).

Figure 1-5 Multiple Heading Development Rates of Rio Tinto's Operations (Wolgram, 2011)



*The data has been derived from Rio Tinto's operations

1.1.4 The Oyu Tolgoi Panel Cave Mine Project

The Oyu Tolgoi Mine is a joint venture between the Ivanhoe Mines, Rio Tinto and the Government of Mongolia. Oyu Tolgoi contains a number of predominantly copper-gold mineral deposits including Hugo North, Hugo South, Heruga and Southern Oyu (Ivanhoe Mines Ltd, 2010).

Hugo North, the first underground deposit to be developed, will be mined with two lifts using panel caving techniques. Lift 1 is currently being developed. A total of 200 km of lateral development, five vertical shafts ranging from 6.7 m to 11 m diameter plus one ventilation raise, and several massive excavations, will need to be developed and excavated over the life of mine (LOM) (Sinuhaji, Newman, & O'Connor, 2012). A Feasibility Study will be completed by Q3 2012 for the first 2 panels on Lift 1. It is currently projected to operate at a peak production rate of 90,000 tonnes per day (tpd).

Oyu Tolgoi is the largest undeveloped copper-gold complex located in southern Mongolia, 80 kilometers away from the Chinese border. Figure 1-6 shows the location of the Oyu Tolgoi planned mine (Ivanhoe Mines Ltd, 2012). The Hugo North

Lift 1 is accessed by Shaft 1 which is1300m below the surface. The Southern Oyu open pit deposit will begin commercial production in the first half of 2013 and the underground Hugo North Lift 1 is schedule to start production in 2016. (Ivanhoe Mines Ltd, 2012). Figure 1-7 shows a section map of the Oyu Tolgoi mining complex and Hugo North Lift 1.

Figure 1-6 Location of the Oyu Tolgoi Project (Ivanhoe Mines Ltd, 2010)

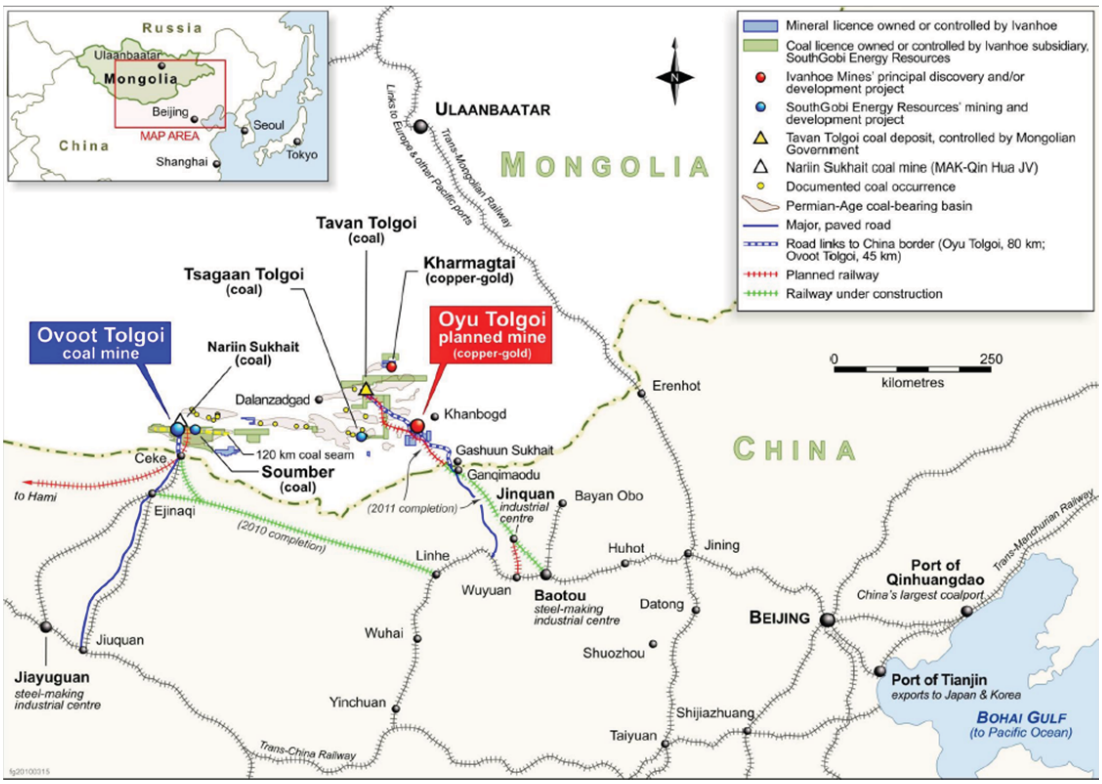
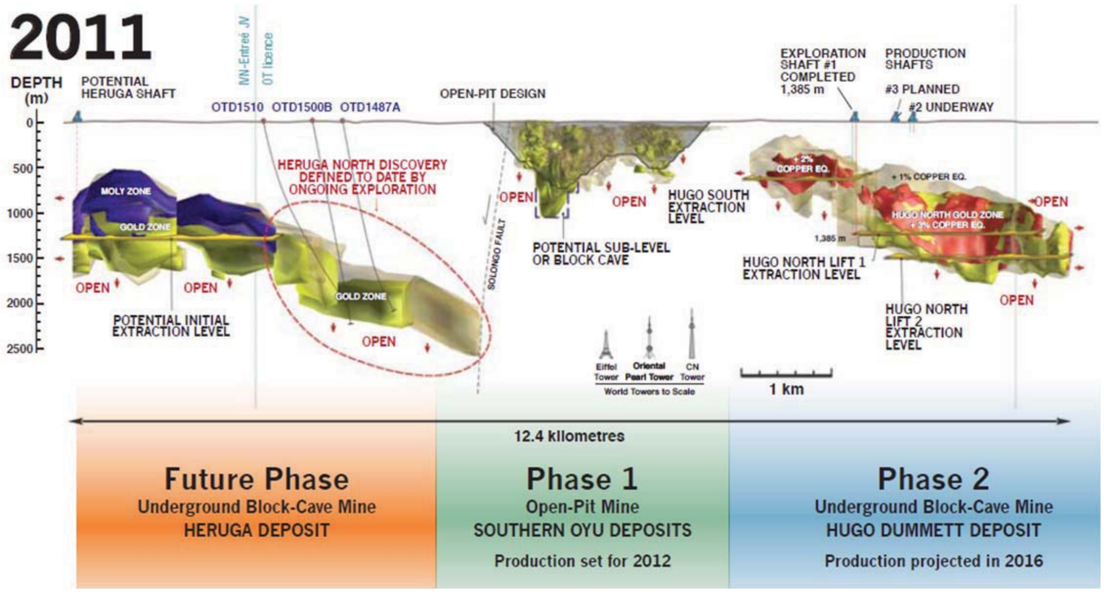


Figure 1-7 Development Phases of the Oyu Tolgoi Project (Ivanhoe Mines Ltd, 2012)



1.2 Problem Statement

The conventional scheduling of underground mines is traditionally based on the experience of mine planners and empirical data from past or existing mines. For example, the development rate in the pre-feasibility study of the Hugo North Lift 1 was determined using first principle spreadsheet analysis and benchmarked against data from best practice mining operations. This means that the working days in a particular drift were calculated by dividing the drift length by the assumed rate and adding some contingency. The drawback of the conventional scheduling approach is that it potentially lacks in accuracy and reliability.

The case study considered in this thesis, preproduction development (PPD) of Hugo North Lift 1, will have over 40 km of lateral development, 70,000 m^3 of mass excavation, and three 1300 m deep shafts. The development and construction activities will take 25 years over the life of mine. Figure 1-4 outlines the major pre-production development and infrastructures of the Milestone 2 (MS2) feasibility study.

At its peak, underground development and construction activity will require 12 development and 6 construction crews with 180 pieces of active mobile equipment (Wolgram, Li, & Scoble, 2012). Congestion will be a critical issue with so many pieces of mobile equipment working underground, and this cannot be addressed by traditional scheduling methods. The total capital cost for this underground development will be very significant (Ivanhoe Mines Ltd, 2012). A more accurate schedule could potentially reduce the significant financial risks for such a project.

The planning of the Hugo North Lift 1 will be very challenging because of the complexity of the mine development excavation systems and mine design. In order to produce a schedule that is more reliable for such a Feasibility Study, a more scientific approach has been proposed.

Conventional scheduling methods such as the Critical Path Method (CMP) and Project Evaluation and Review Technique (PERT) are limited in application to such complex projects and need to be improved. A new adaption of discrete-even simulation, however, will be identified and evaluated in this research to deal with such planning challenges.

1.3 Research Objectives and Scope

1.3.1 Thesis Objectives

In order to address the issues outlined previously three major objectives were proposed for this study:

- Evaluate the use of simulation techniques in planning large and complex underground mining projects
- Contribute to the development of a simulation model applicable to the development planning phase of cave mining systems, focusing particularly on development and construction rate performance
- Advance cave mine planning precision and reliability

After the research work was started, several other objectives were later proposed to improve development performance and the simulation program.

- Improve the current development of simulation software for the mining industry
- Optimize the selection of development equipment fleet and crews
- Test different development scenarios to maximize development rates
- Determine equipment utilization and ventilation requirements during pre-production period (PPD)

1.3.2 Study Scope

Several study milestones of the Oyu Tolgoi underground mine were aligned with this research project:

- Milestone 1 (MS1) was the Pre-feasibility Study (PFS)
- Milestone 2 (MS2) was the transition between the Pre-feasibility Study (PFS) and the Feasibility Study (FS)
- Milestone 3 (MS3) was the Feasibility Study (FS).

Two models were built to achieve the simulation objectives. The first model included four years of lateral development based on the MS2 mine layout and design. It was used to experiment with the lateral development and mass excavation rates based on different heading profiles and support requirements. Several sensitivity and trade-off tests were evaluated in this model after it was completed. This model is called the MS2 model in this thesis.

The second model considered three years of development based on the MS3 mine layout and design. It was used to analyze in detail the equipment utilization and validate the schedule of the first two years from December, 2012 until the end of 2014. This model is called the MS3 model in this thesis.

1.4 Research Project Overview

When this research project was initiated Oyu Tolgoi was under the MS2 but the outcomes of this research facilitated the scheduling of the MS3 Feasibility Study of the Hugo North Deposit.

The model was utilized to determine trends in development rates as working areas would expand away from the main access points, to schedule ventilation requirements, and to optimize equipment fleet characteristics. Simulation outputs assisted with equipment purchase schedules and decision making regarding schedules' critical paths, and in identifying potential risks in pre-production development. This thesis outlines this form of the application of mine development simulation tools on large underground development excavation systems. It reviews the main findings from model construction, experimentation and optimization phases. A comparison with benchmarked rates and key implications for the project are described.

The challenges of this research lie in the uncertainties resulting from the large number of variables, the time effect over the decades of the development period, and the different options for mine designs. Simulation tools were evaluated, selected and adapted for the Oyu Tolgoi Mine development study because they offered the benefit of a) testing various hypotheses at a fraction of the cost of performing the actual activity, b) revealing how real-world activity performs in different scenarios, c) potentially minimizing the associated risks by optimizing the design and planning phases, and as an additional incentive d) enhancing the safety of mine development.

1.4.1 Thesis Outline

Chapter 1 provides the background information on the simulation process, current trends in the global mining context, and the planning of block/panel caving mining systems. It also introduces the Oyu Tolgoi underground case study.

Chapter 2 refers to and discusses previous work on relevant aspects related to this thesis. The areas reviewed include the history of the simulation technique and its pros and cons, and applications of simulation in open pit and underground mines. This review mainly focuses on underground mining, especially block/panel cave systems.

Chapter 3 outlines the methodology used in this research. Firstly, part of the chapter shows the data collection and preparation processes which were based on the site activity time study and statistical analysis. It then describes the simulation program employed in this research and the adopted modeling processes.

Chapter 4 details the simulation model buildup process and describes each input related to the model and the simulation environment.

Chapter 5 discusses the model verification and validation steps and the results from base case and experimentation tests. It also lists findings from simulation modeling.

Chapter 6 details the conclusions of the study, discussing the limitations of the simulation model and recommending future research.

Appendix I documents the time and motion study which was the main data source related to this research.

Appendix II documents all the test outcomes and screenshots from the simulation runs.

1.4.2 Rio Tinto Mitacs Collaboration

Rio Tinto's Oyu Tolgoi Study Team based in Vancouver initiated this research project at the end of 2010 to explore and evaluate the characteristics and benefits of a new simulation technology in planning the Hugo North Lift 1 cave. The project was a collaboration between the Mining Engineering Department of the University of British Columbia (UBC), Mitacs-Accelerate, and Rio Tinto. The latter two parties jointly funded the research project. Mitacs-Accelerate is Canada's premier research internship program. Rio Tinto PLC. is a leading global mining company.

1.4.3 Significance and Contributions of the Research

This is believed to be the first thesis research that attempts to explore the simulation of development and excavation systems in an underground cave mine. Some researchers have applied discrete event simulation to underground mining, but due to the complexity of such modeling few were related to cave mine development planning. This thesis aims to be a valuable reference for future researchers and mining engineers who are interested in this topic. This work stems from a significant underground development simulation based on a case study of a world-class block caving project.

The most important contributions of this simulation study are: a) it assisted the scheduling process of an important Feasibility Study; b) it provided the ability to more comprehensively understand the inherent variability in the development process of caving mines and to potentially mitigate project risks; and c) it contributed to evaluating and advancing simulation techniques for planning massive underground mines. Also, reviewing the model construction phase should contribute some valuable learning to future researchers and mining schedulers. Furthermore, it is felt that the simulation program that was used in this thesis has been benefited in its evolution from the experiences gained in this research project.

2. Literature Review

This chapter describes the history and current status of simulation applications in the mining industry. This literature survey includes the topics of simulation in mining, underground mine planning, operations research in mining, discrete-event simulation in open pit and hard rock underground mining, and caving mine planning. Several papers related to simulation in underground mining, particularly block/panel caving, are reviewed in detail.

The purpose of this review is to understand the principles of prior and current research and to foster interest in the performance of future simulation applications in mine planning. Should more time and study be spent in investigating this area, the industry would be well rewarded with the opportunity for conducting interesting and important simulation applications, potentially saving significant cost and enhancing safety.

Publications related to this research topic can be found in several conference proceedings and journals, and in dissertations and theses. Most research papers on block and panel caving are published in the MASSMIN (Mass Mining) conferences, although only a few of them are related to the simulation technique. Publications on simulation, operations research (OR) and computer modeling were mostly based on specific conferences. These include the Application of Computers and Operations Research in the Mineral Industry (APCOM) Symposium, which has, during the last 50 years in the mining industry, been the major forum for the presentation and discussion of computer and OR technique applications, including simulations (Panagiotou, 1999). Also relevant are various OR and management science databases such as INFORMS.

This literature review has only discussed a few specific publications of simulation research in mining, with a focus on simulation and underground hard rock mine planning. The areas that related to the simulation application in mining have quite extensive literature coverage, although most articles are on open pit optimization, production scheduling, and equipment reliability, with only a few related to block/panel caving mines.

2.1 Simulation Technique

2.1.1 History

The first simulation employed in mining appears to have been performed in the late 1950s at the Kiruna underground iron mine in Sweden. This simulation was created manually. The study modeled the train-transportation system, and consisted of a track way plan, bins for the storage of ores, a signal system, and train movement and dispatching. The simulation took place at a speed 200 times that of the actual train speed; 20 trains were utilized in the system. The results could be displayed in a plot involving the relationship between transportation capacity and the number of trains used (Elbrond, 1964).

Lynch and Morrison (1999) reviewed the history of simulation in mineral processing along with milestones in the PC development history (Lynch & Morrison, 1999). Early modelling (prior to 1960) was concerned with the design and optimization of circuits (Panagiotou, 1999).

In its early stages, the development of simulation modeling was limited by computing power and the complexity of computer programming. However, in recent years, excellent software products have been developed which are made ready–to-use through built-in features which before were required to be programmed (Law & Kelton, 2000). As a result, interest has increased in the application of simulation modeling to the mineral industry (Sturgul & Li, 1997).

With computing power increasing rapidly, it has been possible in recent times for high-precision and more complex simulation models to be constructed. Simulation software has thus become more powerful, accurate and easier to use. Depending on the capabilities of the software, the simulation can also be visualized through animation techniques. Some simulation software can provide a 3D view of the model, facilitating viewing of the simulated systems and the detection of design and plan flaws (Law & Kelton, 2000) and (Sturgul & Li, 1997). More recent studies have attempted to simulate larger portions of the mining system or even complete mines.

2.1.2 The Pros and Cons of Simulation

Sturgul and Li (1997) stated that the advantages of simulation are to provide a future forecast, to permit management an understanding of the underlying problems in the system, and to allow the company to make critical decisions.

Simulation may have its own limitations and drawbacks, including simulation errors, which may greatly impact on results. The more programming and logistics a simulation model has, then the greater the possibility of simulation errors occurring in any model. These benefits and drawbacks have been considered by several researchers, including Banks, Carson II, & Nelson, 1996; Law & Kelton, 2000; Pegden, Shannon, & Sadowski, 1995; and Schriber, 1991. These may be summarized as follows:

Advantages:

- Simulation allows the user to test designs without committing resources to their acquisition
- Once a valid simulation model has been developed, hypotheses can be tested at a fraction of the cost of performing the actual activity
- Simulations demonstrate how the real-world activity (i.e., underground mining) performs in different scenarios
- Simulation aids in an understanding of the interactions between variables and an analysis of the problems involved
- Simulation eliminates uncertainties concerning the impact of changes
- Simulation identifies the inefficiencies, constraints and risks associated with proposed systems
- Simulation helps management teams to make better decisions

Disadvantages:

- Simulation errors cannot be avoided during model construction
- Model building requires special training, and the development of complex models may require great time and effort
- Simulation results may be difficult to interpret
- If the inputs are incorrect, the outputs will have no value (i.e. “garbage in” is equal to “garbage out”)
- The model is, nevertheless, only a simulation. The actual performance of the designed system will never be realized until its designs are executed; e.g., a conveyor system’s true capacity will only be known after the system is in full operation.

2.2 The Application of Simulation to Mining: Case Studies

During the past decade, very few researchers have worked in the field of block caving mine simulation. Moreover, most literature covers traditional open pit or drill-blast underground mining methods. Most of the simulation studies have focused on solving problems related to such areas as mining equipment systems, fleet efficiency and reliability assessments, production scheduling, and ore handling systems. They have been applied to mining operations focused on selected areas of the mining operation (Greberg & Sundqvist, 2011), while little work has been completed on determining the development rates and optimization of the development process in block caving mines. This section describes some of the general work on the application of simulation in the mineral industry.

2.2.1 Open-pit Mines

Simulation has previously been used to evaluate mine throughput and to schedule short- and long-term production in open pit mines. Fytas et al. (1993) employed simulation techniques in open pit mines. The study determined the long-term production of ore and waste in each period subject to sequencing constraints. The production schedule was also constrained to minimum and maximum production limits, processing capacity, strip ratios, and ore grading. Dyer and Jacobsen (2007) validated load and haul capacity during peak periods at the Cortez Gold Mine. The simulation model was constructed using GPSS language software. The results showed that current truck and shovel fleet were well matched and that planned tonnage could be achieved at peak time. The use of simulation mitigated the risks of over/under-predicting production and played a key role in the feasibility study stage. Ben-Awuah et al. (2010) developed a discrete-event simulation model for open pit production scheduling using the SLAM simulation language. The model analyzed the capacities of mining and mineral processing, crusher availability and stockpiling and ore blending strategies in an iron ore case study. The bottlenecks in the system were identified and addressed in the paper. The uncertainties of long- and short-term production schedules were presented and linked in the simulation.

Techniques can also be used to simulate open pit mining equipment. Bradley et al. (1985) used simulation at the Powder River Basin mine, Wyoming. The study performed was to test trade-offs between the number of loading trucks, storage capacity, and production and train filling rates. Data was measured from the mine site. Agioutantis and Stratakis (1998) described a simulation model utilized to study the performance of continuous surface mining equipment at the Northern Field lignite mine in northern Greece. The system included bucket wheel excavators (BWEs) which excavated lignite and waste material, a number of conveyors and five dumping subsystems. Micro Saint, a simulation package allowing simple model construction as well as animation, was used. Shovel/truck systems are the major material-handling systems used in hard-rock open cast mining, quarries or earthmoving operations. Simulation results of equipment’s operating times and down times match actual data, with small deviations. The model was used to evaluate alternatives before building or modifying the actual system, in order to forecast the behaviour and performance of various strategies. Frimpong et al. (2003) presented a robotization and stability control (RASC) model for 400 tonne CAT797dump trucks used in oil sands mining. Truck dynamics in this RASC model was analyzed by a FORTRAN based simulation program. Oraee and Asi (2004) developed a simulation model to analyze truck ramp-up schedules based on a case study of the Songun

Copper Mine in Iran. The study was performed to determine whether the production schedule would meet the desired requirements of the mill.

2.2.2 Underground Mines

Alternate underground mining methods or ore handling systems, e.g. conveyer belt or mucking and haulage equipment, can be evaluated using simulation (A. M. Newman, Rubio, Caro, Weintraub, & Eurek, 2010). Simulation tends to have been most widely applied to underground ore handling systems, being used to measure and analyze the materials movement and logistics for these systems.

Topuz et al. (1982) simulated the haulage systems for an underground coal mine, comparing two different types of haulage equipment: conventional (using a conveyor belt) and diesel (without a conveyor belt) shuttle cars. The differences between the two haulage systems were haulage distance, travel speed, and equipment availability for coal transport, as well as the capacity of the feeder where the shuttle car dumps coals, were evaluated in the simulation. Feeder discharge rates, the amount and type of haulage equipment, and haulage distance were experimented on in a room-and-pillar mining case study.

Sevim (1987) created a dynamic simulation for a coal mine to model a system that transported coal with water. After the coal was mined from the face, it was then pumped directly into the coal preparation plant using water pressure. When this cycle was completed, the equipment would be repositioned for the next round. A delay could occur during the process because of equipment breakdown. Two alternatives were considered in the study: merging pipelines from different mine areas, and adding a surge tank into the piping system to store the slurry. The model was experimented on in two cases, namely, those of room-and-pillar and longwall coal mines. Results showed pumping times, surge tank overflows, the concentration of slurry in the processing plant, and operating costs.

Mutagwaba and Hudson (1993) constructed a simulation model to assess underground transportation systems. The model evaluated various components of these systems. Simulation was applied to a mine in the UK. Based on the mine's layout, hoisting systems, and target production rate, it assisted the mine in selecting the right equipment to optimize performance and reduce costs.

McNearny and Nie (2000) performed a simulation study on conveying systems in coal mines. The mines employed longwall and continuous mining methods. Coal was moved from a mine face to the surface by conveyors. Simulation techniques were able to balance the costs and performance of the conveyors. The study identified the bottlenecks in the conveying system and tried to eliminate them by adding in surge bins. Conveyers of various sizes and speeds were tested, using the model, on a mine in southern Utah, and the optimized choice was ascertained by simulation techniques, potentially increasing productivity by 13%.

Hall, B. E. (2000) discussed the requirements for the successful simulation modeling of mining systems. The first case study in this paper was performed to investigate an underground truck haulage fleet used on a decline. The objective was to determine the optimum number of trucks required to achieve the production targets over a number of years as the mine extended deeper. The study concluded that, should the daily production rate equal that of the target even while being backfilled with cement, it would be necessary, in later years, to incorporate an additional two trucks. Adding more trucks to the fleet would not contribute to production because the system is limited to the developmental, drilling and blasting capacities.

Simsir and Ozfirat (2008) presented a simulation model as a case study for a Turkish coal mine which employed the longwall mining method. The model assessed the efficiency of crushers, loaders and conveyors. However, it did not take into consideration the geotechnical conditions within the mine that affected the performance of some longwall mining equipment.

Salama and Greberg (2012) used SimMine® simulation software to study the haulage system in an underground mine which operated with LHDs and dump trucks. The model evaluated the effect of increasing the number of trucks on the overall mine throughput. The simulation was conducted on three different production levels.

On each production drift, simulation was run at the end, center and near the loading point to determine the effect of reducing the tramming distance for the production cycles. The study found that the optimum combination of loading and haulage equipment, 1 LHD, could be assigned to load two trucks if the production stopes were near the dumping point or on upper levels; and three trucks if the stopes were on the mine's lower or middle levels.

Next, simulation was applied to measure and forecast underground mobile equipment's performance, including its availability, utilization and efficiency. It also demonstrated the technique's ability to assist mining engineers and management in equipment selection.

Runciman (1997) explored WITNESS®, a discrete events simulation package to model such underground activities as load-haul-dump, ground support, drilling and charging in the INCO Limited's Copper Cliff operations. He found that tele-remote systems can increase underground development efficiency by up to 45%.

Vayenas et al. (1998) developed a simulation model to study the interaction between the stope designs (geometry and sequencing), development and machine systems of vertical retreat mining (VRM) using 3D simulation software Automod™. They measured the productivity and reliability of such mining equipment as loaders, trucks and drill rigs. Two scenarios were studied in stopes: a conventional mining technique using manned equipment and an automated mining method using tele-operated equipment. They found that a 15% increase in equipment utilization could be achieved through tele-operating . The key difference between the two tests was the travel time to and from the workface. Also, a 23% increase in production was forecast for tele-remote LHDs.

Kocsis (2009) created a discrete-event simulation model using Automod™ in collaboration with Penguin Automated Systems. This activity based model simulated the mining process of an underground VRM mine, and estimated the life of mine (LOM) intake air volume requirements. Based on the outcomes of this Automod™ simulation, the "life-cycle" airflow demand schedule was determined for "traditional" versus “activity-based” ventilation practices. In addition, a ventilation simulation program was used to solve and balance the ventilation system based on the LOM fresh air demand schedule.

Hall, R.A. (2000&2003) created a reliability and maintenance model for underground haulage equipment. In his model, he applied discrete simulations for a proposed new face with: case 1) one loader and three trucks; and case 2) one loader plus one spare and three trucks employed as a production haulage fleet. The model was programmed in Raptor™, a reliability simulation software package. He compared the results from simulation and heuristic approaches, and found that the simulated results were more conservative than the heuristic ones. The model was used as a tool for managing equipment selection and maintenance. His study illustrates the potential use of this type of approach in the equipment decision making process. He suggested including cost studies in simulation models and using them to run production and mine planning studies.

Vayenas and Yuji (2005 & 2008) conducted a study to assess the impact of equipment failure on a mine’s production throughput. The model was built by a 2D simulation software package, Simul8. They included random failures to the LHDs on a two-level sublevel, stopping mine operations, and discovered that this scenario would interrupt the development cycle and result in a lack of hauling capacity.

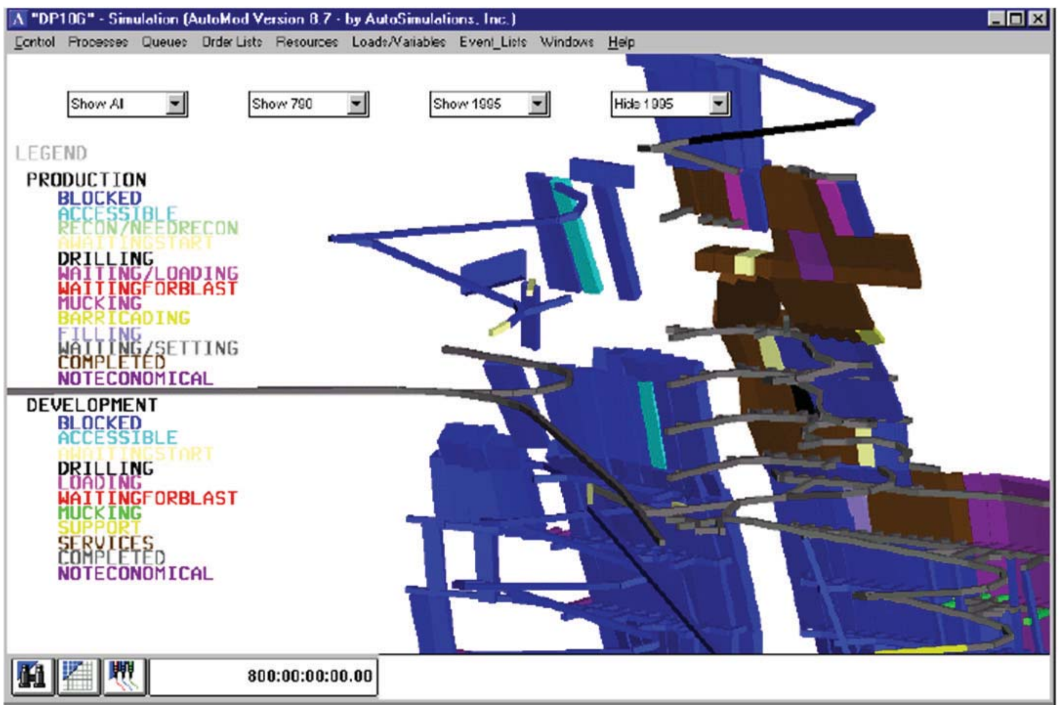
Last but not least, simulation was employed to access the underground development and production processes. Parameters, such as development rates and equipment capacity, were able to be evaluated using simulation techniques.

Brunner et al. (1999) described a discrete-event simulation model utilized to represent the development and production process in a “CSSM” underground mine in Sudbury, Ontario. The study was used to support the decision making process of mine planning and design changes.

The group’s first model was built in 1995 to mimic the material handling system conducted from the dump point to the surface. Later that year, a more detailed “DP” model was developed to support the planning and design of a “CSSM” mine using

AutoMod™ In the model, mine designs and orebodies (entire drifts and stopes) were represented via many 3D "material blocks". The simulation evolved with time and changing colors in the blocks to show different events and status of them. In 1997 and 1998, the group revised the DP model by breaking down the development and production process. The data was collected and updated from a tested mine and short- and long-term tasks were incorporated into the resources.

Figure 2-1 Mine Geometry in the DP Model (Brunner, Yazici, & Baiden, 1999)



The 1996 intermediate version of the DP model determined that the development rates depended on the number of working faces in a non-linear relationship. The resources are allowed to work on up to three available faces. Their findings were very similar to those of this research thesis.

Their paper concluded that:

a) Simulation has proven to be a useful tool for tradeoff studies on mine designs and technological alternatives. It has the ability to test various scenarios on plans, methods, equipment and people

b) The output of simulation can be used in the cost model to assist the engineer in making the correct decisions

c) Coding and programming takes a great deal of time because there is no simulation package extant for immediate employment in underground mining

d) The first DP model used historical data of development rates as its input, so that the advanced rate and overall progress of development was reasonably close to that of the actual. However, in the revised model, the advanced rate was replaced by development cycle times and became the output. Thus, the output results of the advance rate did not match the field data because the model contained too many assumptions and was too complex

This model is still valid for comparisons between two scenarios and to assist with decision making.

Botha et al. (2009) employed discrete-event simulation techniques to evaluate the underground development at Petra Diamonds' Cullinan Diamond Mine. They determined the effect of various developmental strategies and equipment capacities on the underground development rate, and identified potential bottlenecks in the mine development cycle.

2.2.3 Block and Panel Caving Mines

In block caving mines, simulation techniques are often applied to production aspects such as production scheduling, production rates, drawpoint availability, and fragmentation and the secondary breakings. Based on the same principles as those employed in other underground methods, some simulation models focus on ore handling systems and underground logistics in block/panel cave mines. The materials handling systems, however, are much more complex than those of other types of mines. Alternate footprint layouts can also be evaluated through simulation modeling. Limited works, however, have been completed on the development processes used in block/panel caving mines.

Chanda (1990) constructed a model that integrated simulation and optimization into an underground block caving mine. The model attempted to ascertain an optimal schedule for extracting ore from drawpoints that would balance the ore grade in successive periods. The simulation input was based on a production schedule determined by integer programming. The model was applied to a copper mine in Zambia. The results show a decrease in ore grade fluctuation and in the number of open drawpoints.

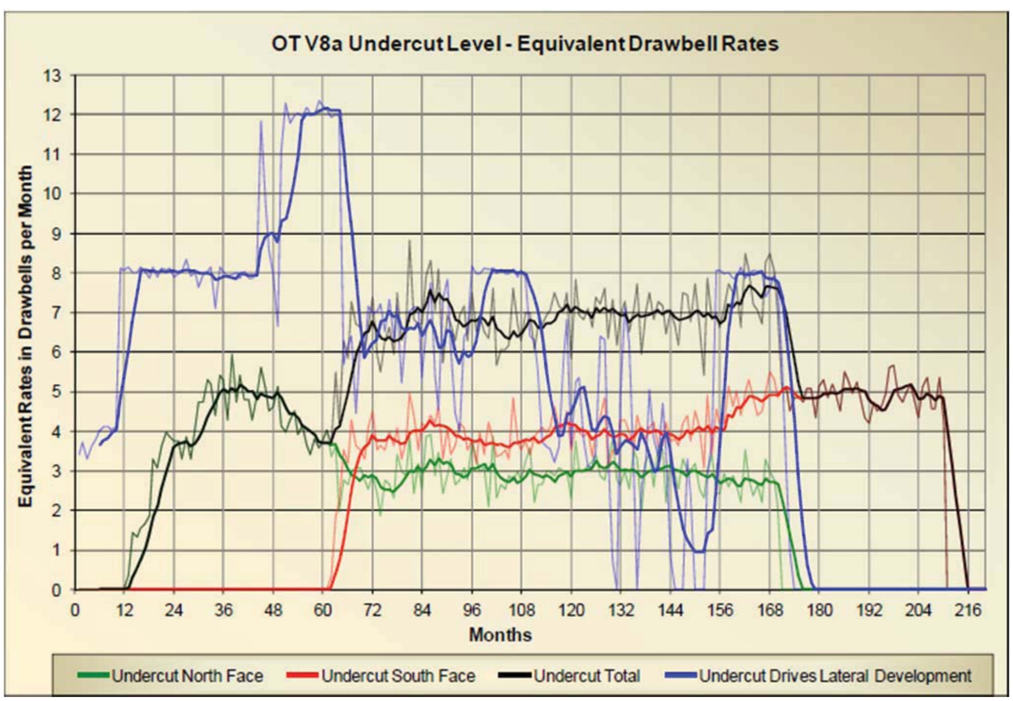
Dessureault et al. (2000) analyzed a production information management tool in block caving operations. They built an Extend® model to study the interaction between secondary breaking and Load Haul Dump (LHD) production performance. Simulation of the simplified load haul dumping process and the random blockage of drawpoints in five production drifts of block cave mines were experimented on to determine the optimum number possible for secondary breaking drill crews.

Hall, B. E. (2000) discussed the requirements needed for the successful simulation modeling of mining systems in block caving production. The case study modeled the mucking and secondary breaking activities in the DOZ block cave. The system included 30 production drifts, orepasses and dumps on the extraction level, two truck loops, loading chutes and dumping areas at the haulage level, and a simplified ore handling system consisting of a crusher and conveyors. The study found simulation to be a powerful tool for mining engineers and that it can provide useful information for project sponsors, but believed that its outcomes would be limited by the imagination of the modeling team and the simulation software. They felt that a large number of runs would be necessary to analyze the interactions of each parameter in a complex block cave system.

The Technical Services groups of Rio Tinto and Kennecott Utah Copper applied simulation to determine fragmentation, the frequency of secondary breakage, and production rates at the Bingham Canyon underground study. Simulation tools, the Fragmentation Model (BCF®), the Hang-up Model (HANG-UP®), and Discrete Simulation (ARENA®) were selected for use in their study. The BCF® provided a fragmentation profile of three main rock types at the production level and a cave column. BCF® results were then input into the HANG-UP® program to estimate the percentage of hang-ups over column height. Each type of blockage event along with other production cycles was modeled by discrete-event simulation models. This model was created by ARENA® software to study the production rate (Carter & Russell, 2000). Their simulation model determined the number of production resources, e.g. LHDs and secondary breaking rigs. The production rates were also evaluated for proposed footprint layout and equipment. In conclusion, they proved simulation to be a useful tool to test the sensitivity of input parameters in the production rate estimate.

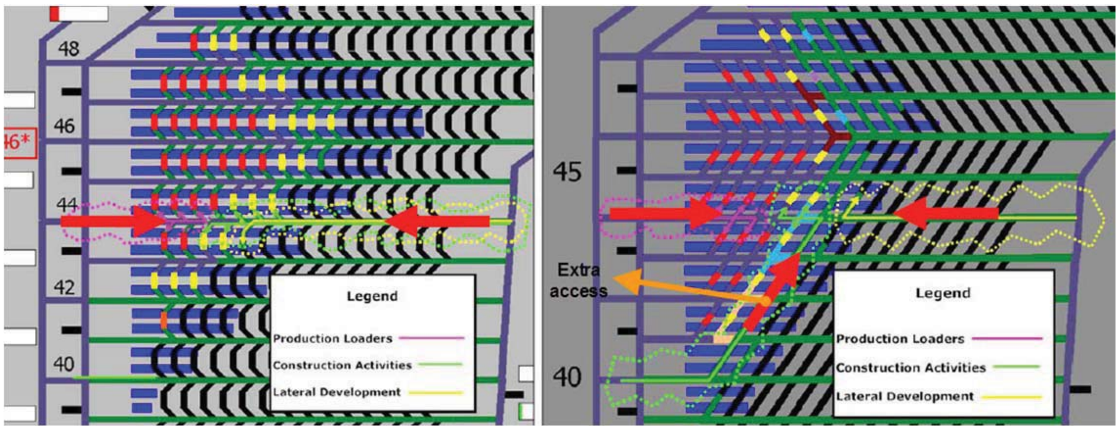
Labrecque et al. (2012) integrated simulation into mine planning for the Oyu Tolgoi block cave Feasibility Study. The group explained how simulation was used to drive mine design and production plan changes. The group outlined the outcomes from the Arena® simulation model of the drawbell construction rate over a 20-year period beginning with the first bell blasted. Typical results of drawbell blasting per month are illustrated in Figure 2-2.

Figure 2-2 Equivalent Drawbells per Month of Hugo North Lift 1 (Labrecque, Newman, & Dudley, 2012)



Key trade-off studies on the El Teniente vs. Herringbone drawpoint layouts were tested in this simulation model. The results indicate that the El Teniente layout can reduce the drawpoint construction time by giving one additional access to the construction areas (shown in Figure 2-3). As supported by simulation, the footprint changed to the El Teniente Layout in the Milestone 3 Feasibility Study.

Figure 2-3 Drawpoint Layout and Construction Access (Labrecque, Newman, & Dudley, 2012)



In the next trade-off, the group compared the "V" and diamond shaped undercut patterns. The model optimized the initiation location and showed the benefits of the "V" shaped undercut. In the last model, simulation assisted the group in determining the LHD and truck ramp-up, and equipment operating hours, in footprint areas up until the full production. Their study concluded that Arena® simulation can assist users in creating a better understanding and determination of the key drivers in the block caving's footprint design and planning. A number of mine planning trade-offs and system capacities, for which it was impossible to employ traditional mine planning techniques, have been tested by simulation. The decisions guided by simulation resulted in several changes in mine designs that improved productivity and reduced risk.

Hindle and Mwansa (2012) presented a solution for the Grasberg Block Cave (GBC) mine's underground logistic problems through integrated discrete-event simulation. Four different mines shared a rail system, and the transportation of both workers and materials all pass through its AB tunnels. A vertical shaft hoists workers and materials into three levels at GBC. Thus, the logistics of delivering both elements into the mine's levels are challenging.

The purpose for this simulation study was to identify the interactions between rail and shaft hoisting systems and to ensure that the overall system can achieve the movements of personnel within the desired time period during shift changes, and all material movements during peak development period.

The model was developed in Arena®. It identified the results of adding trains to the fleet at overall personnel transportation times (from when the first person boards the train to when the last person exits) as well as the traffic congestion time per day. The results demonstrated that 112 minutes would be required to transport all personnel with the six trains available. The target of completing shift change within 1 hour cannot be achieved by adding trains. Therefore, an alternate staggered shift strategy was required.

The group studied the impact on function of adding trains to the fleet. The results showed that by adding an extra train the personnel transportation time could be reduced, but the incremental benefit would be diminished as congestion and idle time between trains was increased. They also pointed out the key congestion areas in the rail systems.

Their paper concluded that the rail system and shaft were not constraints on worker and material movement in the mine development phase. The model identified that the optimized fleet size was three trains and that staggered shift changes for each mine was needed. This application demonstrates that simulation can increase the efficiency of logistics in block cave mines and that it can also ensure the timely arrival of both personnel and materials during the mine development stage.

3. Methodology

This chapter is divided into three sections. The first considers the data collection process and the methods used to prepare and analyze the data. When the temporal data was collected, only some of the heading types were developed and some of the planned equipment had not yet been delivered; thus assumptions of activity times were initially employed as inputs. This section discusses the method employed in making such assumptions. The next section concerns the software tools (packages) used to construct the model. Finally, section three introduces the general process utilized in the simulation study, and presents the process map derived from this work and used for the OT simulation case study.

Data such as the cycle time, shift schedule, absenteeism, preventive maintenance schedule, and equipment reliability were sourced directly from the OT mine site. The activity time data was collected by underground engineers, recording the start and stop time of each activity. The engineers also conducted interviews and sent out questionnaires to the operators to gain data on the range and nature of operational time delays. The equipment reliability and maintenance data were accessed from the shift by shift records of the current equipment's operational performance, long-term forecasts by the site's underground maintenance group and the records of the same equipment types in other mines. The mine designs and layout were based on the Oyu Tolgoi MS2 and MS3 Feasibility Study.

The input variables measured included: development cycle times, equipment fleets, shift schedules, development sequences, heading sizes and profiles, ground support designs, mine layout, crusher and mucking bay locations, and shaft capacities. After the input variables and processes were validated, then experiments were undertaken to test the simulation model; this included testing the development rate of different types of headings, accessing the sensitivities of different input parameters, and finding the bottlenecks in the mine development process.

3.1 Data Collection and Preparation

In order to determine inputs for the model, a six-month data collection program was initiated at the OT mine site where on-going lateral development with two jumbos had commenced. Data compilation included mining and equipment cycle times, shift schedules, equipment preventative maintenance timelines, and reliability data.

Several underground mining engineers were involved in the data collection and preparation. The objective of these time studies was to accurately measure the times in development activities and identify issues that would influence the development performance such as delays and wait times. With these measurements, a simulation model that will better represent the OT underground working environments can be developed.

Three sets of data were collected from the Oyu Tolgoi underground development operations. The first set of data was collected in November, 2010 and received on 25 January, 2011. The second set of data was collected in April and received on 27 April, 2011. The third set of data was collected in May, 2011 and received on 13 June, 2011. All data sets were collected manually by Mongolian mining engineers using stopwatches. A formatted data collection sheet was built by the OT site and employed through the third data collection, resulting on this occasion in very high quantity. The quality of time data improved progressively along the six month period as the variances in the data sets decreased and data descriptions became more detailed.

After all the data was received from the site, it was prepared and analyzed for input into the simulation model. The time study was compiled in detail in Appendix I Time and Motion Study. In order to ensure that the raw data was correctly interpreted and processed, sample activity cycle times on “I” type (5.0 m wide by 5.5 m high), “M” type (5.8 m wide by 5.8 m high), and “K” type (6 m wide by 7 m high) headings were presented to site engineers. These sample activity times were reviewed and calibrated with the real activity cycle, and recommended for use as simulation inputs.

3.1.1 Time Data

In order to mathematically analyze the time data and fit it into the simulation model a probability model first needs to be constructed (Ross, 2006). In order to simplify and calibrate the variance, a triangular distribution (Figure 3-1) was selected to represent the data. Since it is the most commonly used distribution in simulation modeling and project decision making if the most likely inputs are known ("Triangular distribution," 2012).

The triangular distribution is represented by the lower limit (a), the upper limit (b) and the mean (c). The lower and upper limits were obtained by the mean value plus or minus one standard deviation (σ), thus, a=b-σ, c=b+σ. One standard deviation only represents approximately 68% of the data collected. In the simulation, however, each time would occur thousands of times, so the overall outcomes would tend to lie in the mean value whether using one, two or three sigma to model the input time data. The lower limit of the bolting time, however, was only calculated by the mean value minus half standard deviation (0.5σ). Extreme outliers were removed from the data sets. Some assumptions were made when there were not enough data points to model their distribution. Therefore, the input data probability density function should resemble the graph below.

Table 3-1 outlines the results of the lateral development mine cycle data collection from the time study based on a 5.0m blasted round. The mine site operates in a multi heading environment with end-of-shift blasting practices.

Figure 3-1 Probability Density Function of Activity Time Distributions

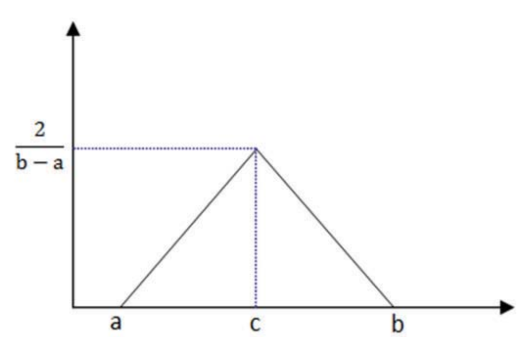


Table 3-1 Summary of Input Activity Time

Activity Time	Min	Avg	Max	Unit	# of Data Point
Drilling prep and teardown	16.2	25.5	34.7	min	10
Drilling	1.5	1.7	1.9	min/hole	14
Drill reaming holes	6.5	10.3	14.2	min	11
Clean drilled face	5.3	13.8	22.2	min	8
Charging prep and teardown	6.1	13.4	20.7	min	20
Charging and tie det.	1.1	1.5	1.9	min/hole	13
Blast and re-entry	60.9	86.9	113.0	min	5
Washdown face	12.5	23.5	34.5	min	19
Geotechnical inspection	13.3	20.8	28.3	min	9
Shotcreting prep and teardown	18.4	26.2	34	min	24
Hydroscaling	3.9	11.2	18.5	min	17
Shotcreting	3.9	4.8	5.6	min/m^3	29
Bolting prep and teardown	13.5	18.6	23.8	min	20
Bolt drilling	1.4	2.1	2.8	min/hole	21
Installing bolts	1.6	2.9	4.2	min/hole	18
Cable Bolting prep and teardown	13	14.1	15.2	min	3
Cable Bolting	29.8	38.1	46.	min/hole	11
Scaling and clean face	20.5	29.5	38.5	min	5
Survey time for jumbo grade line	9.6	15	20.4	min	8

The unit activity times such as face drilling (min/hole) and installing bolting (min/hole) were determined by the total time divided by the number of holes drilled/installed. For example, drilling of the first hole on face commenced at 9:30 pm and the last hole was finished at 11:40 pm. Total of 60 regular face holes were drilled and there was 10 minutes of reaming the initiation holes included in this drilling cycle. So the unit drilling time for regular face holes was (130-10) min/60 holes, i.e. 2 min/hole. This method was used to scale the drilling activity times on future planned headings which would be drilled by the same equipment, after the face drill patterns have been designed.

3.1.2 General Input Data

Oyu Tolgoi currently anticipates that poor ground may be associated with zones of significant faulting and high horizontal stress underground. Ground support regimes currently being used on site, as well as the life of mine support plans were incorporated into the model. This included a probability factor relating to the intersection of very poor ground and the need for additional cable bolts, tighter bolt patterns, and secondary layers of mesh and shotcrete. In the model, poor ground support regimes were randomly generated based on the poor ground proportions assumption (10% to 20%) to slow down the overall advance rate.

Mining areas were established in the model, limiting the number of development crews that could work in a given area. This was to reflect the realistic amount of equipment that can operate in a group of headings. Also such limits can reduce the congestion and travel time of mobile equipment when many crews are working underground. The collected data and site information was built into the model along with the following inputs:

- Specifications and number of individual pieces of equipment
- Equipment reliability and availability data
- Ground support regimes
- Mining areas
- Development crew ramp-up
- Mine layout
- Muck haulage handling methodology and tramming circuits
- Skip loading schedules and availability
- Shotcrete transportation and slick line availability
- Shift schedules, including number of working hours

Once inputs were appropriately established then the model was executed using the actual Oyu Tolgoi mine design. The model was then used to evaluate anticipated development bottlenecks, development rates in different heading and mining areas, and opportunities for improvement.

3.2 Modeling Tools

Prefeasibility study schedule development rates were determined using manual first principles spreadsheet analysis and benchmarking against data from best practice mining operations. In order to generate development rates more reflective of the Oyu Tolgoi mine layout, varying ground conditions, anticipated traffic congestion, and progressively longer muck and material tramming requirements, discrete-event simulation techniques were investigated. This simulation study was completed using several computer programs.

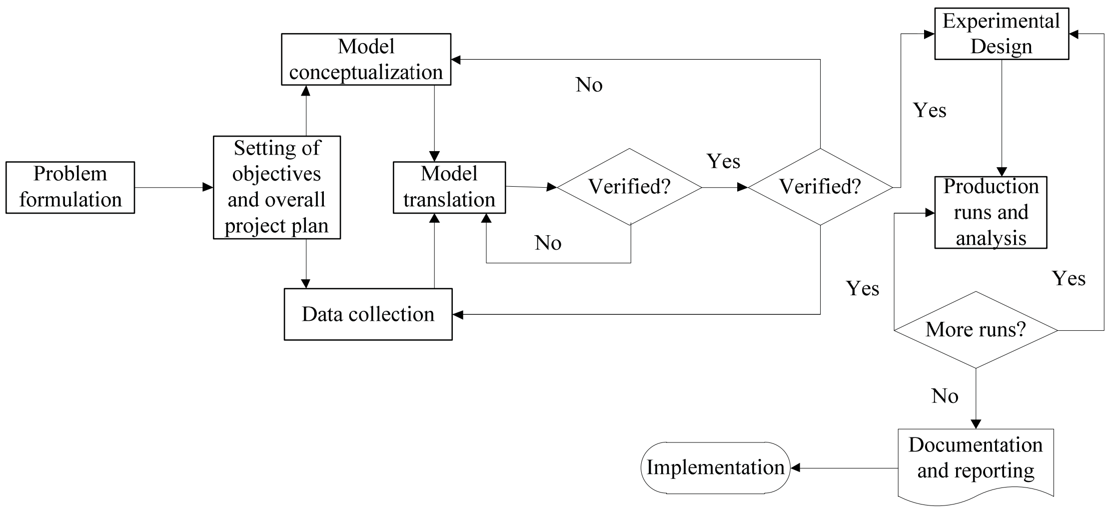
In the model buildup process, various computer tools such as, AutoCAD®, Vulcan® and Mine2-4D/EPS® were employed. A three-dimensional mine layout was created by Vulcan®. It was exported as DXF files and processed in AutoCAD®, a powerful and easy-to-use software utilized to edit the three-dimensional layout. SimMine® could then import these AutoCAD® files as input layout for the simulation model. Arena® was used in parallel to create other discrete event models for the footprint development construction activities and ore handling systems (Labrecque, Newman, & Dudley, 2012). This Arena® model will not be discussed in this thesis. Mine 2-4D/EPS® is mine planning and scheduling software that has been designed for evaluating cost and performance improvements in mine planning (GijimaAst, 2010). It was used to complete the final Feasibility schedule that can be viewed in 3D animation and can also be exported as a gantt chart.

Currently most of simulation software packages have been developed for general purpose modeling. A three-dimensional simulation tool, SimMine®, however, was specifically designed recently for underground mine development (SimMine®, 2010). This software was considered to be appropriate and selected to develop the model for this case study.

3.3 Simulation Process

A set of steps to guide a model builder in a thorough and sound simulation study is shown in Figure 3-2. Similar figures and their interpretation can be found in other sources, such as Pegden et al. (1995) and Law and Kelton (1991). This process map was built on that of Banks et al. (1996).

Figure 3-2 Process Map in a Simulation Study (Banks, Carson II, & Nelson, 1996)



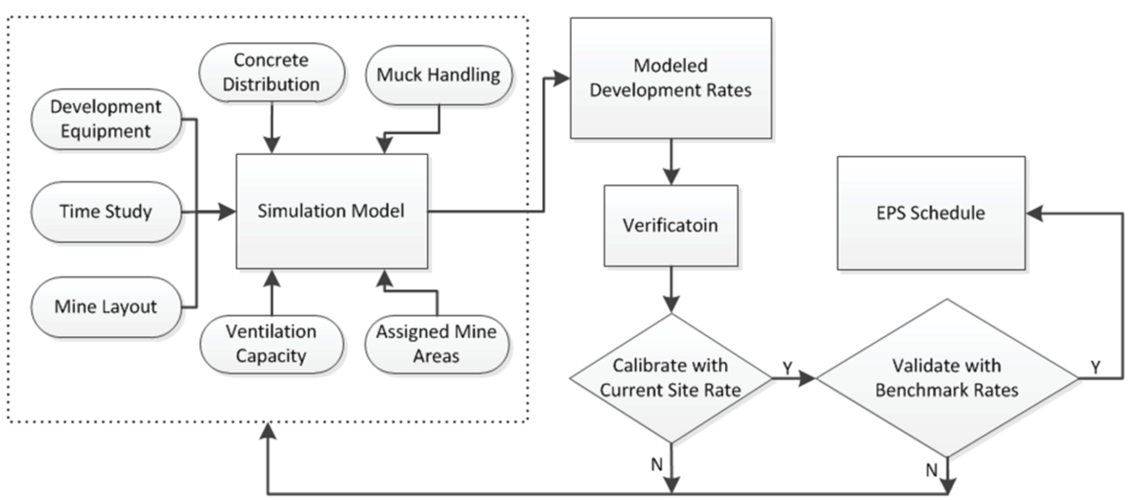
The general steps in simulation modeling were described by Banks et al. (1996) as follows.

1. Problem formulation: State the problems in the study. Programmer, analyst and decision makers need to understand early the problems.
2. Setting of objectives and overall project plan: to indicate the questions that will be answered by the simulation model and to discuss whether the simulation is the appropriate technique for that particular study's purpose. The overall project schedule needs to be established during this phase.
3. Model conceptualization: to abstract the features of the problems and how they might be included in the model, as well as to fully understand how the system that is to be simulated works in the real world and select useful assumptions and input parameters for it.
4. Data collection: The input data is required when developing the model. As the model evolves, this input data may also become complex. This process should start in the early stage of model construction.
5. Model translation: Transfer the model into a computer program and decide which program to be used in order to achieve the objectives.

6. Verification: Check the model's logic and programing errors and make sure it is free of any kinds of errors.
7. Validation: Determine that the model correctly represents the real system or problem. Calibrate the model by comparing the model to the actual system and using the differences between the two to improve the simulation model until it accurately reflects the system's behavior.
8. Experimental Design: When the model is validated and ready to use then the alternatives that need to be simulated must be determined.
9. Production runs and analyses: Measure and analyse the performance of each experimented system's designs that have been simulated.
10. More runs: Based on the results of the completed runs, the analyst needs to decide whether to perform more experimental runs and, if required, how to design these runs.
11. Documentation and reporting: The progress report, outcomes, inputs and model databases or programs need to be documented.

The modeling process of the OT case study was based on the steps discussed above but it was modified to suit the specific goals of this study. Figure 3-3 shows an example of this OT development simulation model construction process.

Figure 3-3 OT Development Simulation Modeling Process Map (Wolgram, Li, & Scoble, 2012)



First, the input data needed to be gathered for the simulation model. There were several examples, although not limited to the model input data, i.e. a) The activity times, shift schedule, and equipment preventive maintenance and breakdowns were collected from the time study; b) the equipment build-up and crew ramp-up were based on the MS2 assumptions and ventilation model respectively; c) the development fleet, mine layout , and muck handling and concrete distribution systems were input from the MS2 and MS3 mine designs; and d) the assigned mine areas for development crews were based on empirical assumptions and ventilation constraints.

Next, all the input data were fed into the simulation model. The model simulated the underground development advance and excavation system at the OT underground site. The simulation model then generated important outcomes, such as future development rates.

However, the outcomes needed to be verified by checking the input, system logics and program and ensuring that they are correct. Once this step was completed then the results were input into the validation process. The validation process consisted of two steps: the first step was to calibrate the simulated development outcomes with OT's actual development records. Basically, the model realistically reproduced the development process of the11 month period observed in the OT underground mine. The second step was to compare the outcome with the benchmarked data from other similar underground mines.

Lastly, after validating the outcomes using these two steps, it was then used as inputs to EPS/Mine2-4d software which produced the Feasibility schedule.

4. Simulation Model Setup

This chapter discusses the model construction process and the details of its input parameters and assumptions. Most of the input parameters and their values of the simulation model are outlined here. Activity time inputs for this simulation were collected during a 6-month time study of the OT site's underground development activities. The time study data is documented in Appendix I: Time and Motion Study.

The chapter first begins by presenting each type of the single pass heading profile and the massive excavations modeled in this study. The footprint area and cave zone are introduced later as well as the distribution of the ground conditions through the OT underground mine. Ground support requirements which are the most important part of the development simulation modeling are discussed in depth.

Next, the chapter outlines the development activity cycles in the model. The differences in these cycles between the on and off footprint areas are compared. The activity times of the off footprint areas are mostly summarized in the time study. However, the assumptions for the on footprint areas are listed in this chapter.

The development equipment fleet used in the simulation is introduced. After this, the fleet's characteristic parameters, such as build-ups, tramming speed, maintenance and breakdowns are presented in detail. The development crew's ramp-up schedule and allowed work areas are illustrated. Finally, the development muck flow logistics are described. The muck flows are verified in terms of local systems and the entire mine.

4.1 Heading Profiles and Ground Support

4.1.1 Single Pass Heading Profiles

Most of the lateral development in the simulation model was excavated by single pass drill-and-blast drifting. The dimensions of the drift profiles in this thesis were recorded as a m_W(width) x m_H(height) format; for example, the regular

“5.0mWx5.5mH” profile indicates that he drift dimensions are 5 meters wide by 5.5 meters high. Different types of profiles had diverse input parameters including activity times, rock properties, development activity cycles, and ground support requirements.

In the MS2 model, 13 different heading profiles were designed and employed for PPD development simulation. The dimensions and descriptions of these profiles can be found in Table 4-1. Some headings around the Shaft 1 area were slightly different from the regular sized ones. For example, the OT site changed the regular 5.0mWx5.5mH dimensions in some drifts to those of 5.5mWx5.5mH. These types of headings were grouped together with the regular “I” type heading in this simulation. The “X” type were the on footprint perimeter drifts which had the same dimensions as the 5mWx5.5mH “I” type profile.

Table 4-1 Heading Profiles and Dimensions

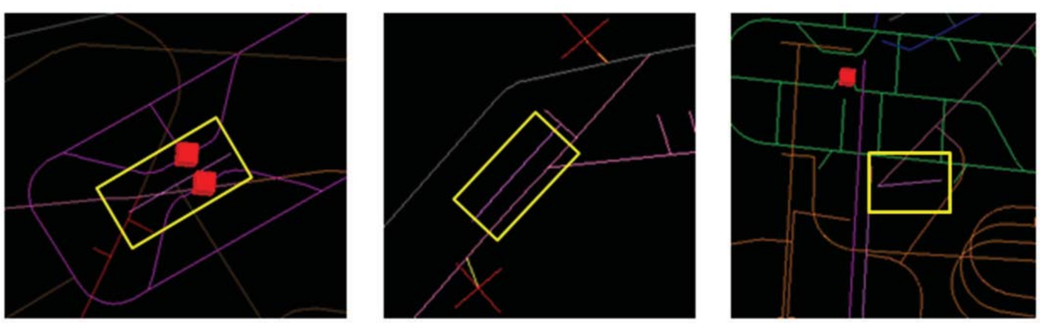
Face Profile	H (m)	W (m)	Arc. Rad	Area (m^2)	Peri. (m)	Description
D	4	4	2	14.3	10.3	Extraction drive
E	4.5	4.5	2.25	18.0	11.6	Undercut drive
F	4.5	4.2	1	16.7	11.0	Drawpoint
H	5	5	2.5	22.3	12.8	Crosscut
I	5.5	5	2.5	24.8	13.8	Ramp access
J	6	6	1	35.6	17.1	Main shop
K	7	6	3	38.1	17.4	Exhaust drift
M	5.8	5.8	2.25	31.6	15.2	Haulage drift
O	7.7	12	2.5	83.2	23.8	LOD1
P	10.5	12	2.5	113.4	28.9	LOD2
Q	5.8	6.3	2.5	32.7	15.7	TH680
R	9	7.6	2.5	61.6	22.6	Truck Chute
S	5.5	6.5	2.5	32.2	14.0	Conveyor drift
T	5.4	4.5	2.5	21.9	15.3	3 cut bay top
U	3.7	7	2.5	23.3	14.4	2 cut bay bottom
V	5	7	2.5	31.5	17.0	2 cut bay top
X	5.5	5	2.5	24.8	13.8	Rim

In the MS3 model, however, the heading profile types were made even more complex. 24 types of single pass heading profiles were used in this model. They were grouped into 11 categories based on their dimensions, ground support requirements and main functions. All the input parameters for the profiles in this model were derived from the MS2 model, and this model was not utilized for key simulated experimentation. Therefore, the details of the heading profiles and groups of the MS3 model are not discussed in this thesis.

4.1.2 Mass Excavations

During the development of the OT case study some of the large excavations such as workshop bays and conveyor transfer chambers were cut by two or multiple passes, for example, benching the top pass first and then with a bottom pass. Modeling such excavation was very challenging. Three mass excavations were created in the MS2 model, including the Crusher 1 chamber, the Transfer chamber, and Bin 2 tail end. One single centerline with a very large profile was employed to represent these mass excavations. The volumes of these created drifts were adjusted to equal those of the mass excavations. Figure 4-1 shows the three mass excavations represented by single lines in the MS2 model.

Figure 4-1 Mass Excavation in Simulation Model

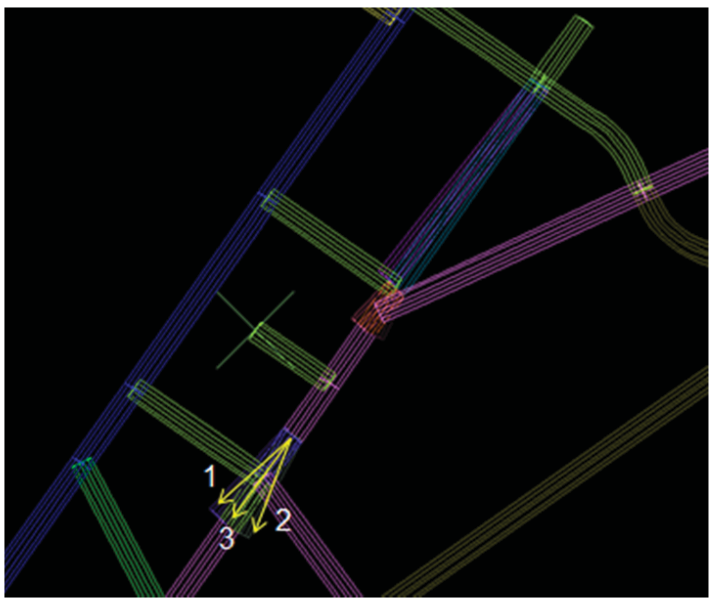


Crusher 1 Transfer Chamber Bin 2 Tail End

In the MS2 model an experimentation test was completed on six bays of the Shaft 1 workshop (S1W). The bays are 7.0mHx8.7mHm, and needed to be excavated by two passes. The details of the input assumption and test setups can be found in Appendix II: Simulation Test 9.

All mass excavations were modeled by the two or three passes in the MS3 Model. Two or three lines were created to represent the sequence of passes in which the excavation would be developed (as shown in Figure 4-2). Each of the passes had different activity times and profiles. Less ground support was assumed on the temporary walls and backs which would be blasted by the later pass.

Figure 4-2 Modeling of Mass Excavation



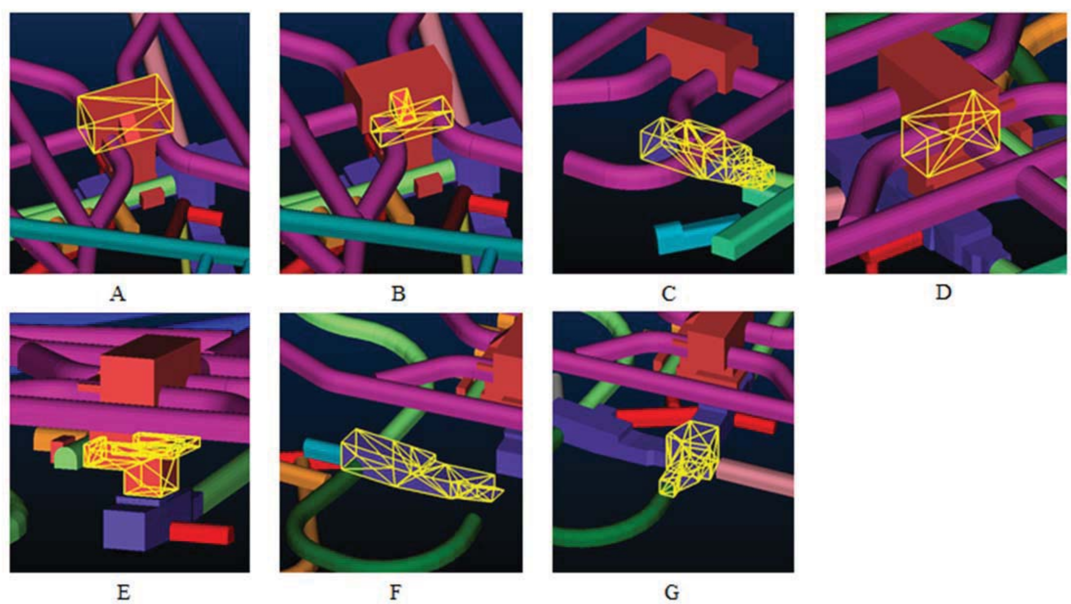
The Crusher 1 complex was included in the MS3 model. One crew was assigned to work progressively on the Crusher 1 to complete its 7 sections. The sections and excavation sequence are shown in Figure 4-3. Each section followed a regular drill-basting cycle with cable bolting. The development duration of each section was adjusted to mimic the EPS schedule. The working days used in the simulation model were compared with those of the EPS schedule in Table 4-2.

The purpose was to keep one development crew working on the crusher complex during this fixed period and incorporating the congestion and other interactions into the simulated system.

Table 4-2 Schedule of the Crusher 1 Complex

	Section	EPS (days)	Model Input (days)
Top cut 1	A	74.5	71
Top cut 2	B	17	19
Bottom 1	C	79	79
Middle 1	D	66.5	66
Middle 2	E	57.5	56
Bottom 2	F	99.6	100
Bottom 3	G	77.3	76

Figure 4-3 Crusher Excavation Sequence



4.1.3 Good and Poor Ground Distribution

The good and poor ground conditions tend to be randomly distributed around the OT underground mine. The ground conditions inside the footprint area are worse than those found outside. The caving zone surrounds the footprint area at 100 meters from its boundaries. The ground conditions in this zone were assumed to be eventually affected by the cave.

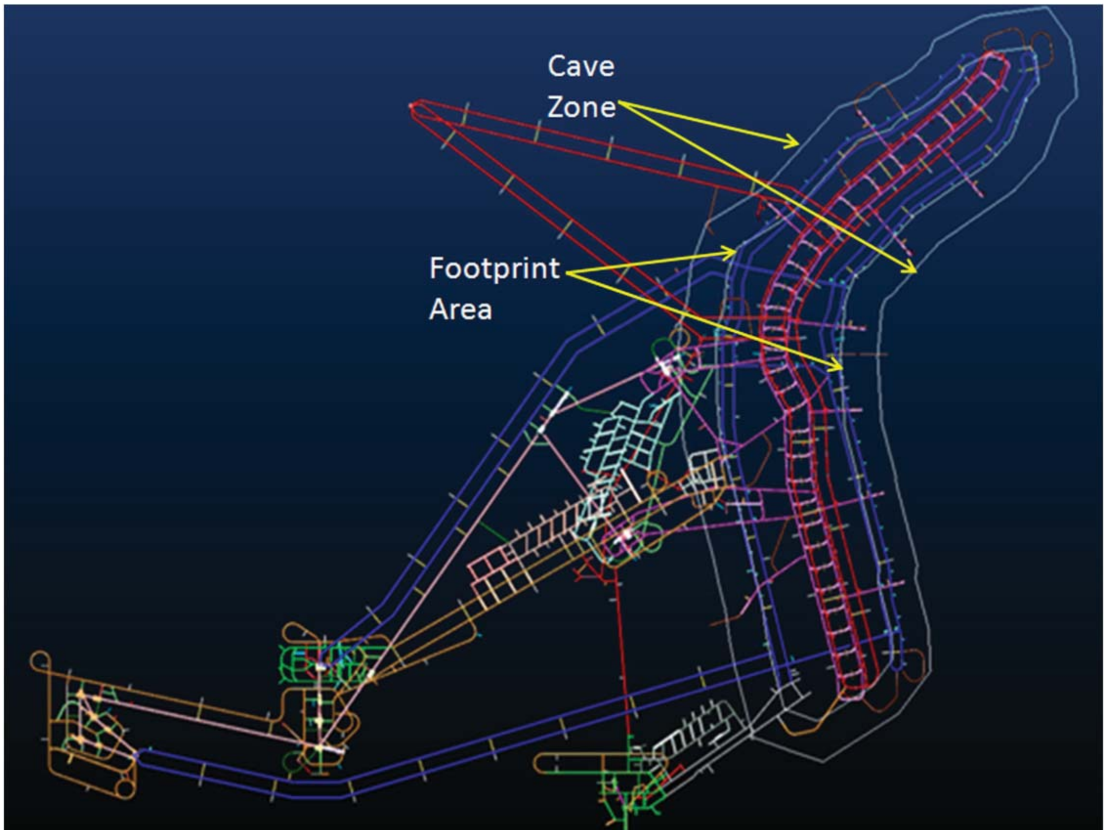
It was very difficult to accurately model the ground conditions in the study stage. However, assumptions were made based on the existing geotechnical information. Table 4-3 shows the distributions of good and poor ground with in- and outside the footprint areas. The Type 1 ground support category applies to good ground conditions, whereas Type 2 applies to poor ground conditions, which require more extensive ground support.

Table 4-3 The Distribution of Ground Support Categories

	Prefeasibility		Actual Data		Simulation Model	
Description	Good	Poor	Good	Poor	Good	Poor
Inside Footprint	70%	30%	n/a	n/a	80%	20%
Outside Footprint	80%	20%	90%	10%	90%	10%

The pre-feasibility study's estimates for the good and poor ground distributions were 80% and 20% outside the footprint area (McIntosh Engineering, 2009). According to the data observed from the OT site, however, there was approximately 10% poor ground on the existing development outside the footprint area. Thus, this simulation model used the assumption of 10% Type 1 and 90% Type 2 for the off-footprint areas. This simulation only covered a small portion of the footprint area development such as one perimeter drift and a few undercut and extraction drifts. The ground distributions on footprint were assumed to be 80% for Type 1 and 20%for Type 2, which was the same as that of the Feasibility Study.

Figure 4-4 Footprint and Cave Zone Boundaries of Hugo North Lift 1 (Wolgram, 2011)



4.1.4 Ground Support

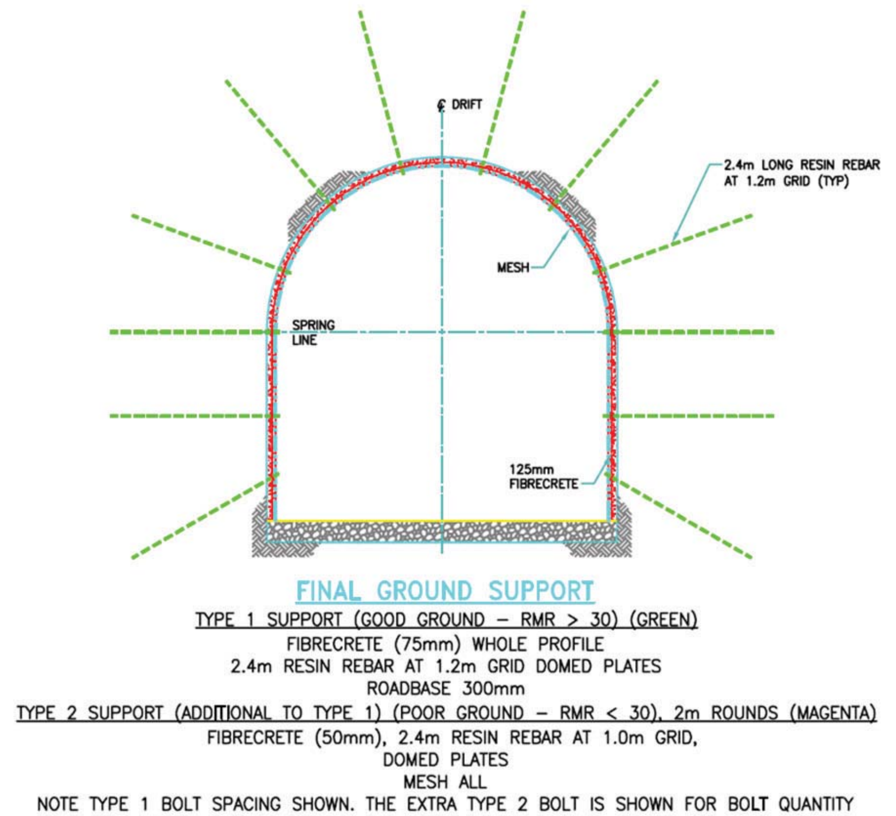
Ground support regimes in this simulation model were based on the MS2 and MS3 mine design. They can be divided into four major categories: off footprint Type 1 and 2 and on footprint Type 1 and 2. Ground support accounted for the longest time period in the simulated development cycle. It was extremely long when the ground conditions were poor. Thus, it is critical to study these activities in detail.

1) The Off Footprint Ground Support Regime

In the off footprint areas, the Type 1 ground support protocol followed the regular shotcrete, bolting and cable bolting (if needed) approach. For regions with poor ground conditions, the same approach still applied but the second layer of shotcrete was added and meshed between the two layers. The rock bolt spacing was reduced and cable bolts needed to be installed when required in the Type 2 ground support.

The ground support was completed in-cycle for off footprint developments. In-cycle ground support means that the drift was only allowed to advance if the whole support cycle were complete. An example of a ground support design of the regular "I" type ramp drift is shown in Figure 4-5. All development on the cave zone was assumed to require 100% Type 2 ground support.

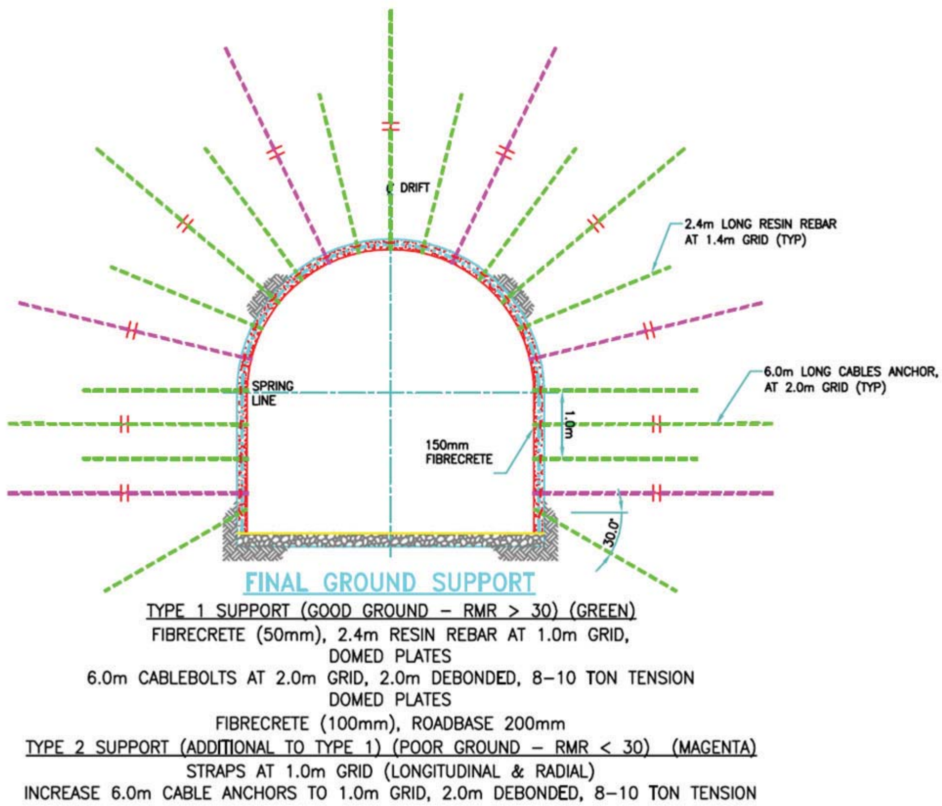
Figure 4-5 Ground Support Design of "I" Type Profile (AMEC Engineering, 2011)



2) The On Footprint Ground Support Regime

In the footprint area, the rock bolt spacing was reduced compared with that of the footprint area. Cable bolts and second layers of shotcreting were required for all types of headings except for undercut drive Type 1 support. These two activities were assumed to be campaigned every 10 rounds. Campaigned support means that the drift advances for several rounds with partial ground support, and is then halted to complete the remainder of the ground support for the already excavated drift. For example, in extraction drifts, the drift advances every 50 meters on shotcrete, mesh, straps (where needed) and resin bolts, and then the second layer of shotcrete cable bolts is applied to this 50 m drift. An example of a ground support design of the "D" type extraction drive is shown in Figure 4-6.

Figure 4-6 Ground Support Design of "D" Type Profile (AMEC Engineering, 2011)



The shotcrete amount used in footprint area headings was estimated based on the site data. The 5mWx5.5mH drift Type 1 (75 mm of shotcrete) was used as the benchmark, applying 10 m^3 of shotcrete for the 4.8 m round. Thus, it was an approximately 2 m^3 per meter advance for an application of 75 mm thickness. The shotcrete amount assumed for on footprint drifts can be found in the table below.

Table 4-4 Footprint Shotcrete Amount in Different Headings

Type of Heading	Profile	First Layer 50mm Shotcrete amount (m^3/m advance)	Second Layer 100mm Shotcrete amount (m^3/m advance)
Undercut drive	4.1mWx4.1mH	1.5	2
Extraction drive	4.5mWx4.5mH	1.5	2
Perimeter drift	5.0mWx5.5mH	1.5	2.5

As shown in Figure 4-6, the cable bolt requirements for extraction drives are 8 cables at 2 m ring spacing (along the drift axis) for good ground and increasing to 1m ring spacing for poor ground. Cable bolting was only needed for Type 2 on the undercut drives with a 2m grid pattern. The perimeter drifts (5mWx5.5mH) were assumed to require a 100% Type 2 pattern at 1 m ring spacing. Since at intersections the total number of cable bolts should not be more than the cable bolts in the drifts, 2 m x 2 m for good ground and 1 m x 1 m for poor ground cable bolt patterns were assumed. In summary, the total numbers of cable bolts needed per round and per meter advance are listed in Table 4-5.

Table 4-5 Footprint Heading Cable Bolts Requirement

Heading	Profile	# of cables/round		# of cables/m		
		Type 1	Type 2	Type 1	Type 2	Weighted Avg
Turnout	4.5mWx4.5mH	6	21	1.2	4.2	1.8
Extraction drive	4.5mWx4.5mH	15	50	3	10	4.4
Extraction and Turnout	4.5mWx4.5mH	12	40	2.4	8	3.5
Undercut drive	4.1mWx4.1mH	0	10	0	2	0.4
Extraction perimeter	5mWx5.5mH	65	65	13	13	13
Undercut perimeter	5mWx5.5mH	65	65	13	13	13

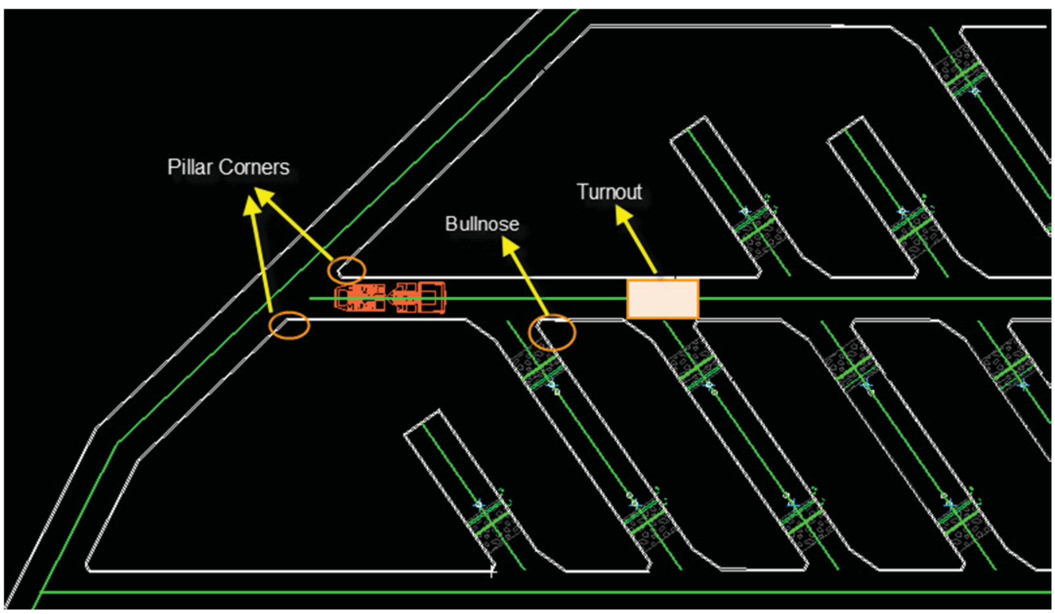
Detailed descriptions of footprint ground support regimes are listed as follows:

i. Undercut drives

Good Ground (RMR>30) Type 1	Poor Ground (RMR<30) Type 2
50mm shotcrete whole profile	50mm shotcrete first layer
2.4m resin rebar at 1m grid (9@1m)	Mesh with 0.5m splitset at 2m grid
	2.4m resin rebar with straps at 1m grid (10@1m)
	6m cable bolts at 2m grid (5@2m)
	75mm second layer shotcrete

ii. Extraction drives (turnouts not included)

Figure 4-7 Typical Extraction Level Layout



Good Ground (RMR>30) Type 1	Poor Ground (RMR<30) Type 2
50mm first layer shotcrete	50mm first layer shotcrete
Mesh with 0.5m splitset at 2m grid	Mesh with 0.5m splitset at 2m grid
2.4m resin rebar at 1m grid (12@1m)	2.4m resin rebar with straps at 1m grid (12@1m)
6m cable bolts at 2m grid (6@2m)	6m cable bolts at 1m grid (10@1m)
100mm second layer shotcrete	100mm second layer shotcrete

iii. Turnouts

Turnouts account for 35% of the total plan area of extraction drives, where only the back and bullnose need to be cable bolted (the walls are excavated for drawpoints). A typical layout is shown in Figure 4-7. In plan view, the area of the turnout is 25 m^2 and the area of the turnouts and extraction drives are 15 * 4.5 = 67.5 m^2, so the proportion of the turnouts on the extraction drives is about 35%.

Good Ground (RMR>30) Type 1	Poor Ground (RMR<30) Type 2
50mm first layer shotcrete	50mm first layer shotcrete
Mesh with 0.5m splitset at 2m grid	Mesh with 0.5m splitset at 2m grid
2.4m resin rebar with straps at 1m grid on back (4@1m)	2.4m resin rebar with straps at 1m grid on back (4@1m)
2.4m resin rebar with straps at 1m grid on bullnose (4@1m for 6m)	2.4m resin rebar with straps at 1m grid on bullnose (4@1m for 6m)
6m cable bolts at 1m grid (4@1m)	6m cable bolts at 1m grid (4@1m)
3 cable on 1m vert spacing on bull nose (3 cables)	3 cable on 1m vert spacing on bull nose (3 cables)
100mm second layer shotcrete	100mm second layer shotcrete

iv. Undercut and extraction perimeter drives (Type 1 and 2 are the same)

On the rims there is 1 pillar corner in every 30 m spacing (Figure 4-7). This only needs be mesh strapped. Therefore, this will not change the number of cable bolts required for the campaigned rounds on the perimeter drift.

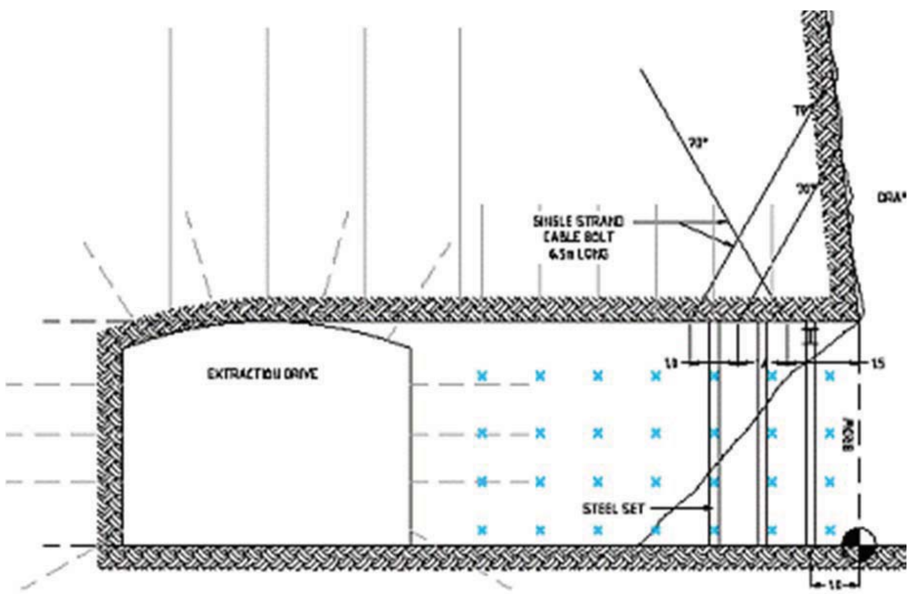
Good Ground (RMR>30) Type 1	Poor Ground (RMR<30) Type 2
50mm first layer shotcrete	50mm first layer shotcrete
Mesh with 0.5m splitset at 2m grid	Mesh with 0.5m splitset at 2m grid
2.4m resin rebar with straps at 1m grid (14 bolts@1m)	2.4m resin rebar at 1m grid (14botls@1m)
6m cable bolts at 1m grid (13 cables@1m)	6m cable bolts at 1m grid (14 cables@1m)
100mm second layer shotcrete	100mm second layer shotcrete
50mm first layer shotcrete	50mm first layer shotcrete

v. Drawpoints

Typical drawpoint support is illustrated in Figure 4-8.

Good Ground (RMR>30) Type 1	Poor Ground (RMR<30) Type 2
50mm first layer shotcrete	50mm first layer shotcrete
Mesh with 0.5m splitset at 2m grid	Mesh with 0.5m splitset at 2m grid (chain mesh with higher deformation capability)
2.4m resin rebar with straps at 1m grid (12 bolts@1m)	2.4m resin rebar with straps at 1m grid (12 bolts@1m)
Cable bolts for 2 rows of 5 cables in brow area (10 cables)	6m cable bolts at 1m grid (9 cables@1m for 3m)
Steelset (2 sets@1m)	Steelset (4 sets@1m, 1m grid cables inside)
100mm second layer shotcrete	100mm second layer shotcrete

Figure 4-8 Example of Drawpoint Support Pattern



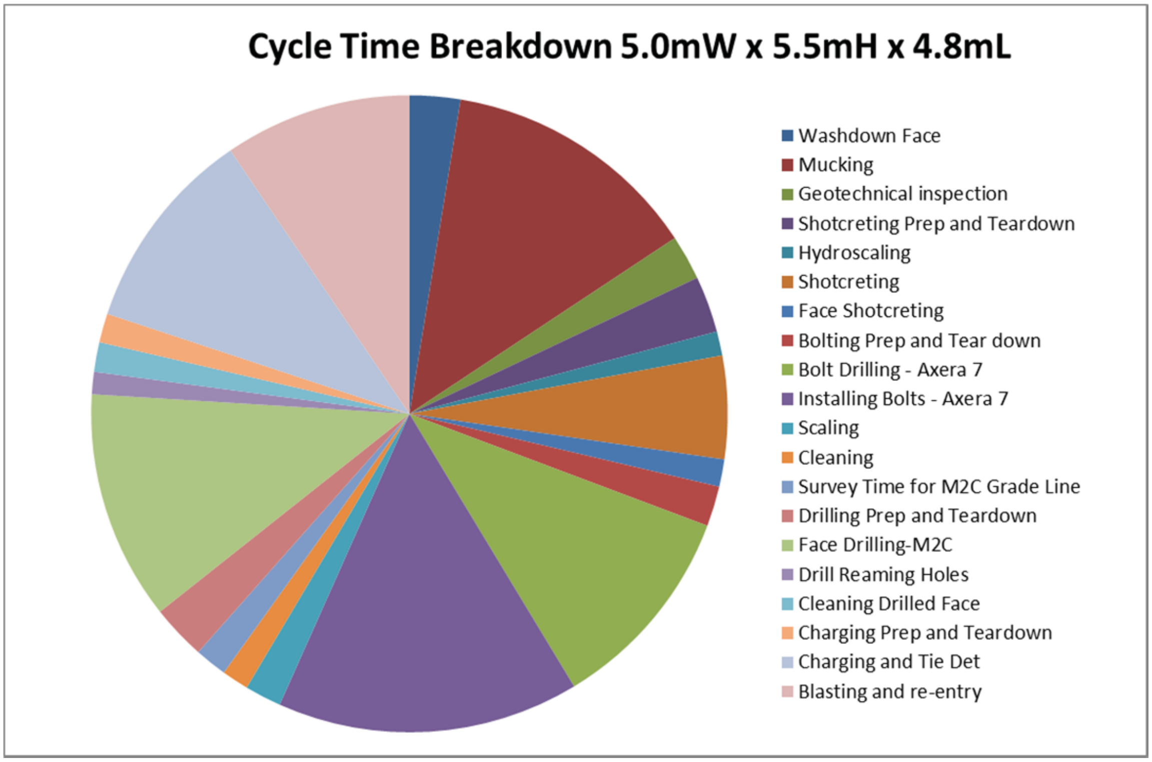
(*drawing from unknown caving mine)

4.2 Development Activity Cycles

4.2.1 Regular Activity Cycle

The development round in the model started from the washdown face and ended in the blasting and re-entry portion. The regular activity cycle was applied to all off-footprint area developments with fixed activities which occurred every cycle and random activities which presented through probability on different profiles. An example of the regular activity cycle of “I” type profile is shown in the pie chart below.

Figure 4-9 Example Activity Cycle of "I" Type Profile



1) Fixed Development Activities

The fixed development activity cycle in this model was described as including drilling, charging, blasting, loading and transport, shotcreting, and bolting. These development activities presented themselves in every round and in all ground conditions. Some headings with larger profiles, for example, the “J” type (workshop, 6mWx6mH) and “L” type (conveyor drift, 6.5mWx5.5mH) require in-cycle cable bolting as well.

During the development of the OT underground mine, all intersections will need to be cable bolted. Unfortunately, this process could not be modeled by the program employed in this study, since the intersection and regular cable bolting activity could not be separated in the program. There was only one input parameter for those two activities. However, an approach was used to approximate the intersection cable bolting activity in the model. The approach was to create another regular cable bolting activity which could occur randomly on the development drifts to represent the cable bolting activity at intersections.

In the MS2 model, an additional 10% probability was assumed for this second cable bolting activity which represented intersection cable bolting on regular drifts. This percentage was calculated by the total length of the intersections over the total development distance in the model. For example, when one 200 m drift intersected with two 5 m wide remucks and one 6 m wide exhaust drift, the intersections then accounted for (5 x 2 + 6)/200 * 1.5 = 12%. Based on this method, the overall intersection percentage over the total development until the end of 2013 was approximately 10%.

However, in the MS3 model the intersection cable bolting was changed to various percentage value from 0 to 20% based on the number of intersections in that mine area. For example, in the undercut drift there were fewer intersections and all the pillar corner cable bolting had already been included in the regular cable bolting activity. Thus, the additional intersection cable bolting was not assumed (at 0%) for the extraction and undercut drifts in the footprint area. However, in the Shaft 2 station areas, there were more intersections than regular drifts, so this percentage was increased to 20%.

2) Random Development Activities

Some activities such as cleaning the fly rocks after blasting, face shotcreting, and face supports (scaling, cleaning and meshing of faces) occurred randomly during the development process. The probabilities of random activities were calculated by the number of occurrences over the total number of data points recorded. An example of this is the ratio of six data points of revealed cleaning drilled face activity out of 11 data points recorded during the face drilling. The probability of cleaning drilled face would then be 6/11 = 55%. The random activities identified from the site data collections included:

- An additional 50 mm of shotcrete and screening for poor ground condition in off-footprint area
- Cleaning drilled face after face drilling using a loader
- Cleaning fly rocks (over-sizes) using an LHD
- Face shotcreting during shotcreting walls and backs using a sprayer
- Face support during bolting
- Scaling using a bolter
- Installing mesh on a face using a bolter
- Cleaning after scaling a face using an LHD

Using the method mentioned above, all the probabilities of these random activities are summarized in Table 4-6.

Table 4-6 Random Development Activities and Probabilities

Random Activities	# of Occurrences	Total # Recorded	Prob.	Remarks
Extra Shotcreting (1)	n/a	n/a	10%	Up to date percentage of poor ground
Cleaning Drilled Face	6	11	55%	Third data
Cleaning Fly Rock	1	2	33% *	First data, data points limited
Face Shotcreting	1	n/a	33% *	Probability unknown
Face Scaling	3	15	20%	Third data
Face Meshing	4	15	25%	Third data
Cleaning Supported Face	6	15	40%	Third data
(1) is only for poor ground conditions			* is assumption	

4.2.2 Footprint Activities Cycle

The development activity cycles in footprint areas were different from regular activity cycles. Type 1 and 2 ground support accounted for 80% and 20% respectively in footprint areas. Table 4-7 and Table 4-8 list the development activity cycles in the order in which they were performed on the undercut and extraction drives.

Cable bolting and second layer shotcrete in the footprint area were campaigned every 10 rounds (5m blast round). The campaigned ground support means that the heading advanced several rounds on drilling, charging, blasting, mucking, shotcreting, bolting, meshing and strapping (where needed), and then stopped for cable bolting and the application of the second layer on the already excavated and partially supported drifts.

Table 4-7 Undercut Drives Development Sequence

#	Activities	Probability	Description
1	Drilling	100%	
2	Face cleaning	50%	
3	Charging	100%	
4	Blasting	100%	
5	Washing	100%	
6	Loading and transport	100%	
7	Geotechnical inspection	50%	
8	First layer shotcreting	100%	50mm, modified to 75mm.
9	Meshing& install split set	100%	0.5m split
10	Bolting	100%	
11	Strapping	20%	0.3 width strap at 1x1 grid. Poor ground only
12	Cable bolting	Every 10 rounds	Activity time is 10 times 1 round.
13	Second layer shotcreting	Every 10 rounds	100mm, modified to 75mm. Activity time is 10 times 1 round
14	Face scaling	70%	
15	Cleaning	80%	
16	Survey	100%	

Table 4-8 Extraction Drives Development Sequence

#	Activities	Probability	Description
1	Drilling	100%	
2	Face cleaning	50%	
3	Charging	100%	
4	Blasting	100%	
5	Washing	100%	
6	Loading and transport	100%	
7	Geotechnical inspection	50%	
8	First layer shotcreting	100%	50mm, modified to 75mm.
9	Meshing& install split set	100%	0.5m split
10	Bolting	100%	
11	Strapping	20%	0.3 width strap at 1x1 grid. Poor ground
12	Cable bolting	Every 10 rounds	Activity time is 10 times 1 round.
13	Second layer shotcreting	Every 10 rounds	100mm, modified to 75mm. Activity time is 10 times 1 round
14	Face scaling	70%	
15	Cleaning	80%	
16	Survey	100%	

4.3 Activity Times

4.3.1 Off Footprint Activity Time

The activity times of this discrete-event simulation was based on site time study and assumptions under the feasibility mine design. The OT site conducted a 6 month time study to collect data for the simulation model. The details of the activity time can be found in Appendix I: Time and Motion Study. Most of the data was collected on the “I”, “K”, and “M” type headings. After the raw time data was processed, the activity time data was then fitted by triangular distribution. Example activity times of these headings (shown in Table 4-9 to Table 4-11) were sent back to the site for review to minimize misunderstandings and errors for simulation inputs. These activity time inputs were then verified by the OT site. The activities shown in blue were non-vehicle activities.

Table 4-9 Sample Activity Time of 5mWx5.5mH Heading

5.0x5.5 Face - Support with Axera						
Activities	**Min(- 1Std)**	**Mean**	**Max(+1Std)**	**Std**	**Probability**	**Remark**
Washdown Face	12.5	23.5	34.5	11.0	100%	
Mucking	90.0	120.0	150.0	30.0	100%	Example time - SimMine will calculate
Geotechnical inspection	13.3	20.8	28.3	7.5	50%	
Shotcreting Prep and Teardown	18.4	26.2	34.0	7.8	100%	
Hydroscaling	3.9	11.2	18.5	7.3	50%	
Shotcreting	38.6	47.8	57.0	9.2	100%	4.8minx10m^3
Face Shotcreting	12.3	12.7	13.0	0.3	100%	
Bolting Prep and Tear down	13.5	18.6	23.8	5.2	100%	
Bolt Drilling - Axera 7	80.7	97.1	130.0	32.8	100%	4 rows of 12 bolts
Installing Bolts - Axera 7	110.1	140.3	200.6	60.4	100%	4 rows of 12 bolts
Scaling	12.0	17.0	22.0	5.0	70%	
Cleaning	8.5	12.5	16.5	4.0	80%	
Survey Time for M2C Grade Line	9.6	15.0	20.4	5.4	100%	
Drilling Prep and Teardown	16.2	25.5	34.7	9.2	100%	
Face Drilling-M2C	91.0	106.5	122.0	15.5	100%	1.7 min x 63 holes
Drill Reaming Holes	6.5	10.3	14.2	3.8	100%	2 reaming holes
Cleaning Drilled Face	5.3	13.8	22.2	8.5	50%	
Charging Prep and Teardown	6.1	13.4	20.7	7.3	100%	
Charging and Tie Det	71.5	95.1	118.8	23.6	100%	1.5 min x 63 holes
Blasting and re-entry	60.9	86.9	113.0	26.1	100%	
Total (min)	**664.4**	**883.7**	**1149.6**			
Total (hrs)	**11.1**	**14.7**	**19.2**			

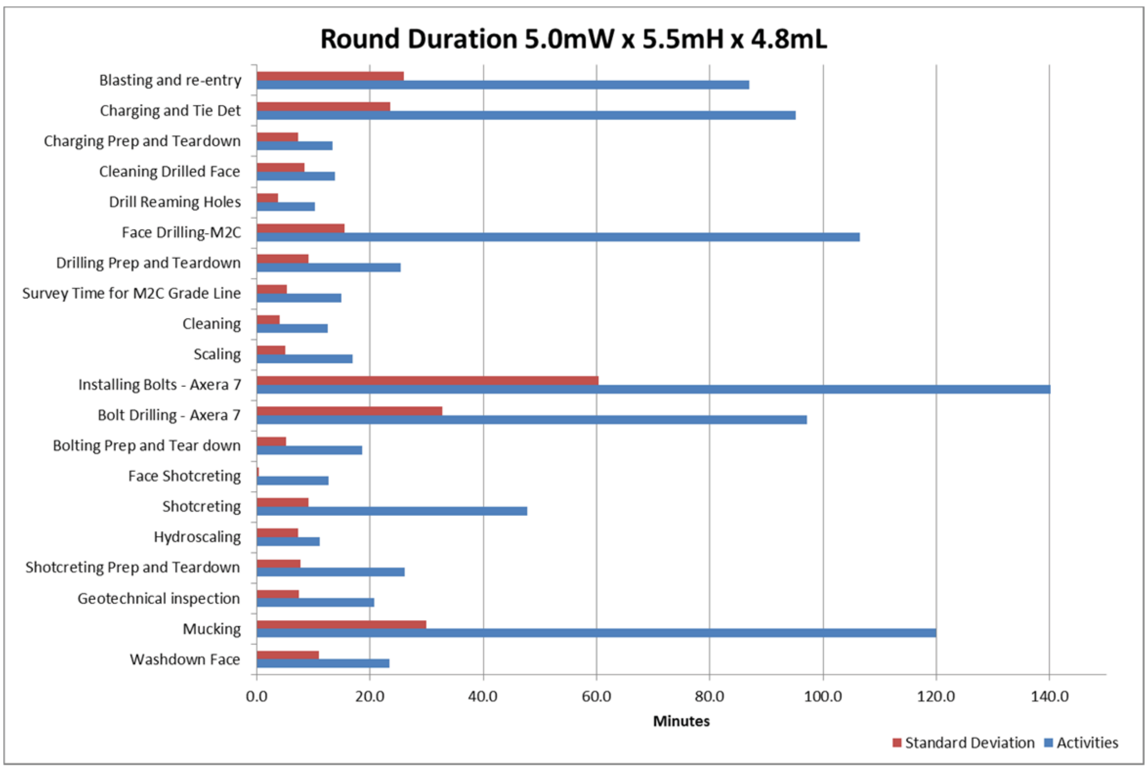


Table 4-10 Sample Activity Time of 5.8mWx5.8mH Heading

5.8x5.8 Face						
Activities	**Min(- 1Std)**	**Mean**	**Max(+1Std)**	**Std**	**Probability**	**Remark**
Washdown Face	12.5	23.5	34.5	11.0	100%	
Mucking	90.0	130.0	170.0	40.0	100%	Example time - SimMine will calculate
Geotechnical inspection	13.3	20.8	28.3	7.5	50%	
Shotcreting Prep and Teardown	18.4	26.2	34.0	7.8	100%	
Hydroscaling	3.9	11.2	18.5	7.3	50%	
Shotcreting	57.9	71.7	85.5	13.8	100%	4.8min x 15m^3
Face Shotcreting	12.0	15.0	18.0	3.0	100%	
Bolting Prep and Tear down	13.5	18.6	23.8	5.2	100%	
Bolt Drilling - Axera 7	71.1	85.2	113.4	28.1	100%	4 rows of 10 bolts
Installing Bolts - Axera 7	91.7	116.9	167.2	50.3	100%	4 rows of 10 bolts
Scaling	12.0	17.0	22.0	5.0	70%	
Cleaning	10.5	14.5	18.5	4.0	80%	
Survey Time for M2C Grade Line	9.6	15.0	20.4	5.4	100%	
Drilling Prep and Teardown	16.2	25.5	34.7	9.2	100%	
Face Drilling-M2C	115.5	135.2	154.9	19.7	100%	1.7 min x 80 holes
Drill Reaming Holes	6.5	10.3	14.2	3.8	100%	2 reaming holes
Cleaning Drilled Face	5.3	13.8	22.2	8.5	50%	
Charging Prep and Teardown	6.1	13.4	20.7	7.3	100%	
Charging and Tie Det	90.8	120.8	150.8	30.0	100%	1.5 min x 80 holes
Blasting and re-entry	60.9	86.9	113.0	26.1	100%	
Total (min)	**700.9**	**940.6**	**1219.6**			
Total (hrs)	**11.7**	**15.7**	**20.3**			

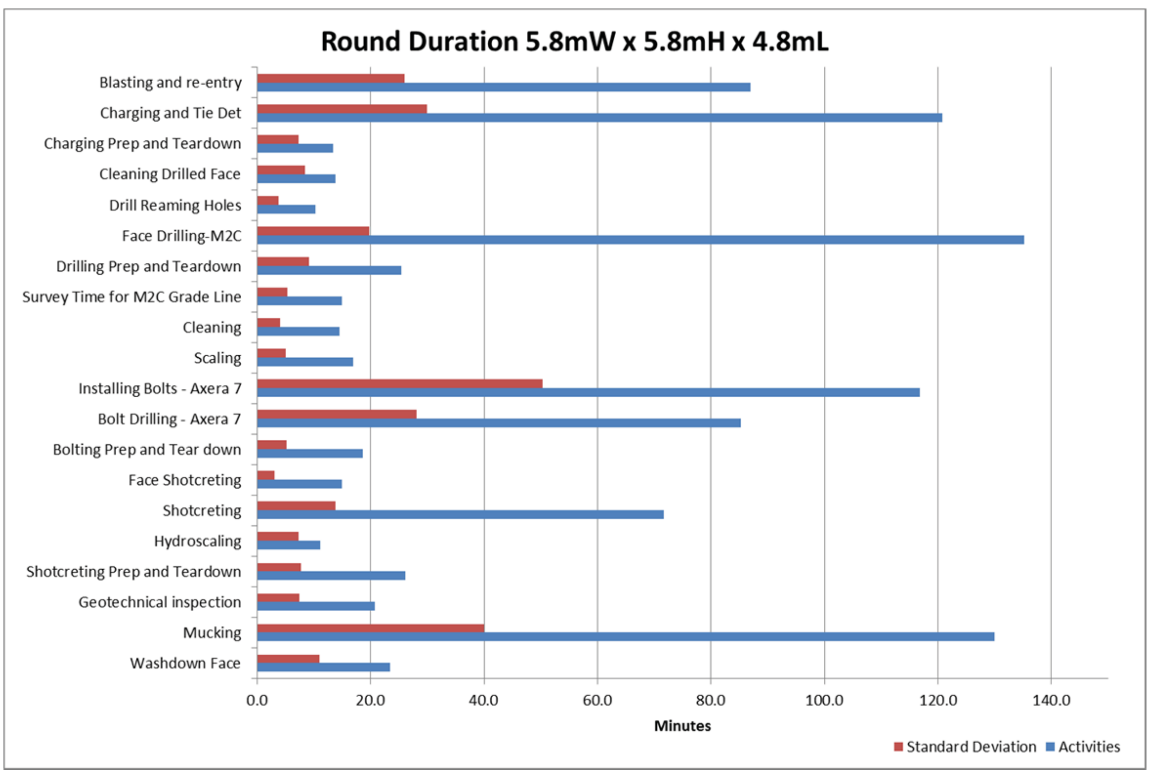
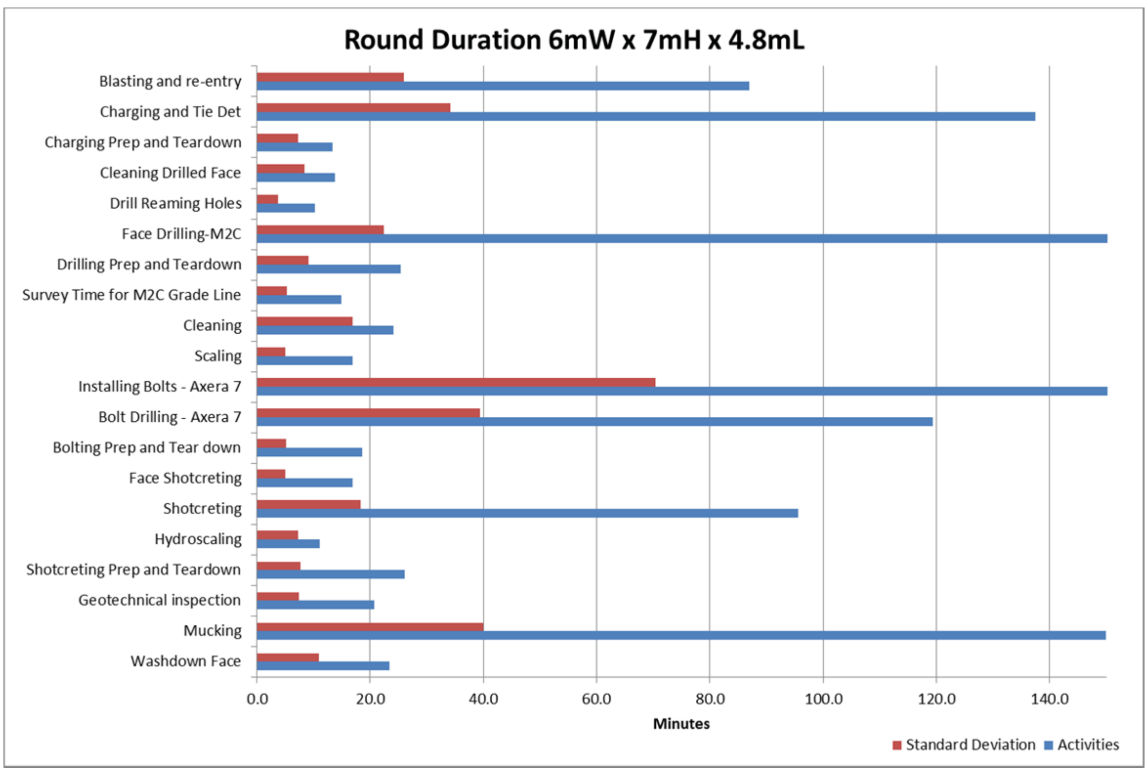


Table 4-11 Sample Activity Time of 5.8mWx5.8mH Heading

6.0x7.0 Face						
Activities	**Min(- 1Std)**	**Mean**	**Max(+1Std)**	**Std**	**Probability**	**Remark**
Washdown Face	12.5	23.5	34.5	11.0	100%	
Mucking	110.0	150.0	190.0	40.0	100%	Example time - SimMine will calculate
Geotechnical inspection	13.3	20.8	28.3	7.5	50%	
Shotcreting Prep and Teardown	18.4	26.2	34.0	7.8	100%	
Hydroscaling	3.9	11.2	18.5	7.3	50%	
Shotcreting	77.3	95.6	114.0	18.4	100%	4.8 min x 20 m^3
Face Shotcreting	12.0	17.0	22.0	5.0	100%	
Bolting Prep and Tear down	13.5	18.6	23.8	5.2	100%	
Bolt Drilling - Axera 7	99.6	119.3	158.7	39.4	100%	4 rows of 14 bolts
Installing Bolts - Axera 7	128.4	163.6	234.1	70.5	100%	4 row x 14 bolts
Scaling	12.0	17.0	22.0	5.0	70%	
Cleaning	7.2	24.1	41.0	16.9	80%	
Survey Time for M2C Grade Line	9.6	15.0	20.4	5.4	100%	
Drilling Prep and Teardown	16.2	25.5	34.7	9.2	100%	
Face Drilling-M2C	131.4	153.8	176.2	22.4	100%	1.7 min x 91 holes
Drill Reaming Holes	6.5	10.3	14.2	3.8	100%	2 reaming holes
Cleaning Drilled Face	5.3	13.8	22.2	8.5	50%	
Charging Prep and Teardown	6.1	13.4	20.7	7.3	100%	
Charging and Tie Det	103.3	137.4	171.6	34.1	100%	1.5 min x 91 holes
Blasting and re-entry	60.9	86.9	113.0	26.1	100%	
Total (min)	**831.0**	**1110.3**	**1444.4**			
Total (hrs)	**13.9**	**18.5**	**24.1**			



The mean values and standard deviations of the unit activity times are listed in Table 4-12. The different models of LHDs were considered to have the same loading, tramming and dumping time in the model. Only the LH517 loader's data was collected, and the loading and dumping (to a truck) time were each 42 seconds. The trucks' dumping time to the shaft was assumed to be 1 minute. The general category table lists all the unit activity times that were applied to all of the types of headings. For example, drilling a hole on a face would take 1.7 minutes no matter on the size of heading onto which it was drilled. This activity times only depend on the types of drill jumbo utilized. The next three tables show unique activity times that varies with the heading profiles. For example, cleaning a face for a 6mWx7mH heading may take longer than the regular 5mWx5.5mH heading because there are more face drill holes in the 6mWx7mH heading. The unit cable bolt time is listed in the last table.

Table 4-12 Mean Value and Standard Deviations of Unit Activity Times

	SimMine Inputs		
Loading and Transport Times	**Std**	**Mean**	**Units**
LHD Loading, tramming and dumping-LH517			
LH517 loading time	10.0	**41.5**	Second
LH517 tramming speed		**7.9**	km/hour
LH517 dumping time	0.0	**42.0**	Second
LHD Loading, tramming and dumping-Toro7			
Toro7 loading time			Second
Toro7 tramming speed			Meter/sec
Toro7 dumping time			Second
LHD Dumping to Truck-LH517			
Toro7			Second/bucket
LH517	21.0	**42.0**	Second/bucket

General	SimMine Inputs		
Activities Times	Std	Mean	Units
Washdown Face	11.0	**23.5**	Minute
Cleaning Drilled Face	8.5	**13.8**	Minute
Hydroscaling	7.3	**11.2**	Minute
Survey Time for M2C Grade Line	5.4	**15.0**	Minute
Drilling Prep and Teardown	9.2	**25.5**	Minute
Charging Prep and Teardown	7.3	**13.4**	Minute
Charging and Tie Det	0.4	**1.5**	Minute/hole
Blasting and re-entry	26.1	**86.9**	Minute
Geotechnical inspection	7.5	**20.8**	Minute
Shotcreting Prep and Teardown	7.8	**26.2**	Minute
Shotcreting	0.9	**4.8**	Minute/m3
Transmixer Preparation	5.0	**15.0**	Minute
Bolting Prep and Tear down	5.2	**18.6**	Minute
Face Drilling-M2C	0.2	**1.7**	Minute/hole
Face Reaming Holes	3.8	**10.3**	Minute
Installing Bolts			
Axera 7	1.3	**2.9**	Minute/bolt
DS310	2.9	**6.3**	Minute/bolt

5.0x5.5 Face	SimMine Inputs		
Activities Times	Std	Mean	Units
Face Shotcreting	0.3	**12.7**	Minute
Bolt Drilling			
Axera 7	0.7	**2.0**	Minute/bolt
DS310	0.6	**3.1**	Minute/bolt
Face Support			
Scaling	5.0	**17.0**	Minute
Screening	10.0	**49.0**	Minute
Cleaning	4.0	**12.5**	Minute

6.0x7.0 Face	SimMine Inputs		
Activities Times	Std	Mean	Units
Face Shotcreting			Minute
Bolt Drilling			
Axera 7	0.7	**2.1**	Minute/bolt
DS310			Minute/bolt
Face Support			
Scaling	5.0	**17.0**	Minute
Screening	5.5	**33.7**	Minute
Cleaning	16.9	**24.1**	Minute

5.8x5.8 Face	SimMine Inputs		
Activities Times	Std	Mean	Units
Cleaning Drilled Face			Minute
Face Shotcreting	3.0	**15.0**	Minute
Bolt Drilling			
Axera 7	0.7	**2.1**	Minute/bolt
DS310			Minute/bolt
Face Support			
Scaling			Minute
Screening	5.0	**19.0**	Minute
Cleaning	4.0	**14.5**	Minute

Intersection Cable Bolting	SimMine Inputs		
Activities Times	Std	Mean	Units
Setup	1.1	**14.1**	Minute
Drill	3.9	**16.0**	Minute/hole
Pumping Cement	0.7	**5.2**	Minute/hole
Install	3.7	**16.9**	Minute/bolt
Idle	42.0	**42.0**	Minute

4.3.2 On Footprint Activity Time

The ground support requirements for the on footprint development drifts were quite different because the geotechnical conditions changed significantly. The support regime included more rock bolts and cable bolts, mesh, mesh straps (if needed) and a second layer of shotcrete. Thus, activity times such as cable bolting, shotcreting, and meshing varied from the off footprint areas. The other activity times except for the ground supports were the same as those for the off-footprint drifts with the same profiles. The sample activity times of on footprint ground support activities are listed in Table 4-13 to Table 4-16. The shotcreting activity time in the model was calculated by unit spraying time (cubic meter sprayed) by the amount of shotcrete applied per meter advance. On the footprint area, two layers of shotcrete were normally applied but there was only one input parameter of shotcrete amount for each heading. Therefore, two shotcreting activities were created for undercut drives, extraction drives and perimeter drifts. The average amount of 75mm from the two layers (50mm and 100mm) was input for these drifts. This would not change the total shotcreting activity time for two layers.

Table 4-13 Sample Undercut and Extraction Drives Ground Support and Activity Times

*Yellow color is assumption			
Undercut drives			
(RMR>30) Type 1	**Activity time (min)**	**(RMR<30) Type 2**	**Activity time (min)**
50mm shotcrete whole profile	10m^3	50mm shotcrete first layer	10m^3
2.4m resin rebar at 1m grid (9@1m)	231	Mesh with 0.5m splitset at 2m grid	120
		2.4m resin rebar with straps at 1m grid (9@1m)	311
		6m cable bolts at 2m grid (5@2m)	422
		75mm second layer shotcrete	10m^3
Extraction drives (turnouts not included)- 65% of extraction drives			
50mm first layer shotcrete	10m^3	50mm first layer shotcrete	10m^3
Mesh with 0.5m splitset at 2m grid	150	Mesh with 0.5m splitset at 2m grid	150
2.4m resin rebar at 1m grid (10@1m)	254	2.4m resin rebar with straps at 1m grid (10@1m)	344
6m cable bolts at 2m grid (6@2m)	504	6m cable bolts at 1m grid (10@1m)	1646
100mm second layer shotcrete	15m^3	100mm second layer shotcrete	15m^3

Table 4-14 Sample Extraction Level Turnout Ground Support and Activity Times

*Yellow color is assumption			
(RMR>30) Type 1	**Activity time (min)**	**(RMR<30) Type 2**	**Activity time (min)**
Turnouts (7mLx4.5mW)- 35% of extraction drives			
50mm first layer shotcrete	5m^3	50mm first layer shotcrete	5m^3
Mesh with 0.5m splitset at 2m grid	90	Mesh with 0.5m splitset at 2m grid	90
2.4m resin rebar at 1m grid on back (4@1m)	113	2.4m resin rebar with straps at 1m grid on back (4@1m)	173
2.4m resin rebar with straps at 1m grid on bullnose (4@1m for 6m)	173	2.4m resin rebar with straps at 1m grid on bullnose (4@1m for 6m)	173
6m cable bolts at 2m grid on back (2@2m)	177	6m cable bolts at 1m grid (4@1m)	667
3 cable on 1m vert spacing on bull nose (3 cables)	102	3 cable on 1m vert spacing on bull nose (3 cables)	102
100mm second layer shotcrete	5m^3	100mm second layer shotcrete	5m^3

Table 4-15 Sample Perimeter Drift Ground Support and Activity Times

*Yellow color is assumption			
(RMR>30) Type 1	Activity time (min)	(RMR<30) Type 2	Activity time (min)
Undercut and extraction perimeter drifts (Type 1 and 2 same)			
50mm first layer shotcrete	10m^3	50mm first layer shotcrete	10m^3
Mesh with 0.5m splitset at 2m grid	200	Mesh with 0.5m splitset at 2m grid	200
2.4m resin rebar with straps at 1m grid (14 bolts@1m)	548	2.4m resin rebar at 1m grid (14botls@1m)	498
6m cable bolts at 1m grid (14 cables@1m)	2299	6m cable bolts at 1m grid (14 cables@1m)	2299
100mm second layer shotcrete	10m^3	100mm second layer shotcrete	10m^3

Table 4-16 Sample Drawpoint Ground Support and Activity Times

*Red colour is assumption			
(RMR>30) Type 1	Activity time (min)	(RMR<30) Type 2	Activity time (min)
Drawpoints			
50mm first layer shotcrete	10m^3	50mm first layer shotcrete	10m^3
Mesh with 0.5m splitset at 2m grid	150	Mesh with 0.5m splitset at 2m grid (chain mesh with higher deformation capability)	150
2.4m resin rebar with straps at 1m grid (10 bolts@1m)	404	2.4m resin rebar with straps at 1m grid (10 bolts@1m)	344
Cable bolts for 2 rows of 3 cables in brow area (6 cables)	204	6m cable bolts at 1m grid (10 cables@1m for 3m)	1034
Steelset (2 sets@1m)	300	Steelset (4 sets@1m, 1m grid cables inside)	1634
100mm second layer shotcrete	10m^3	100mm second layer shotcrete	10m^3

4.4 Shift Schedule and Working Days

A standard 9.5 hour shift and two shifts per day were employed in this model. The day shift started at 6 am and the night shift started at 6 pm. Both shifts included one hour and 30 minutes safety meetings at the outset, meal breaks half-way through the shifts and travel time back to surface. Underground blasting took place during the shift changing times. However, blasting and ventilation re-entry usually took 87 minutes and might take longer than the shift change time. Thus, there was an approximately 27-minute delay for the gas test which checked the contents of hazard gas resulted from blasting before the underground work could start. Table 4-17 illustrates the detailed day and night shift schedule in the simulation. The model utilized 350 work days per year. 15 days were removed each year to account for dust storms during the 6-month summer period. Table 4-18 lists the scheduled non-working days each year. The underground mine at the OT was scheduled to shut down during the Shaft 1 changeover, which was included in the model. Figure 4-10 shows the scheduled monthly work days until 2017. During the changeover, Shaft 1 would be retrofitted with a 3500 tpd skipping capacity system and 1.8 meter diameter ventilation ducts.

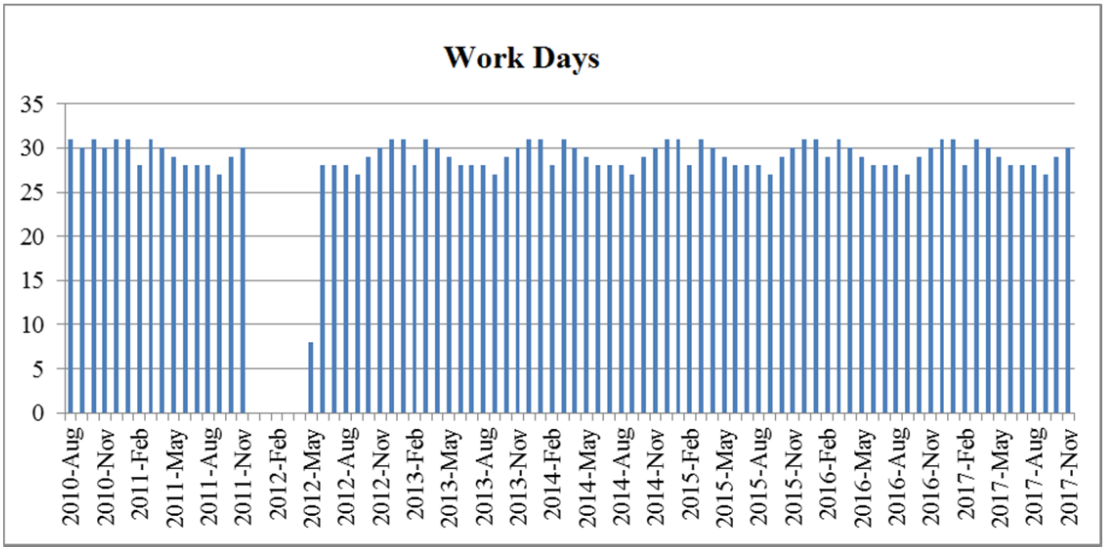
Table 4-17 Shift Schedule

Day Shift	From	To		**Night Shift**	From	To
Safety meeting	6:00	6:30		Safety meeting	18:00	18:30
Head underground	6:30	7:00		Head underground	18:30	19:00
Work	7:00	12:00		Work	19:00	0:00
Lunch	12:00	13:00		Lunch	0:00	1:00
Work	13:00	17:30		Work	1:00	5:30
Return to Shaft	17:30	17:45		Return to Shaft	5:30	5:45
Go to surface	17:45	18:00		Go to surface	5:45	6:00

Table 4-18 Scheduled Non-working Days Every Year

Date	Days
May 1-2	2
June 6-7	2
July 4-6	3
Aug 7-9	3
Sept 4-6	3
Oct 2-3	2
Total	15

Figure 4-10 Monthly Work Days and the Mine Shutdown



4.5 Development Equipment

The underground mobile equipment utilized in this simulation model was grouped by development crews. Each standard development crew grouping includes separate equipment and equipment which is shared between the crews. The separate equipment fleet include one 17-tonne LHD (LH517), one bolter (Boltec), one drill jumbo (M2C), one cable bolter (Cabletec), one charger (Charmec), and one shotcrete sprayer (Spraymec). 50-tonne trucks (TH550) and 5m^3 agi-trucks (Utimec 1600) were shared by all the development crews and cable bolters were share between two crews.

The current development fleet at the OT underground mine had two development crews. However, only one drill jumbo (M2C) was utilized primarily for face drilling. Axera 7 was occasionally used for face drilling so it was switched to a bolter in the simulation. However, in the MS2 model's validation period a multi-functional machine which allowed drilling, bolting and screening was included to represent the Axera 7's activities.

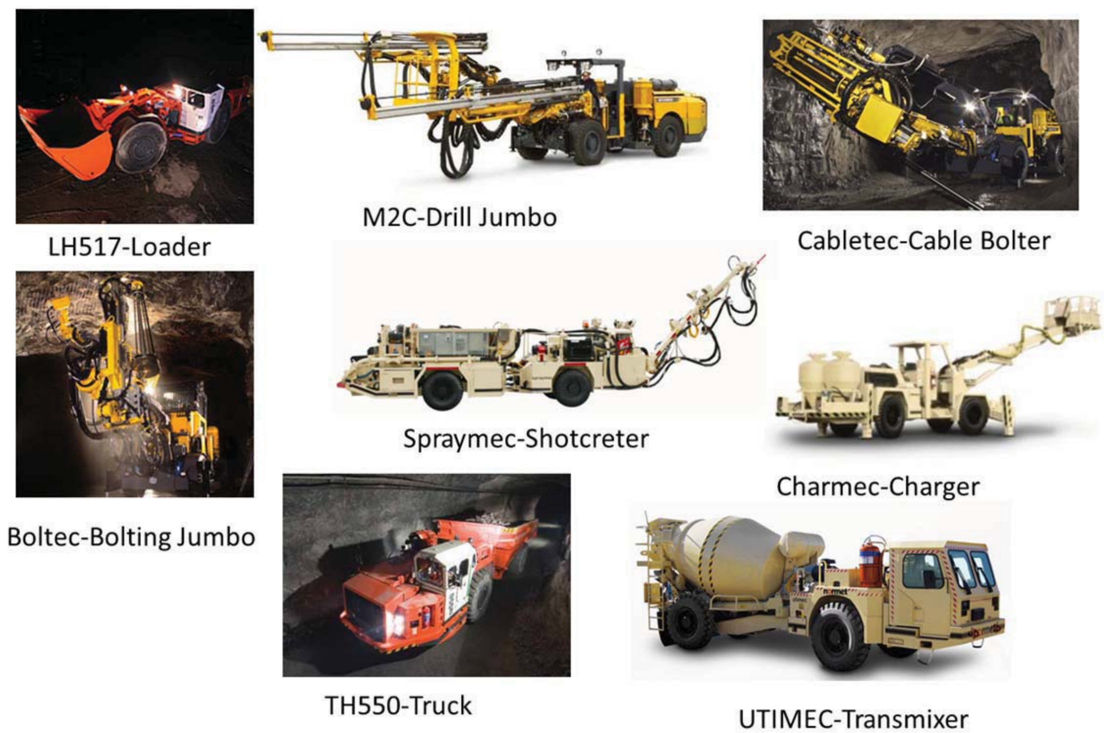
The current underground loading and hauling fleet at the OT included one 17-tonne loader (LH517), one 10-tonne loader (Toro 7), one 6.7-tonne loader (Toro 6); one

30-tonne truck (EJC530) and one 50-tonne truck (TH550). It was found that in reality the loading and hauling equipment could not achieve the manufacturer's suggested maximum capacity. Based on the empirical data from the other block caving copper mines, the LHDs could load about 85% of the maximum capacity and trucks could load about 90%. Therefore, all the LHDs' true bucket weights were reduced by 15% and trucks were reduced by 10%

In the model, each type of development equipment (as illustrated in Figure 4-11) was only allowed to be involved in certain development activities. The permitted activities for each type were:

- Drill jumbo: face drilling
- Bolter: bolt drilling, bolting, and screening
- Cable bolter: cable bolting
- Charger: charging
- Sprayer: shotcreting
- Loader: mucking and cleaning
- Truck and Agi-truck: loading and transporting

Figure 4-11 Overview of Typical Development Equipment (Sandvik and Atlas Copco, 2011)



4.5.1 Equipment Tramming Speed

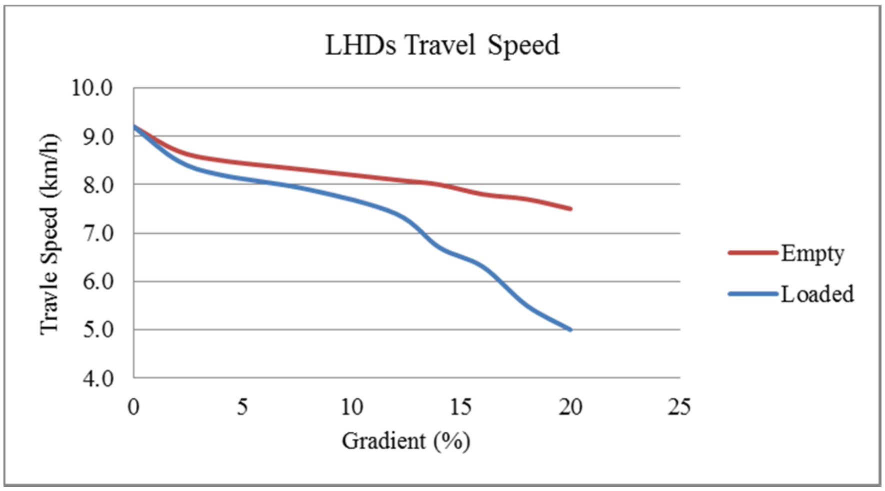
Development equipment usually travels very slowly in underground mines. The speed input parameter is critical for frequently tramming vehicles such as LHDs, trucks and agi-trucks. For the other types of equipment tramming speeds only affect the preparation and teardown processes when they travel between the parking locations and the faces. Most of the times these types of equipment are parked near the working face to reduce tramming time. The tramming speeds of the drill jumbo, truck and transmixer were measured from the OT site. The LHD's tramming speeds were based on the manufacturer's suggestions in Figure 4-12. The speeds used in the simulation are listed below:

- Drill Jumbo: 5km/h
- Charger: 6km/h (assumption)
- Sprayer: 6km/h (assumption)

- Transmixer: 6km/h
- Bolting Jumbo: 5km/h (assumption)
- Trucks:6.6km/h upward, 7.8km/h flat and descending

  *Speeds are based on site data of TH550, DS310 and UTIMEC: the time of arrival at the face divided by the tramming distance

Figure 4-12 Manufacturer's Suggested Travel Speed for Underground Loaders



4.5.2 Crew Ramp-up and Allowed Mine Areas

1) Crew Ramp-up

Development crew ramp-up was restricted by the ventilation capacity in the PPD simulation model. Figure 4-13 shows the ramp-up of each piece of development equipment until July, 2013. Figure 4-14 illustrates the ramp-up curve of development crews. Two crews continued to work until the completion of Shaft 1's changeover and commissioning of Ventilation Raise 1 (VR 1) in June, 2012. In the following two months one crew was added each month. In July, 2013 VR 3 would be commissioned and the crews would ramp up from four to six. In December, 2013, VR 2 would be commissioned. An additional development crew was added at this time. The ventilation shaft would be commissioned in December, 2014 at which point the ventilation capacity would no longer be a constraint to the number of development crews. At this time the total number of development crews would start to ramp up from 7 to 12 (Wolgram, 2011).

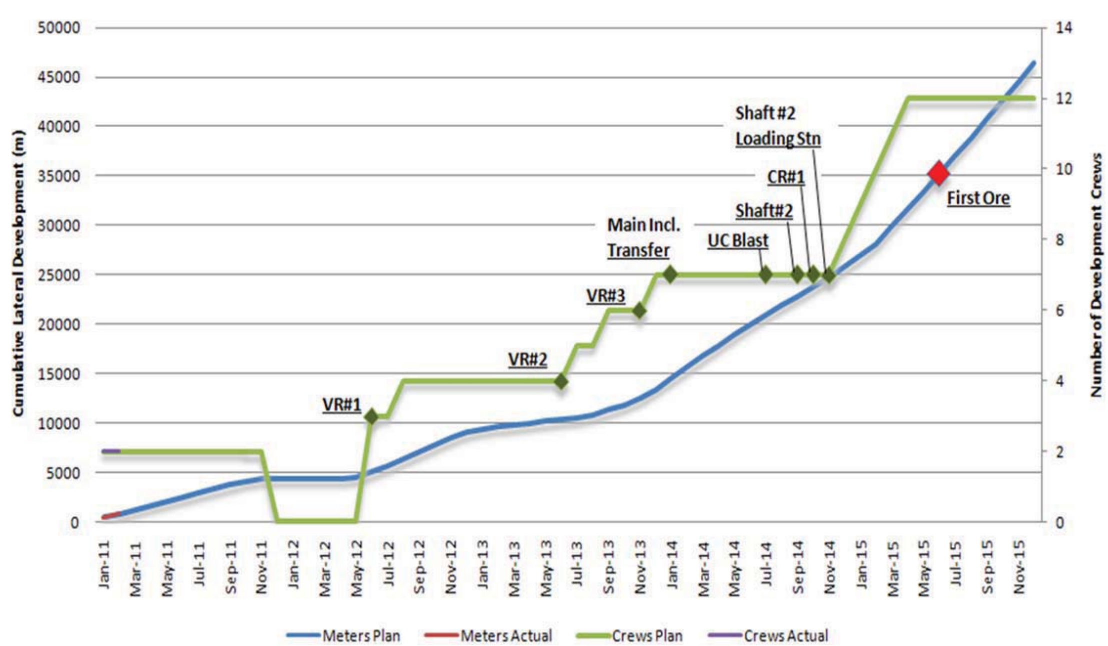
Figure 4-13 Example of Equipment Ramp-up until July 2013

Aug-10	Aug-11	Sep-11	Jun-12	Aug-12	Jul-13
J-Atlas Copco M2C-01	J-Atlas Copco M2C-01	J-Atlas Copco M2C-01	J-Atlas Copco M2C-01	J-Atlas Copco M2C-01	J-Atlas Copco M2C-01
B-Sandvik DS310-01	B-Sandvik DS310-01	J-Atlas Copco E2C-02	J-Atlas Copco E2C-13	J-Atlas Copco E2C-15	J-Atlas Copco E2C-26
B-Sandvik Axera 7-0	B-Sandvik Axera 7-0	B-Sandvik DS310-01	J-Atlas Copco M2C-03	J-Atlas Copco M2C-03	J-Atlas Copco M2C-03
T-Sandvik EJC530-1	B-Atlas Copco Boltec-02	B-Sandvik Axera 7-0	B-Sandvik DS310-01	J-Atlas Copco M2C-04	J-Atlas Copco M2C-04
T-Sandvik 50-1	T-Sandvik EJC530-1	B-Atlas Copco Boltec-02	B-Sandvik Axera 7-0	B-Sandvik DS310-01	J-Atlas Copco M2C-05
L-Sandvik LH307-0	T-Sandvik 50-1	T-Sandvik EJC530-1	B-Atlas Copco Boltec-02	B-Sandvik Axera 7-0	B-Sandvik DS310-01
L-Sandvik LH140-02	L-Sandvik LH307-0	T-Sandvik 50-1	B-Atlas Copco Boltec-03	B-Atlas Copco Boltec-02	B-Sandvik Axera 7-0
L-Sandvik LH517-01	L-Sandvik LH140-02	L-Sandvik LH307-0	T-Sandvik EJC530-1	B-Atlas Copco Boltec-03	B-Atlas Copco Boltec-02
CR-Normet Spaymec-01	L-Sandvik LH517-01	L-Sandvik LH140-02	T-Sandvik 50-1	B-Atlas Copco Boltec-04	B-Atlas Copco Boltec-03
CR-Normet Spaymec-02	CR-Normet Spaymec-01	L-Sandvik LH517-01	T-Sandvik 50-2	T-Sandvik EJC530-1	B-Atlas Copco Boltec-04
CR-Normet UTIMEC-0	CR-Normet Spaymec-02	CR-Normet Spaymec-01	L-Sandvik LH307-0	T-Sandvik 50-1	B-Atlas Copco Boltec-05
CR-Normet UTIMEC-01	CR-Normet UTIMEC-0	CR-Normet Spaymec-02	L-Sandvik LH140-02	T-Sandvik 50-2	T-Sandvik EJC530-1
CR-Normet UTIMEC-02	CR-Normet UTIMEC-01	CR-Normet UTIMEC-0	L-Sandvik LH517-01	L-Sandvik LH307-0	T-Sandvik 50-1
CB-Sandvik Cabletec-1_2	CR-Normet UTIMEC-02	CR-Normet UTIMEC-01	L-Sandvik LH517-03	L-Sandvik LH140-02	T-Sandvik 50-2
CG-Charmec LC-01	CB-Sandvik Cabletec-1_2	CR-Normet UTIMEC-02	CR-Normet Spaymec-01	L-Sandvik LH517-01	T-Sandvik 50-3
CG-Charmec LC-02	CG-Charmec LC-01	CB-Sandvik Cabletec-1_2	CR-Normet Spaymec-02	L-Sandvik LH517-03	L-Sandvik LH307-0
	CG-Charmec LC-02	CG-Charmec LC-01	CR-Normet Spaymec-03	L-Sandvik LH517-04	L-Sandvik LH140-02
		CG-Charmec LC-02	CR-Normet UTIMEC-0	CR-Normet Spaymec-01	L-Sandvik LH517-01
			CR-Normet UTIMEC-01	CR-Normet Spaymec-02	L-Sandvik LH517-03
			CR-Normet UTIMEC-02	CR-Normet Spaymec-03	L-Sandvik LH517-04
			CR-Normet UTIMEC-03	CR-Normet Spaymec-04	L-Sandvik LH517-05
			CB-Sandvik Cabletec-1_2	CR-Normet UTIMEC-0	CR-Normet Spaymec-01
			CB-Sandvik Cabletec-3_4	CR-Normet UTIMEC-01	CR-Normet Spaymec-02
			CG-Charmec LC-01	CR-Normet UTIMEC-02	CR-Normet Spaymec-03
			CG-Charmec LC-02	CR-Normet UTIMEC-03	CR-Normet Spaymec-04
			CG-Charmec LC-03	CR-Normet UTIMEC-04	CR-Normet Spaymec-05
				CB-Sandvik Cabletec-1_2	CR-Normet UTIMEC-0
				CB-Sandvik Cabletec-3_4	CR-Normet UTIMEC-01
				CG-Charmec LC-01	CR-Normet UTIMEC-02
				CG-Charmec LC-02	CR-Normet UTIMEC-03
				CG-Charmec LC-03	CR-Normet UTIMEC-04
				CG-Charmec LC-04	CR-Normet UTIMEC-05
					CB-Sandvik Cabletec-1_2
					CB-Sandvik Cabletec-3_4
					CB-Sandvik Cabletec-5_6
					CG-Charmec LC-01
					CG-Charmec LC-02
					CG-Charmec LC-03
					CG-Charmec LC-04
					CG-Charmec LC-05

J-Jumbo
B-Bolter
T-Truck
L-LHD
S-Shotcrete
CR-Concrete
CB-Cablebolter
CG-Charger

*Equipment starts on the beginning of each month

Figure 4-14 Lateral Development Crews Ramp-up Graph (Wolgram, 2011)

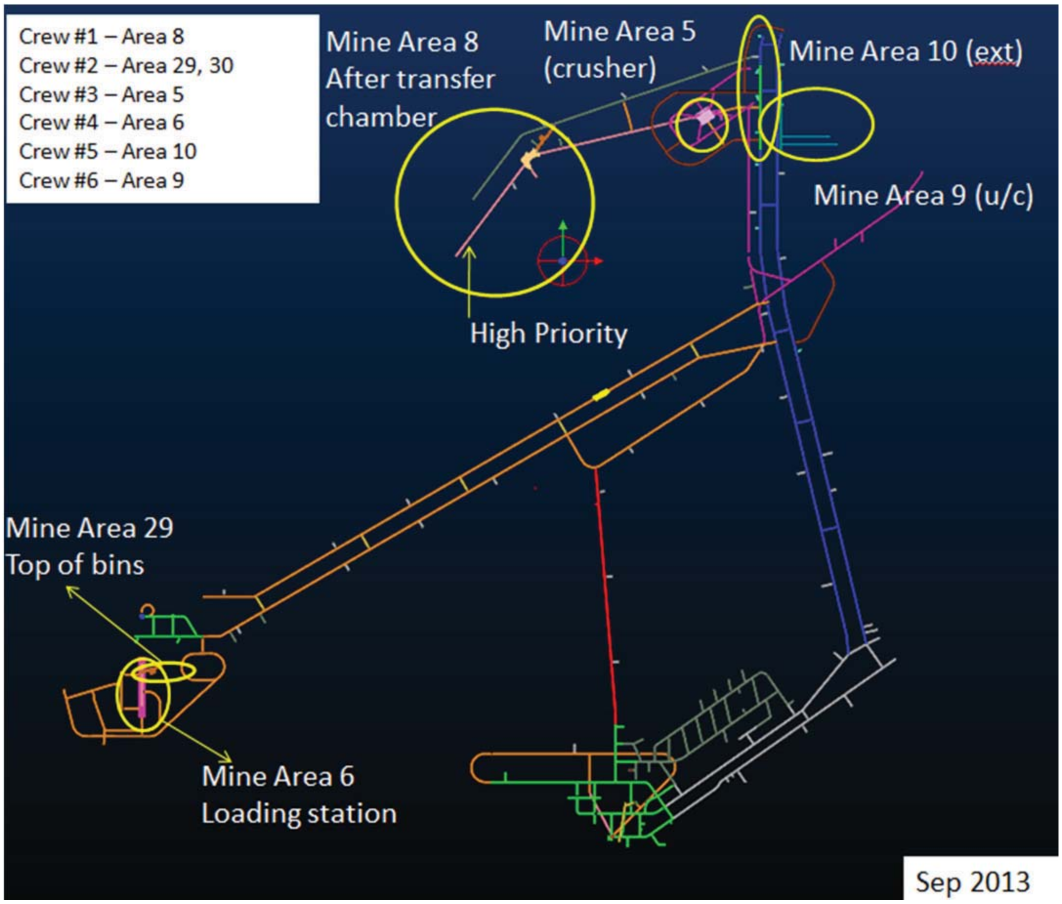


2) Mine Areas

The entire mine has been divided into several Mine Areas. Each crew was allowed to work in several mine areas, but usually in one area at a time. Occasionally, if two mine areas were close together, one crew could work at both. The purpose of the mine area allotment for development crews was to try to minimize equipment travel time and reduce congestion in the key travel routes. However, when available crew resources exceeded the available mine areas, for example, when there were 9 crews and 6 mine areas available at a given period, more than one crew would then be assigned to critical mine areas to more quickly advance the latter. An example of some of the mine area allocations is shown in Table 4-15. Six development crews were designated to work on six different mine areas.

Development headings were established as high priority or low priority. After defining the priorities, one resource would choose to work on the high priority headings if both the high and low priority headings were available at the same time. In the dual heading development environment, critical paths need to be driven as quickly as possible as high priority headings, while low priority ones could be delayed. Accounting to the example in Table 4-15, during dual heading development in Mine Area 8, the critical path of the main conveyor drift (pink) should be given higher priority.

Figure 4-15 Example of Mine Area Allocation



4.5.3 Preventive Maintenance and Availability

Preventive maintenance (PM) and breakdowns (random failures) of development equipment are directly related to the development rate. The PM schedules and random failures of each piece of equipment were studied based on 6 month data recorded from the OT site. This study can be found in Appendix I: A3. The maintenance schedule used in this simulation was derived from this study, which was listed in Table 4-19. The equipment availability data from the OT site, however, could not be used for this simulation program because the equipment's operating times were not recorded. The simulation model defined the availability as breakdown time / (operating time + breakdown time)%. After discussion with the OT's maintenance engineer and based on the same type of equipment's performance data from other mines, a conservative assumption of 80% availability was used for all underground development equipment. When the equipment was working, random failures would be generated in a lognormal distribution in the simulation model.

Table 4-19 Underground Mobile Equipment Preventive Maintenance Schedule

Equipment Name	Weekly	Biweekly	Monthly	250 HR	500 HR	1000 HR	2000 HR	Unit
Underground Loader				10	12	12	12	hour
Drill Jumbo	10			12	24	24	24	hour
Bolter		10		12	20	20	20	hour
Cable Bolter	10			12	20	20	20	hour
Underground Truck				8	12	12	12	hour
Sprayer		8		10	18	18	18	hour
Agi-truck		8		10	12	12	12	hour
Charger		6		8	12	12	12	hour

4.6 Development Muck Flows

Two types of rock were used in the simulation model. All drifts on or close to the footprint area were defined as ore type rock and off footprint area were defined as waste type rock. The specific density (SG) of the ore type rock was 2.8, and the waste type rock was 2.6. Both of the two types of rock were considered waste and would not be processed in the PPD period.

The development muck loading logistics in this simulation model were as follows: The mucks was first loaded from the face to the nearest remuck; when the remuck was full, a truck would arrive to empty the remuck with an LHD. The truck would then transport and dump the muck at the endpoints. An endpoint in the model was where the muck would go outside of the model. The endpoints included Shaft_1, Shaft_1_after_shutdown, Shaft_2 and Crusher_1. The time study revealed that in the real OT trucking operation a loader might sometimes load a truck directly from the face. The simulation program, however, was not capable of establishing complex logistics by which LHDs, for example, could intelligently choose between whether to load a truck directly when the latter was nearby or whether to load the nearest remuck. The total mucking time of the two loading logistics, however, should be close together, particularly when the development faces move farther from the shaft.

Shaft_2 and crusher_1 were given a start date of December 1, 2014. By this time, three end points would be available, and the development mucks would exit through the closest one. For example, in 2015, the development mucks close to the Shaft 2 areas would exit the model from the end point at Shaft_2. Shaft 1's changeover was modeled by two endpoints of different hoisting capacity. Before Shaft 1's changeover (mine shutdown), the mucks would exit from Shaft_1 which had a 1,100 tpd capacity, but after mine shutdown, Shaft_1 would be closed and the muck would exit from Shaft_1_after_shutdown which had a 3,600 tpd capacity.

4.6.1 Remuck Size and Locations

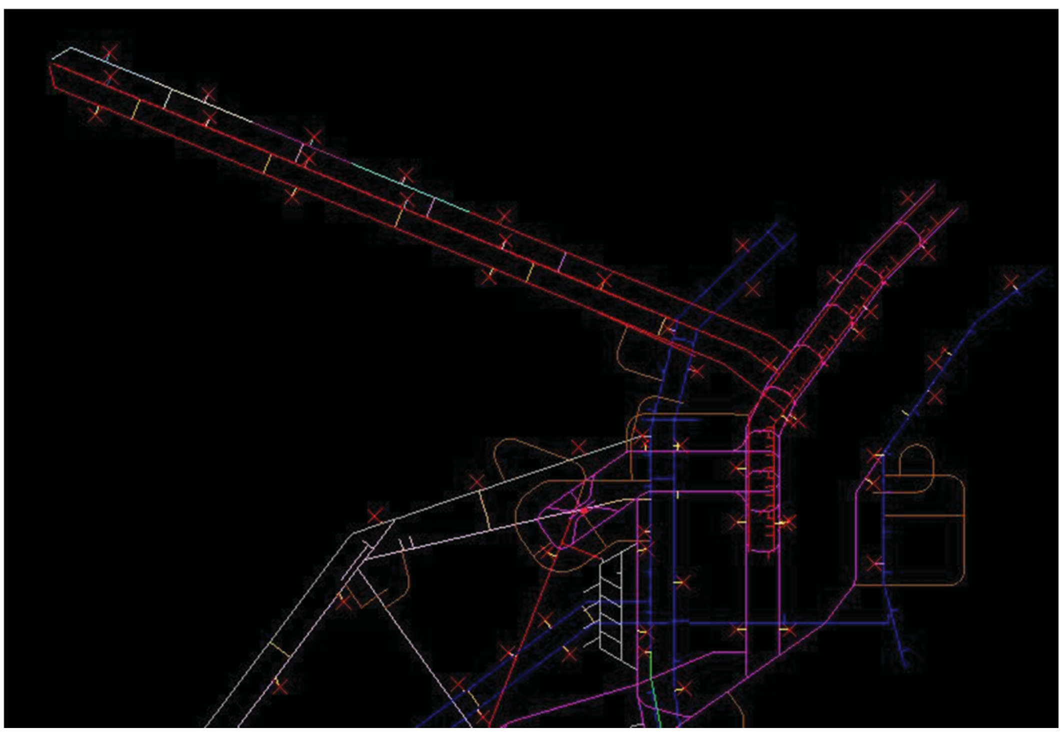
Remuck lengths were designed at 11, 15 and 20 meters. The standard 15-meter long (400 tonnes) lengths were used for all remucks in the MS2 model. After the model was completed, the 400-tonne remuck was found to not be large enough to hold one blasted round of mucks generated from headings larger than the "I" type (5.0mWx5.5mH). In the MS3 model, the remucks were changed to various lengths depending on the size of the headings.

Table 4-20 Remuck Sizing

Remuck Length (m)	Cross Section (m^2)	Available Volume(m^3)	% Utilized	Available Capacity (t)
11	26.53	292	60%	294
15	26.53	398	60%	401
20	26.53	531	60%	535

Figure 4-16 Remuck Locations: Plan View, Southern Portion

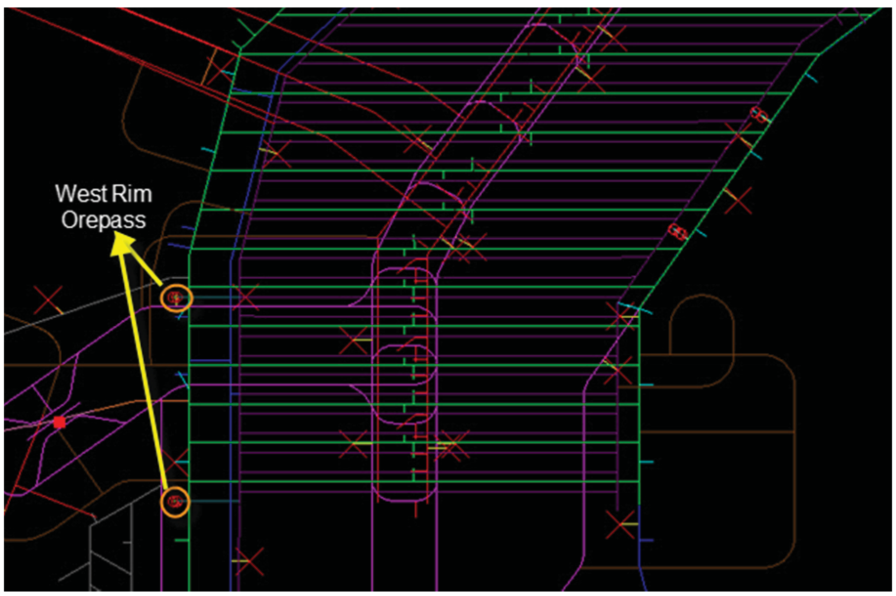
Figure 4-17 Remuck Locations: Plan View, Northern Portion



4.6.2 Extraction and Undercut Level Muck Flows

In the footprint area, the development mucks were all of ore type rock with a specific density of 2.8. They was loaded by LHDs and then dumped into the closest orepass located on the rims (perimeter drifts). Two orepasses were located on the west rim and the other two on the east rim (Figure 4-18). Eight extraction drives advancing from west to east were tested. Therefore, only the west rim orepasses were utilized in the model.

Figure 4-18 Locations of Footprint Orepasses



4.6.3 Muck Flow Reconciliation

The purpose of this reconciliation was to verify the muck loading logistics in the model. The reconciliation process was completed by comparing the mass of the muck generated from the development and exited from the end points. Before the mine’s shutdown, the development muck was stored in the loading zone near Shaft 1, and then loaded to the shaft bucket by a small LHD. There was 587,495 tons of development muck before the mine’s shutdown, and the same amount exited the model through Shaft_1. Details of the muck flows before the mine’s shutdown were well validated. In the footprint area, the development muck was loaded from the face and then dumped into the orepass connected with the extraction, undercut and haulage levels.

Table 4-24 shows that 153,341 tons of development muck was generated in the footprint areas and the same amount were loaded from the bottom of the orepasses. Therefore, the development muck flows before the mine shutdown and in the footprint areas were verified.

The total muck exiting the model through Shafts 1 and 2 and crusher 1 was 2,291,225 tonnes (Table 4-21). The total muck generated during the PPD period was 2,282,987 tonnes. This was the sum of the muck stored and moved from all the remucks (Table 4-23) and the muck stored at the orepass bottom (Table 4-24). The 8,000-tonne difference was because when the simulation ended some rocks still remained in the remucks. Therefore, the total muck flows during the PPD period were also verified.

Table 4-21 Summary of Muck Exiting the Model

	Ore	Waste	TOTAL MUCK
Shaft_1	220785	366703	**587488**
Shaft_1_after_shutdown	417003	1187398	**1604401**
Shaft_2	0	30644	**30644**
Crusher_1	27209	41483	**68692**
TOTAL	**664997**	**1626228**	**2291225**

Table 4-22 Shaft 1 Loading Zone Muck before Mine Shutdown

LB_shaft1_ore	**220788**	**0**	**587495**
LB_shaft1_waste	**0**	**366707**	

Table 4-23 Total Development Muck Stored in the Remucks (LB)

Waste Total (LB only)	**1464299**
Ore Total (LB only)	**665347**
Muck Total	**2129646**

Table 4-24 Orepass Muck Flows

Transfer_pass_ex_1	61257	0
Transfer_pass_ex_3_ramp	0	25939
Transfer_pass_ex_4	66145	0
OP_LB_ex_1	61257	0
OP_LB_ex_4	66145	0
OP_LB_ramp	0	25939

5. Results and Analysis

This chapter discusses the simulation tests and results from the Milestone 2 (MS2) and the Milestone 3 (MS3) models for the OT project. It begins with the model validation and verification process. It then demonstrates the simulated results of the development rate, considered the most important part of this research. The results from sensitivity and tradeoff tests from these two models are also outlined. Lastly, it provides an analysis of the key outcomes of the simulation research project.

5.1 Model Verification and Validation

Verification is the process for checking whether a model works correctly and whether it accurately represents the conceptual model. The validation process determines if the verified model can substitute for the real-world system, for instance in the development excavation system of an underground mine (Banks, 1998). Model verification ensures that the computer program underlying a software model and its implementations are correct. A model is valid when the theories and assumptions underlying it are correct and the simulations of this model represent the real system (Sargent, 2003).

The verification process of this study considered two major aspects:

1) Fixing the errors of the simulation program. This was conducted through a software debugging process in close collaboration with the software programmer. The simulation programming errors were reported to the software programmer once they were found. In this way, the computer program was improved interactively after over 20 major flaws and mistakes were repaired including the change of loading logics of the ore handling system.

2) Finding the input and logistic errors in the model. Due to their complexity, the errors could not be avoided due to the input parameters and logistics of the model. The model verification process included checking the mine layout, development cycles, activity times, muck handling logics, equipment fleet, and development

sequences. The mine development advancing sequences and the equipment movement over time could be visualized in 3D while the model was running. Therefore, the model was run many times to detect possible flaws before it was verified. The input activity times were reviewed by site engineers and the model itself was reviewed by the researcher, assisted by Rio Tinto – Ivanhoe – AMEC personnel.

Validation is performed to determine whether the verified model can be substituted for a real system such as the conventional underground excavation and development schedule at OT for the purpose of experimentation (Banks, 1998). The validation approach employed in this study was basically done by initiating the simulation a period earlier than the current date to repeat all development activities that had occurred at the OT underground mine during this period. This period was defined as the validation period. The model would then be run several times during this validation period to compare the difference in lateral development meters between the real-world system (OT's actual underground development performance) and the simulated model performance.

5.1.1 The MS2 Model's Validation

Simulation was begun in August, 2010. However, when the MS2 model was validated, the site already had actual development meters which would continue until July, 2011. These 11 months were used as the validation period (August 01, 2010 to June 30, 2011). The lateral development meters from the simulation were compared with the actual ones from the OT site. The mine as-built imported into this model was the same as the real OT site as-built as of July, 2010. The development sequence and available work headings were also set to be the same as those of the real site during these 11 months. All equipment was allowed to work in all areas. Nine validation runs were completed. All of the simulated results were within a 5% (mostly within a 3%) difference from the actual meters of the OT site. The major bottleneck of the whole development system was also identified by the simulation model. In this validation period, the OT development system was bottlenecked by Shaft 1's hoisting capacity, which was only 1200 tonnes per day. This resulted in a significant amount of development muck stored in the drifts and bays near Shaft 1. Table 5-1 shows nine simulated lateral development meter data over the validation period as compared with actual monthly development meters from the OT site. Figure 5-2 and Figure 5-3 compare the actual OT data and simulated mine as-built data until the end of the validation period of 30 June, 2011.

Table 5-1 The MS2 Model Validation Runs from August, 2010 to April, 211

	Actual m	Run 1	Run 2	Run 3	Run 4	Run 5	Run 6	Run 7	Run 8	Run 9
Aug-10	431	458	462	453	482	482	482	482	482	453
Sep-10	366	441	462	465	465	465	465	465	465	465
Oct-10	420	452	452	457	490	490	490	490	490	461
Nov-10	411	410	429	386	484	484	484	470	484	427
Dec-10	447	445	454	432	437	437	461	460	468	443
Jan-11	448	367	341	356	376	376	393	402	413	384
Feb-11	381	343	412	322	398	398	410	392	410	355
Mar-11	431	414	405	427	377	377	377	405	410	437
Apr-11	403	376	403	380	393	393	388	369	369	388
May-11	417	431	407	369	353	370	370	361	349	352
Jun-11	361	395	396	326	377	343	316	269	290	304
Total	4516	4412	4473	4325	4559	4495	4564	4493	4542	4401
Difference (%)		-2.3%	-1.0%	-4.2%	1.0%	-0.5%	1.1%	-0.5%	0.6%	-2.5%

Figure 5-1 Model Calibration to Actual Development – August, 2010 to June, 2011

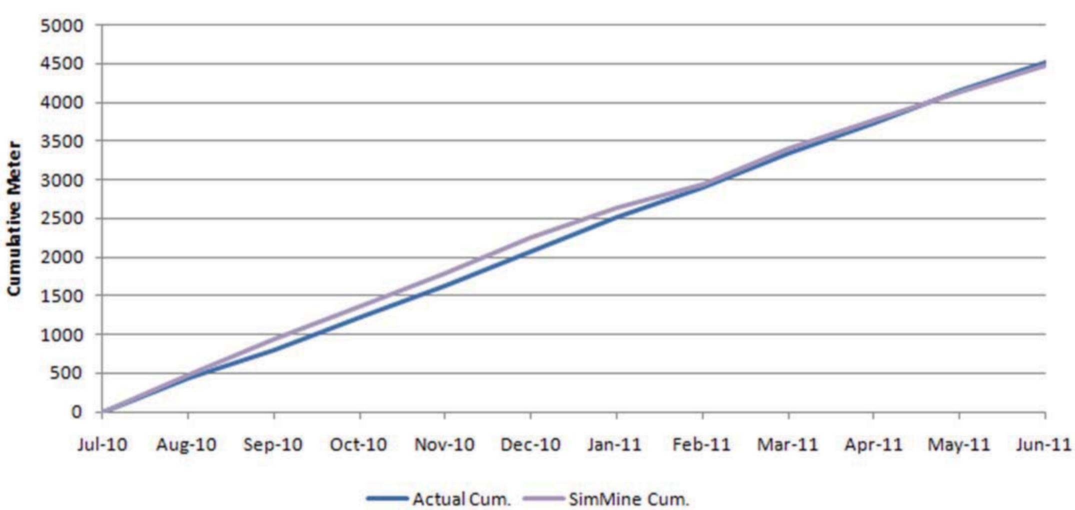


Figure 5-2 Actual Development Asbuilt of July 11, 2011

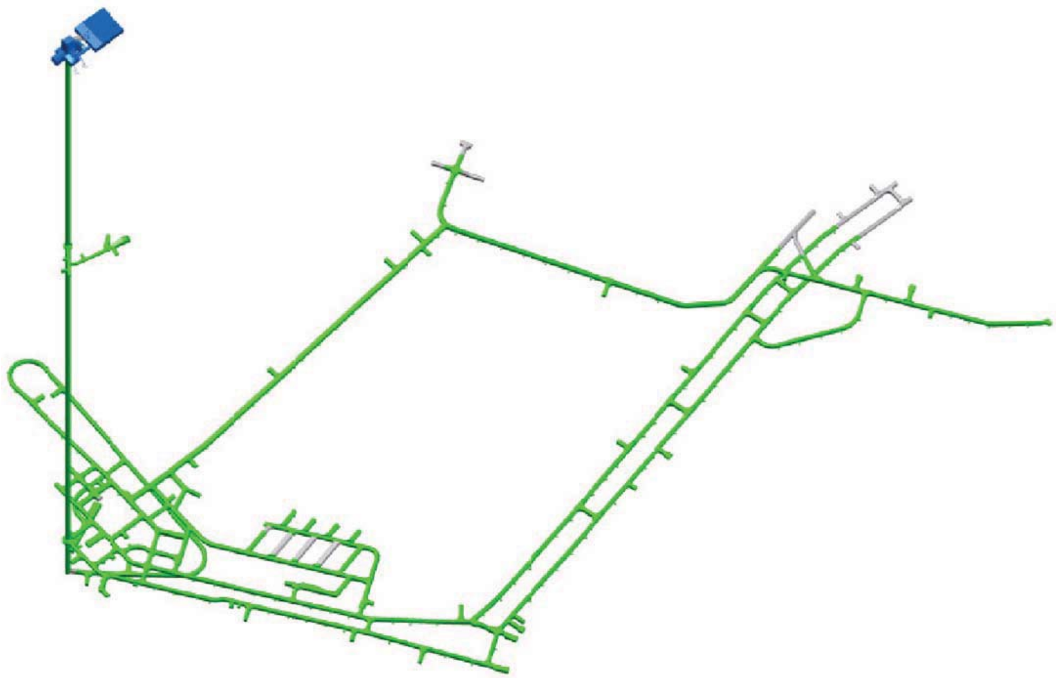
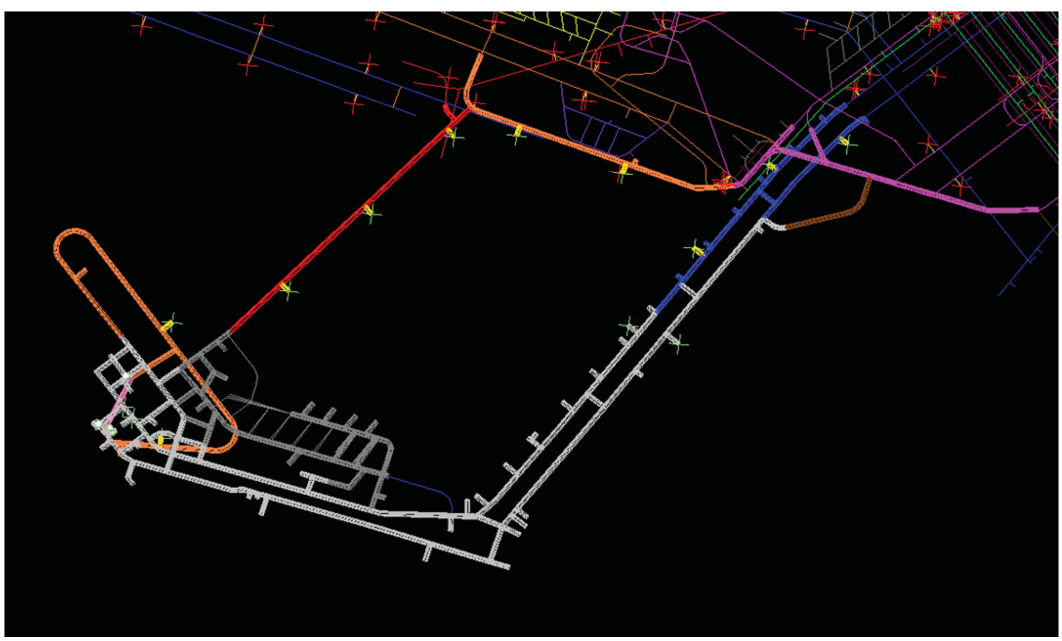


Figure 5-3 Simulated Development Asbuilt of 30 June, 2011



5.1.2 The MS3 Model's Validation

In the MS3 model, the validation method was similar to that used in the MS2 model. The inputs were the same as those of the MS2 model except that the mine design and as-built were changed. The first nine weeks of 2012 was used as the validation period. The validation results were compared with the actual lateral development meters that were recorded by the OT site.

Three validation runs were completed in the MS3 model. These results ranged from a total of 805 to 817 m during these nine weeks (as shown in Table 5-2), which was shown to be very close to the planned number of meters at the OT site. However, the actual recorded lateral development was approximately 100 meters less than the planned and simulated one. This was due to unexpected delays and equipment breakdowns at the OT site during these nine weeks.

Table 5-2 The MS3 Model Validation Results from 1 January, 2012 to 1 March, 2012

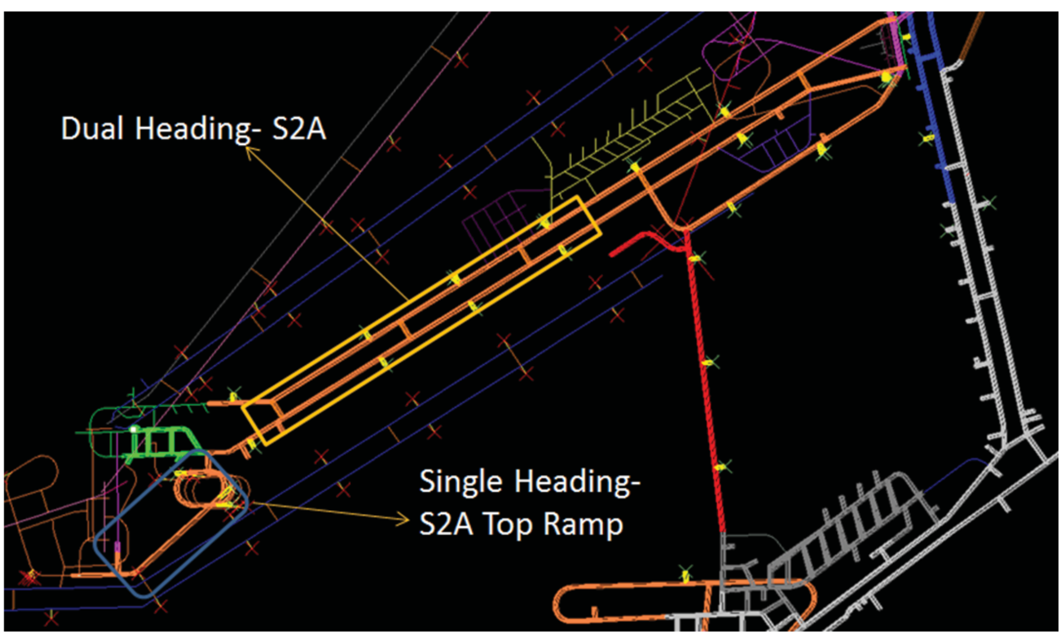
	OT Site		Simulated		
2012	Planned	Actual	Run #1	Run #2	Run #3
Week 1	112	77	80	113	116
Week 2	101	121	99	87	101
Week 3	103	95	92	118	134
Week 4	102	84	80	80	100
Week 5	102	57	122	89	102
Week 6	90	73	95	75	84
Week 7	83	79	99	90	65
Week 8	79	72	70	81	58
Week 9	42	50	80	75	45
Total	814	708	817	808	805

5.2 Base Case Test

The base case test had targeted specific development of Shaft 2 access drives and Shaft 2 top ramps to determine local development rates. Crew Number 2 was designated to work on these areas. Other possible working areas were disabled during the simulation. Therefore, this crew would only work on the tested mine areas. The scheduled mine shut down period was removed from the model in order to plot continuous monthly development meter results from the simulation. The tested Shaft 2 access (S2A) area included two primary drives, and some remucks and crosscuts (shown in Figure 5-4). The average rate for Shaft 2 access was 5.9 meters per day per development crew. The single heading development rate was tested on the S2A top ramp. The same crew was designated to work only on this ramp after Shaft 2 access was completed. This area includes a single heading ramp and a few remucks (shown in Figure 5-4). The development rate was 3.4 meters per day per heading.

All the test results have been documented in Appendix II: 1. Test 1. Base Case Shaft 2 Access Dual Headings and Single **Heading Rate**.

Figure 5-4 Targeted Base Case Test Areas



5.3 Development Rates

5.3.1 Lateral Development Rates

Lateral development and mass excavation rates were tested in different areas. Most of the off-footprint area lateral development tests focus on two different development districts with two primary and two secondary headings available. The first development district was a dual heading S2A as two primary headings and the drifts of the offices connected to the S2A as two secondary headings (the relevant test can be found in Appendix II: Test 3). The second district was the service drift (SD) and main conveyor drift (MCD) as two primary headings and the Shaft 2 intake drifts (S2ID) as two secondary headings (the relevant tests can be found in Appendix II: Tests 5, 13, 14, 15 and 16). The lateral development tests on the footprint area focused on undercut, extraction and haulage drives. Four equal priority headings continued to advance during those tests (the relevant tests can be found in Appendix II: Tests 8, 12, and 17).

In order to acknowledge that a model over-optimizes equipment utilization and traffic interactions, and to ensure that rates remained in alignment with the benchmarked data, the modeled rates were reduced by 25%. This percentage was back-calculated based on data benchmarked from various caving operations' development rates which averaged at 6.2 m/day compared with the modeled rate of 8.2 m/d. This allows for a reasonable degrading of the modeled values to reflect the reality that four development headings are not always available.

The simulated lateral development rates are shown in Table 5-3. All profiles close to the 25 m^2 face area were grouped with a resulting standard "I" type as this group's benchmark rate; those close to 30 m^2 were grouped with the resulting "M" type off-footprint as this group's benchmark rate; and those close to 30 m^2 were grouped with the resulting "K" type off footprint as this group's benchmark rate. Smaller headings like 4.0mW x 4.0mH cutout and 4.5mW x 4.5mH ramps were assumed to have the same rate as the "I" type profile.

Some benchmark drifts were tested independently. Conveyor drifts (6.8mWx5.5mH) were 4.5 m/d off footprint and 3.2 m/d in the cave zone. Haulage drifts were mostly on footprints except for a very small portion (less than 100 m) connecting the Shaft 2 access drives. Thus, haulage drifts were assumed to be 100% in footprint areas and to have the same development rates of 3.2 m/d. In the footprint areas, undercuts and extraction drives were 6.0 and 4.3 m/d, respectively.

Table 5-3 Lateral Development Rates Summary

Tested Profile	Dimension (m)	Headings	Modeled Results (m/d)	Model Red. Factor	Off Cave Zone Multiple Heading (m/d)	On Cave Zone Multiple Heading (m/d)
Headings with $25m^2$ profile	5Wx5.5H	2 prim 2 sec	8.2	25%	6.2	4.3
Headings with $30m^2$ profile	5.8mWx5.8mH				5.1	3.6
Headings with $35m^2$ profile	6Wx7H	2 prim 2 sec	6.4	25%	4.8	3.3
Conveyor Drifts	6.8Wx5.5H	2 prim 2 sec	6.0	25%	4.5	3.2
Haulage Drifts	6.1Wx6H	4	4.2	25%	3.2	3.2
Ramps	4.5Wx4.5H				6.2	4.3
Extraction Panel Drifts	4.5Wx4.5H	4	5.7	25%		4.3
Extraction Perimeter Drifts	5Wx5.5H	4	4.3	25%		3.0
Cut-outs	4Wx4H				6.2	4.3
Undercut	4Wx4H	4	8.0	25%		6.0
Mass Excavation Two pass	$<65m^2$		131 m^3/day	25%	100 m^3/day	100 m^3/day
Mass Excavation Three pass	$>65m^2$		131 m^3/day*	25%	100 m^3/day	100 m^3/day

*assumed to have the same rate as a two pass excavation.

It can be found from Table 5-3 that the development rates of on footprint drifts were much slower than those of off footprint drifts of the same size. For example, the "I" type 5mWx5.5mH drift's advance rate was 6.2 m/d but the same profile on footprints was only advancing at 4.3 m/d. It was reduced by 30.6% due to a greater amount of ground support being required for “I” type headings on footprint areas. Development rates in cave zones also decreased by about 30% from the rates of off cave zone areas. This factor was obtained from tests conducted on conveyor and exhaust type drifts, see Appendix II: Tests 15 and 16. The development rates dropped by 37% and 23% for conveyor and exhaust type drifts, respectively, when the ground support on cave zones was increased to 100% Type 2.

5.3.2 Mass Excavation Rates

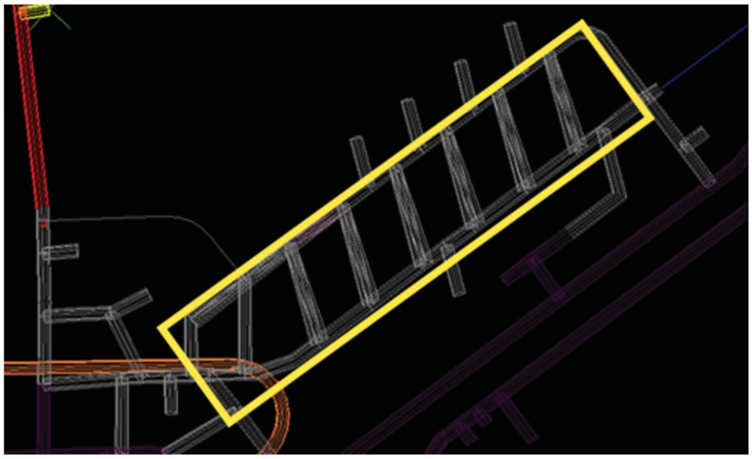
The mass excavation rate was the simulation focus on in two-pass 7 m wide by 8.7 m high (7.0mWx8.7mH) bays in the Shaft 1 workshop (S1W). The relevant test can be found in Appendix II: Test 9.

The 25% model reduction factor was utilized in the results of the 3-bay available case. However, 0 and 15% were used for 1- and 2-bay, cases respectively. After reviewing the ground support designs of the three-pass excavations, the rate was assumed to be the same as that of the two-pass headings. Therefore, the mass excavation rate of 100 m^3/day was suggested for use in both the two-pass excavation with a profile less than 65 m^2 and the three-pass excavation with a profile greater than 65m^2.

Table 5-4 Modeled Mass Excavation Rates after Model Reduction

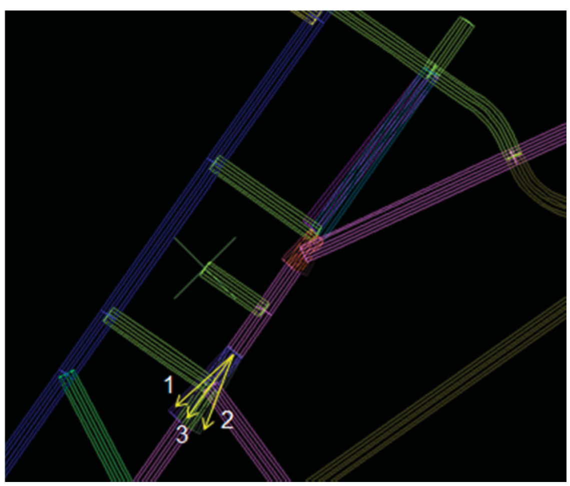
# of Bays Available	Excavation Rate (m^3/day)
Single (1)	50
Dual (2)	85
Triple (3)	100

Figure 5-5 Workshop Mass Excavation Simulation



All mass excavations during the PPD period except for Crusher 1 were modeled by the two or three pass benching in the MS3 Model. The benching followed a sequence of first, second and third pass. Because the MS3 model was only used for validating the schedules, the mass excavation rate was not a focus in this model.

Figure 5-6 Modeling of Mass Excavations in the MS3 Model



5.4 Sensitivity Tests

Several sensitivity tests were completed in the MS2 model to demonstrate the changes in different variables and the subsequent changes in the results. All results were calibrated to record the average development rates (in meters per day) which were obtained from total meters developed over the number of working days. The sensitivity was expressed as a percentage change from the base case using the formula: (experiment result - base case result) / base case result.

The sensitivity tests were focused on two base cases. The first was Test 1. Base Case Shaft 2 Access Dual Headings and Single **Heading Rate** (Appendix II: Test 1). Parameters such as mid-shift blasting (Appendix II: Test 2), change in availability, jumbo with the functions of drilling and bolting (Appendix II: Test 10), and reduction in drill length (Appendix II: Test 11) were tested and the results were compared with the base case development rate of 5.9 m/d. The sensitivities of these parameters are listed below:

- If the mid-shift blasting practice were implemented, the development rate would increase by 8.5% for dual headings and by 3% for single heading.
- If all equipment availability data was reduced by 5% (at a base case of 80%), then the overall development rate would slow by 3.5%.
- If using one jumbo to replace the face drill and bolter, then the development rate would decrease by 8.6% for dual headings and 5.9% for single heading.
- If using the 4 m round length (at a base case of 4.8 m), then the development rate would decrease by 4.7% for dual headings and 6.25% for a single heading.

The second sensitivity base case was that of Test 3. S2A Drives with Two **Secondary Headings** (Appendix II: Test 4). Parameters such as the distance between working area and shafts (Appendix II: Test 3), the number of allowed concurrent activities (Appendix II-Test 6), and number of available headings (Appendix II: Test 7) were tested and their results were compared with those of the base case 8.4 m/d:

- If the development areas were moved closer to Shaft 1 from MCD and SD to S2A resulting in a short tramming and muck handling distance (around 800 m or less), the development rate would increase by 3.4%.

- If the maximum concurrent activities were limited to two in this mine area, then the development rate would increase by 2.3%.
- If the number of available headings were increased to 5 (base case 4), there were seen to be no noticeable improvement.

5.5 Equipment Utilization and Ventilation Requirements

The outcome of equipment utilization over a three-year period was recorded from the simulation model. This was used to assist in determining the equipment ventilation capacity. Table 5-5 shows the outcome of a three-year period equipment utilization from the simulation model. The maximum value of each type of equipment in the "work%" category has been highlighted and selected as the benchmarked utilization for this type of equipment. For example, all LHDs were assumed to be 43% utilized.

Table 5-5 Outcome of Equipment Utilization over 3-year Period

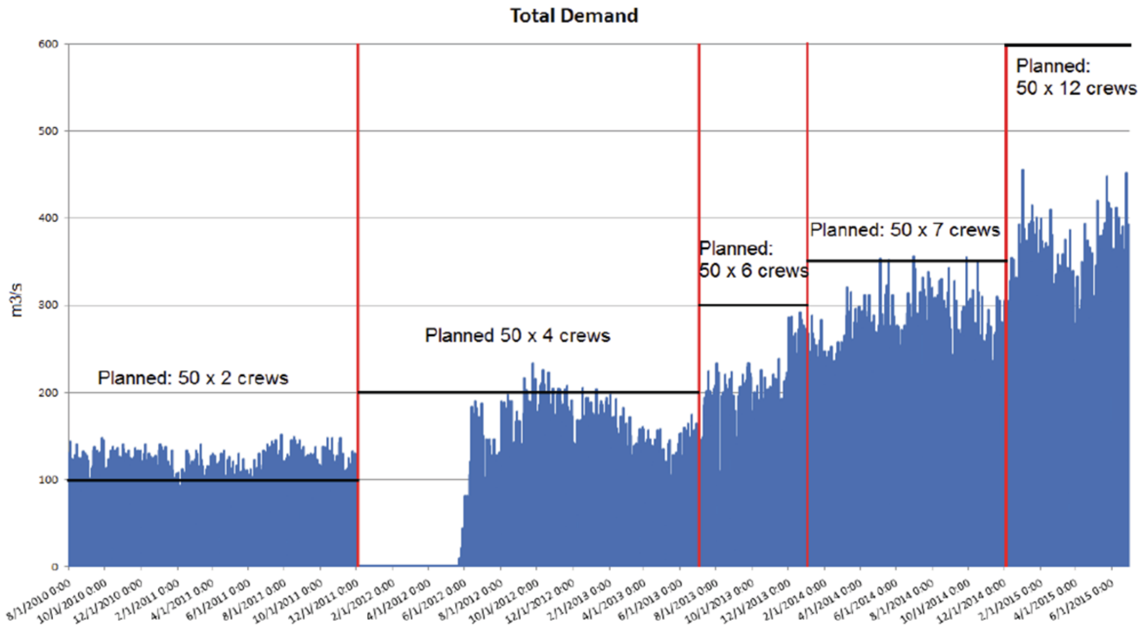
Equipment	Work (%)	Idle (%)	Meals &Shift Change (%)	Downtime (%)	PM (%)
LH307 (TORO6)-0	31.50%	53.30%	9.50%	4.50%	1.20%
LH410 (TORO7)-02	39.00%	39.60%	9.50%	10.30%	1.60%
LH517-01	**43.20%**	35.70%	9.50%	9.70%	1.90%
LH517-03	32.50%	50.70%	9.50%	6.00%	1.30%
LH517-04	18.90%	67.70%	9.50%	3.20%	0.70%
E2C-02	15.20%	60.00%	9.50%	2.80%	12.50%
M2C-01	**30.70%**	43.10%	9.50%	5.70%	11.00%
M2C-03	13.40%	67.50%	9.50%	1.50%	8.10%
M2C-04	13.70%	67.40%	9.50%	1.90%	7.60%
Charmec-01	**23.50%**	61.40%	9.50%	2.00%	3.60%
Charmec-02	7.80%	79.40%	9.50%	0.30%	3.00%
Charmec-03	7.00%	81.10%	9.50%	0.20%	2.30%
Charmec-04	10.40%	77.50%	9.50%	0.20%	2.40%
TH550-1	24.30%	60.90%	9.50%	4.50%	0.90%
TH550-2	**27.50%**	57.50%	9.50%	4.60%	1.00%
EJC530-1	25.20%	56.70%	9.50%	4.00%	4.60%
UTIMEC-0	8.20%	75.50%	9.50%	0.50%	4.00%
UTIMEC-1	**27.20%**	48.20%	9.50%	3.10%	4.80%
UTIMEC-2	18.80%	61.30%	9.50%	0.80%	4.40%
UTIMEC-3	19.40%	62.70%	9.50%	0.70%	4.00%
UTIMEC-4	7.90%	77.60%	9.50%	0.20%	3.20%
Sparymec-01	36.00%	42.30%	9.50%	6.70%	5.60%
Sparymec-02	25.00%	56.40%	9.50%	4.10%	5.00%
Sparymec-03	**36.80%**	45.50%	9.50%	3.60%	4.60%
Sparymec-04	21.90%	62.90%	9.50%	2.00%	3.70%
Boltec-02	30.90%	46.50%	9.50%	5.60%	7.50%
Boltec-03	38.50%	39.50%	9.50%	6.80%	5.70%
Boltec-04	26.30%	52.90%	9.50%	6.40%	4.90%
Axera 7-02	39.50%	31.30%	9.50%	9.90%	9.70%
DS310-01	**47.30%**	26.90%	9.50%	8.90%	7.40%
Cabletec-1_2	**15.20%**	64.60%	9.50%	0.90%	9.80%
Cabletec-3_4	6.10%	77.00%	9.50%	0.10%	7.40%

Cameron and Xiong (2011) imported the database from this study into a separate simulation package to model the real time diesel load within specific mining areas and to calculate the quantity of air required for ventilation modeling. Ventilation demand requirements at different locations in the mine were modeled for the five year PPD period. The results of total airflow requirements during the pre-production period are shown in

Figure 5-7. The black flat line in the graph represents the planned air quantity requirements and the blue wavy line represents the simulated air quantity required for the PPD period.

The model demonstrated that the total ventilation load per crew varied from 48 to 75 m^3/s when the crews were fully utilized and given four or more available headings. The initial assumption of 50 m^3/s crew approximated the quantity required for one development crew but in reality this quantity could vary significantly. As the number of crews increased and the quantity of available headings decreased, so did the required ventilation. This information allowed the mine planner to distribute air in ventilation models in accordance with the types of specific work each individual crew was conducting, and optimized its total distribution. It also provided another method to evaluate equipment performance and distribution within the mine (Wolgram, Li, & Scoble, 2012).

Figure 5-7 Total Airflow Requirement during the Five-year Simulation Period (Cameron & Xiong, 2011)



5.6 Equipment Optimization

This optimization test was focused on the MS3 model. The simulation period for the model was from January 1, 2011 to November 30, 2014. The mine shutdown period was scheduled as being from March 8, 2012 to August, 2012. After the MS3 model was completed, the base case results revealed that, during some periods, the equipment utilization of underground trucks and cable bolters was extremely high. Therefore, the underground muck handling and cable bolting capacity might potentially become bottlenecks for the underground excavation systems. This has been referred to in Appendix II: Test 18.

Three different scenarios of equipment buildups for development crews were tested and compared with the base case results.

- Adding a truck into the trucking fleet after mine shutdown (on March 8, 2012)
- Adding separate cable bolters for Crews 4 and 6 (crews 3,4,5,and 6 will have independent cable bolters)

- Adding a truck into the trucking fleet and adding two cable bolters for crews 4 and 6, respectively

The results were selected from June, 2013 to December, 2013 when the additional equipment demonstrated significant improvement in terms of development rates. In Figure 5-8, it can be found that development performance was positively correlated with the additional resources. Generally over the selected simulation period, the development rates of Scenario 1 increased by 7%, Scenario 2 by 6%, and Scenario 3 by 13% as compared with the MS3 base case. The utilization of underground trucks at peak times was also dropped to a reasonable level (at a 15% decrease).

Figure 5-8 Simulated Monthly Development Meters with Different Equipment Buildups

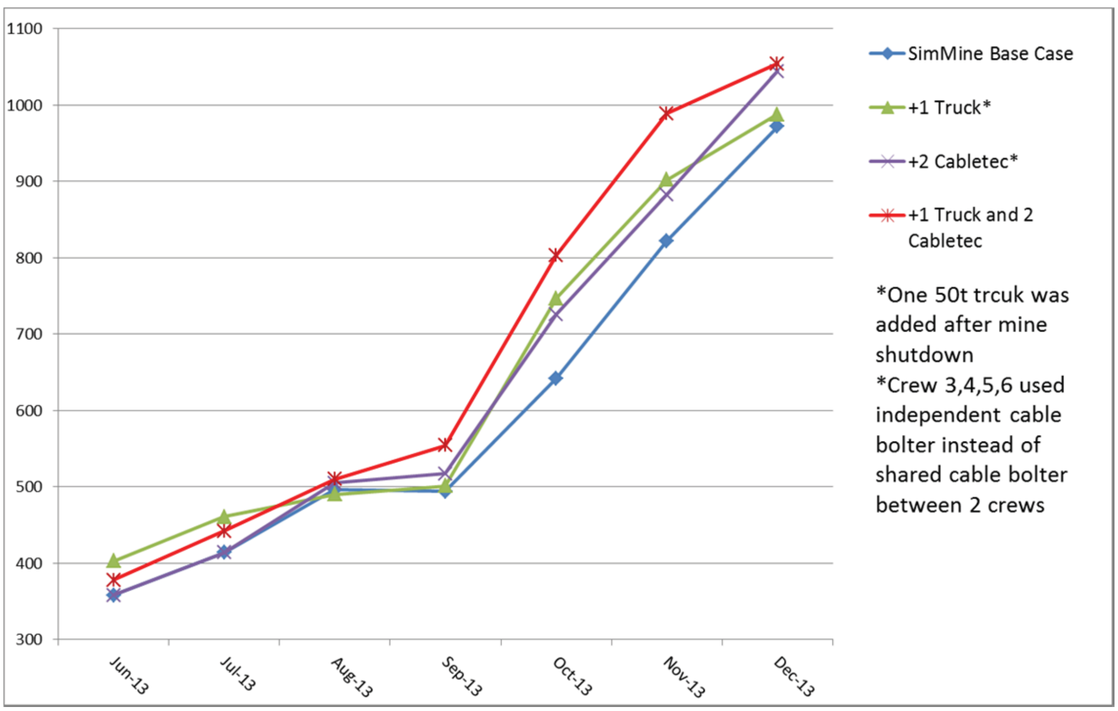


Table 5-6 Equipment Optimization Trails in Selected Simulation Period

m/d Date	SimMine Base Case	+1 Truck*	+2 Cabletec*	+1 Truck and 2 Cabletec
Jun-13	358	403	358	378
Jul-13	414	461	414	442
Aug-13	496	490	505	510
Sep-13	494	501	517	554
Oct-13	642	747	726	803
Nov-13	822	902	883	989
Dec-13	972	988	1,044	1,054
Total	4198	4492	4447	4730
Improve %		7%	6%	13%

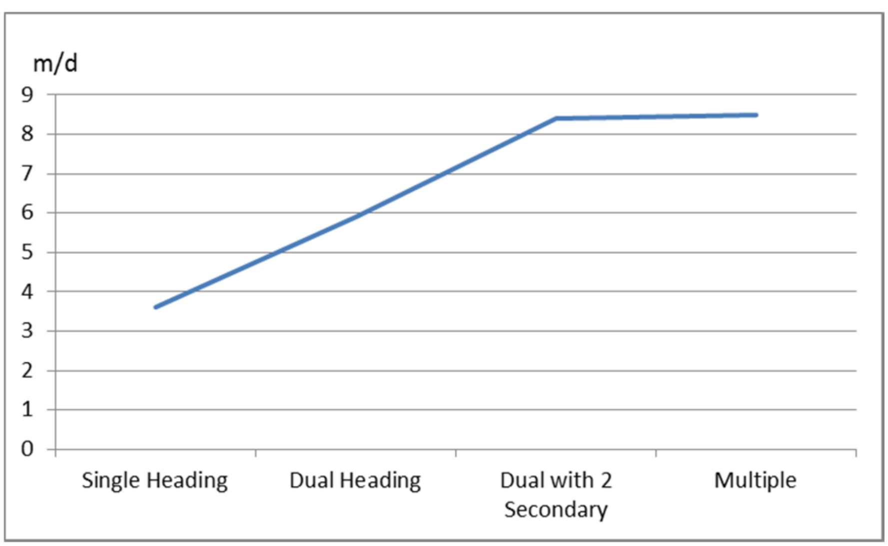
5.7 Discussion and Analysis

The first part of this section lists some simulation study findings related to the OT development case. The objectives are to discuss and analyze them, with the hope of speeding up the PPD. Most of the results and findings were discussed in the previous sections but the following additional findings are also very important:

- Mid-shift blasting may significantly improve the development rate because the charged headings need not await the blast which occurs only twice daily during shift changes. However, this situation must be examined carefully to ensure effective underground safety and ventilation before implementation. It is recommended that mid-shift blasting practice may be implemented in possible mine areas, particularly if certain critical path mining districts can establish isolated ventilation exhaust circuits.
- 400 tons of remuck was found to be unable to hold one round of muck generated from “K” type 6mWx7mH headings. This results in the the remuck having to be emptied more than once during each mucking cycle. A larger remuck was recommended for “K” type headings which would be at least 20 m long.

- Equipment utilizations during the ramp-ups of from seven to twelve crews was found to be very low since, during this period, there is an insufficient number of work headings for the 12 developmental crews. For example, one developmental crew may have only one or two available headings. The optimized number and distribution of these crews warrant further investigation in future work planning.
- The maximum development rate one crew could achieve was 8.4 m/d. The development rate was closely related to the number of available headings. When the number of available headings was increased, then the equipment utilization and development rates also rose. Figure 5-9 shows the relationship between the development rates and the number of active headings. It also demonstrated that the rates would not increase beyond situations with four available headings.
- Shift utilization and equipment availability play key roles in governing development rates. From the time study, it was be found that there were sometimes delays during shift change times and meal breaks. It is suggested that the net shift hours be maximized on the condition that safety still given the highest priority. Furthermore, the mobile equipment's availability should be kept no less than 85% over the long term.

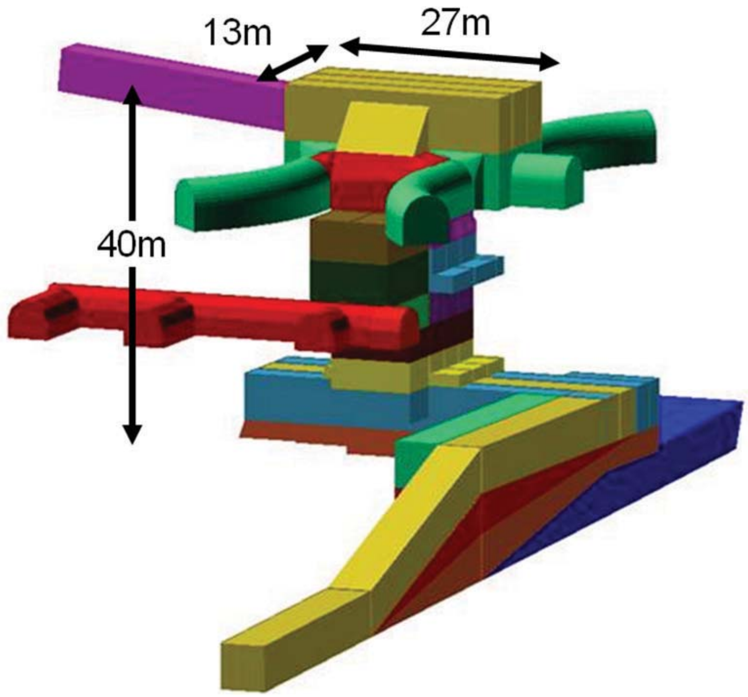
Figure 5-9 The Relationship between Available Headings and Development Rates



The second part of this section lists some problems that identified in the current simulation model. The problems are discussed below and future work is recommended to improve the model:

- The simulated single heading development rate of 3.5 m/d was slower than the benchmarked rate (4.5 m/d). It would be valuable to evaluate and compare these two rates in detail and identify the cause for the discrepancy.
- Traffic routes, parking spots and maintenance shop locations need to be defined in this discrete-event simulation model in order to better mimic the traffic of an underground mine. In the current model, the equipment would identify the closest route to travel, and when the equipment was idle or broke down it would park next to the closest remuck or crosscut or possibly even remain idle on the work face.
- Massive excavations like Crushers were not successfully modeled in the current scenario due to the complex geometry (as shown in Figure 5-10). In the MS2 model, it was represented by a single pass heading with large profile, possessing the same volumes as the crusher chambers. In the MS3 model, the crusher chambers are separated into 7 parts with each part simply following the manual first principle's schedule. However, other mass excavations are simulated by multiple cuts in the MS3 model. In the later study, it is recommended that an enhanced simulation program be developed capable of handling very complex geometry and logistics.
- Cable bolting and intersection cable bolting activities are not separated in the current model. Because intersection cable bolting would take a very long time to perform, particularly for larger intersections. One example is the cable bolting on a 4–way intersection, 6mWx7mH intersecting at 6.5mWx6mH, which may take from two to three days to complete. It is recommended that, in further simulation models, separate cable bolting activities be created for each intersection. The activity times would vary depending on the particular intersection cabling bolting's design pattern.

Figure 5-10 Crusher 1 Mass Excavation



6. Conclusion

This thesis has demonstrated how simulation techniques can significantly contribute to the quality of planning for large scale underground mines. Simulation was successfully applied to the mine planning process in a case study of a large underground panel caving mine. A new planning approach with the assistance of simulation increased the reliability and accuracy of development schedules in contributing to an important Feasibility Study.

The research evaluated the application of simulation to complex underground projects as well as assisting the field testing and development of a new commercial software package. The research clearly demonstrated the significant value of simulation in scheduling such complex mine development projects. Such software and 3-D animated graphics capabilities should prove to be the next generation, standard tool for complex underground development planning.

The simulation programs employed in this study necessitated around 150Mb computing capacity. The constructed model's database file consumed around 40Mb. The model included several thousand events, extremely complex logistics and a massive amount of mathematical calculations. Until recently this type of model could not have been executed with personal computer technology, but now such models can be fully implemented on contemporary desktop computer platforms, with a configuration of 8-core CPU, 16 GB memory, and dual graphic cards. Typical run times for the PPD model were around 30 minutes.

The main findings from this research are:

- During the data collection and preparation process it is critical to ensure the quality of the input data to avoid the "garbage in garbage out" phenomenon.
- During the model construction phase significant time was found to be necessary to devote to debugging such software, particularly through collaboration with its programmers. Developing effective verification of such a

complex simulation program as SimMine® was found to be a critical and time-consuming process.

- Mining equipment systems were effectively modeled in the simulation model. It was found that simulation had the potential to optimize the selection of an underground equipment fleet and determine the utilization and ventilation requirements for a particular development stage.
- Simulation effectiveness was found to be particularly powerful for short term underground excavation planning. The simpler and smaller the model scale then the more effective and reliable the simulation performance was found to be achieved.
- Simulation errors were found to be critical for a large model. These were found to be difficult to avoid in the model construction phase. Simulation results need to be reviewed and validated formally at regular intervals.

Some limitations of the simulation approach were evident from the research:

- SimMine® was not able to model excavations with complex geometry, such as the planned underground Oyu Tolgoi crusher chamber (13m wide x 27m long x 40m high as shown in Figure 5-10)
- The LHD loading logics need to be improved so that the machine can automatically select the dumping locations.
- Vertical development, e.g. shaft sinking and raise boring, were not considered in this study. However, it is recommended that they can be included in such models using separate equipment and crews. This would increase the complexity and size of the models and provide an environment that more closely represents the real underground activities and systems.
- The main traffic routes were not defined in the model. This may have under-estimated the traffic congestion issues in the system in situations with several pieces of mobile equipment
- Current simulation software packages still appear to be inefficient in handling very large models with complex logistics.

Underground mining comprises many systems and sub-systems that include machines, people and material handling. Simulation models should be useful in integrating many different aspects of these systems. Production capacity and equipment reliability are often measures that need to be simulated. There should also be opportunities to employ queuing systems to evaluate mine design factors, for example, congestion in mine traffic systems. Although there has been some work in this area, many such opportunities still exist in modern mine planning to apply simulation. The industry needs evolutionary planning tools and approaches. Currently there has not been such a tool available for the industry. There is some research and development going on to satisfy this need but more attention is required to advance the development of such new technology in planning large mining projects.

In the near future, simulation may be integrated with underground dispatch systems to manage equipment and people more effectively, this integration can create a tool to forecast shift by shift schedules and guide the underground activities and equipment movement. It also can be used to include cost estimation and to forecast cost overruns.

The future advancement of software capacities and computing power will likely lead to the presence of more easy-to-use, effective and reliable simulation software for the short and long term planning of all forms of mines.

The development of simulation techniques for mining would best involve the collaboration of software developers and mining companies who jointly create a software product that is specific to mining. The software would be in a fully interactive, three dimensional working environment with all the functions to import and change the entire mine designs. Equipment and people could be freely added to the model and assigned tasks. The next generation discrete-event simulation tools will more closely imitate the activities in the real mining world. Future tools will have the capability to consider all factors in real mining systems, e.g. relating to operators, equipment, throughput, and so on. Such versatile simulation tools should be significant for mining engineers in reducing costs and risks.

References

Agioutantis, Z. G., & Stratakis, A. (1998). Simulation of a continuous surface mining system using the Micro Saint visual simulation package. *Information Technologies in the Minerals Industry,* Athens, Greece. 85.

AMEC Engineering. (2011). *Drift design drawings: Mine development ground support drift. Rio Tinto Oyu Tolgoi underground feasibility study.* Unpublished manuscript, from Oyu Tolgio underground mine feasibility study.

Banks, J., Carson II, J. S., & Nelson, B. L. (1996). *Discrete-event system simulation* (2nd ed.). Upper Saddle River, N.J.: Prentice Hall.

Banks, J. (1998). Chapter 1: principles of simulation. *Handbook of simulation - principles, methodology, advances, applications, and practice* (2nd ed.) John Wiley & Sons.

Ben-Awuah, E., Kalantari, S., Pourrahimian, Y., & Askari-Nasab, H. (2010). Hierarchical mine production scheduling using discrete-event simulation. *International Journal of Mining and Mineral Engineering, 2*(2), 137-158.

Botha, J., Nichol, S., & Swarts, S. (2009). Rapid underground development optimisation at Cullinan Diamond Mine using computer simulation. *Diamonds – Source to use 2009,* Gaborone, Botswana.

Bradley, C. E., Taylor, S. G., & Gray, W. I. (1985). Sizing storage facilities for open pit coal mines. *IIE Trans, 17*(4), 320-326.

Brunner, D., Yazici, H. J., & Baiden, G. R. (1999). Simulating development in an underground hardrock mine. *1999 SME Annual Meeting,* Denver, Colorado. 4-11.

Cameron, C., & Xiong, W. (2011). *Oyu Tolgoi ventilation demand report.* Unpublished manuscript, from Rio Tinto Oyu Tolgoi Project Internal Report.

Carter, C. J., & Russell, F. M. (2000). Modelling and design of block caving at Bingham Cayon. *MassMin 2000,* Brisbane, Australia. 347-355.

Chanda, E. K. C. (1990). An application of integer programming and simulation to production planning for a stratiform ore body. *Mining Science and Technology, 11*(2), 165-172.

Dessureault, S., Scoble, M., & Rubio, E. (2000). Simulating block cave secondary breakage - an application of information and operations management tools in mass mining systems. *MassMin 2000,* Brisbane, Australia. 893-896.

Dyer, T. L., & Jacobsen, W. L. (2007). Simulation modeling validates load and haul requirements at Cortez gold mines. *Mining Engineering, 59*(1).

Elbrond, J. (1964). Capacity calculations at KLAB, Kiruna. *APCOM Proceedings,* Denver, Colorado. 683-690.

Freeport. (2012). *2011 form 10-K.* Retrieved 13th March, 2012, from http://www.fcx.com/ir/downloads/2011_Form_10-K.pdf

Frimpong, S., Changirwa, R., & Szymanski, J. (2003). Simulation of automated dump trucks for large scale surface mining operations. *International Journal of Surface Mining, Reclamation and Environment, 17*(3), 183-195.

Fytas, K., Hadjigeorgiou, J., & Collins, J. L. (1993). Production scheduling optimization in open pit mines. *Surface Mining Reclamation Environ, 7*(1), 1-9.

GijimaAst. (2010). *Mine2-4D overview.* Retrieved 10th November, 2011, from http://www.mine24d.com/overview_fs.html

Greberg, J., & Sundqvist, F. (2011). Simulation as a tool for mine planning. *Second International Future Mining Conference,* Melbourne, Australia. 273-278.

Hall, B. E. (2000). Simulation modeling of mining systems. *MassMin 2000,* Brisbane, Australia. 83-95.

Hall, R. A. (2000). *Reliability analysis and discrete event simulation as tools for mining equipment management.* (PhD, Queen's University).

Hall, R. A., & Daneshmend, L. K. (2003). Reliability and maintainability models for mobile underground haulage equipment. *CIM Bulletin, 96*(1072), 159-165.

Ivanhoe Mines Ltd. (2010). *2010 integrated development plan.* (Technical Report).

Ivanhoe Mines Ltd. (2012). *Oyu Tolgoi project IDOP technical report.* (Technical Report).

Kindle, K. I., & Mwansa, J.The role of simulation in infrastructure planning at the Grasberg Mine for the movement of men and materials. *MassMin 2012,* Sudbury, ON, Canada.

Kocsis, K. C., (2009). *New ventilation design criteria for underground metal mines based upon the "life-cycle" airflow demand schedule.* (PhD, University of British Columbia).

Labrecque, P., Newman, T., & Dudley, J. (2012). The use of ARENA® simulation to estimate drawpoint construction rate, production rate and costs for the Hugo North Lift 1 panel cave. *MassMin 2012,* Sudbury, Canada.

Laubscher, D. H. (1994). Cave mining – the state of the art. *Journal of the South African Institute of Mining and Metallurgy, 94*(10), 279-293.

Laubscher, D. (1997-2000). *"A practical manual on block Caving", for the international caving study*

Law, A. M., & McComas, M. G. (1992). How to select simulation software for manufacturing applications. *Industrial Engineering, 24*(7), 29-35.

Law, A. M., & Kelton, W. D. (2000). *Simulation modeling and analysis* (3rd ed.). Boston, USA: McGraw-Hill.

Luxford, J. (2000). Reflections of a mine scheduler. *MassMin 2000,* Brisbane, Australia. 119-126.

Lynch, A. J., & Morrison, R. D. (1999). Simulation in mineral processing history, present status and possibilities. *Journal of South African Institute of Mining and Metallurgy, 99*(6), 283-288.

McIntosh Engineering. (2009). *Hugo Dummett mine lift 1 prefeasibility study. p3-9.* (Pre-feasibility Study Report).

McNearny, R., & Nie, Z. (2000). Simulation of a conveyor belt network at an underground coal mine. *Mineral Resources Engineering, 9*(3), 343-355.

Moss, A. (2010). *UBC block caving course. Lecture 1 cave mining system- an introduction. derived from Brook Hunt data.* Unpublished manuscript.

Moss, A. (2011). *An introduction to block and panel caving. Presented at the 2011 global metals & mining conference.* Unpublished manuscript.

Mutagwaba, W., & Hudson, J. (1993). Use of object-oriented simulation model to assess operating and equipment options for underground mine transport system. *Transactions of the institution of mining and metallurgy. section A: Mining technology* (102nd ed., pp. 89-94) Maney Publishing.

Newman, A. M., Rubio, E., Caro, R., Weintraub, A., & Eurek, K. (2010). A review of operations research in mine planning. *Interfaces (Providence), 40*(3), 222-245.

Oraee, K., & Asi, B. (2004). Fuzzy model for truck allocation in surface mines. *Mine Planning Equipment Selection (MPES),* Wroclaw, Poland. , 585-591.

Panagiotou, G. N. (1999). Discrete mine system simulation in Europe. *International Journal of Surface Mining, Reclamation and Environment, 13*(2), 43-46.

Pegden, C. D., Shannon, R. E., & Sadowski, R. P. (1995). *Introduction to simulation using SIMAN* (2nd ed.). New York: McGraw-Hill.

Ross, S. M. (Ed.). (2006). *SIMULATION* (Fourth ed.). London, UK: Elsevier Academic Press.

Runciman, N. A. (1997). *Evaluation of underground mining equipment systems using discrete-event simulation with animation.* (M.A.Sc., Laurentian University of Sudbury (Canada)).

Salama, A. J., & Greberg, J. (2012). Optimization of truck-loader haulage system in an underground mine: A simulation approach using SimMine. *MassMin 2012,* Sudbury, ON, Canada.

Sargent, R. G. (2003). Verification and validation of simulation models. *Proceedings of the 37th Conference on Winter Simulation,* Syracuse, NY, USA. 130-143.

Schriber, T. J. (1991). *An introduction to simulation using GPSS/H*. New York: Wiley.

Sevim, H. (1987). Evaluation of underground coarse-coal slurry transportation systems by a simulation model. *Transactions of the AIME: Part A—Operational research* (280th ed., pp. 1817-1822)

SimMine®. (2010). *SimMine® Development package.* Retrieved 28th January, 2012, from http://www.simmine.com/Content/products/developement-package.aspx

Simsir, F., & Ozfirat, M. K. (2008). Determination of the most effective longwall equipment combination in longwall top coal caving (LTCC) method by simulation modeling. *International Journal of Rock Mechanics and Mining Sciences, 45*(6), 1015-1023.

Sinuhaji, A., Newman, T., & O’Connor, S. (2012). The development of lift 1 mine design at Oyu Tolgoi underground mine. *MassMin 2012,* Sudbury, ON, Canada.

Sturgula, J., & Li, Z. (1997). New developments in simulation technology and applications in the minerals industry. *International Journal of Surface Mining, Reclamation and Environment, 11*(4), 159-162.

Topuz, E., Breeds, C., Karmis, M., & Haycocks, C. (1982). Comparison of two underground haulage systems. *17th APCOM Symposium,* Littleton, CO, USA. 614-619.

Vagenas, N., Scoble, M., Corkal, T., & Baiden, G. (1998). Simulation of vertical retreat mining using Automod. *APCOM '98,* London, England. 91-100.

Vayenas, N., & Yuriy, G. (2005). Using Simul8 to model underground hard rock mining operations. *CIM Bulletin, 98*(1090), 76-81.

Triangular distribution. (n.d.). In Wikipedia. Retrieved April 10, 2012, from http://en.wikipedia.org/wiki/Triangular_distribution#Use_of_the_distribution

Wolgram, J. (2011). *Basis of development schedule feasibility study.* Unpublished manuscript, from Rio Tinto's Oyu Tolgio underground project.

Wolgram, J., Li, Z., & Scoble, M. (2012). Application of simulation techniques in Oyu Tolgoi underground development scheduling. *MassMin 2012,* Sudbury, Canada.

Yuriy, G., & Vayenas, N. (2008). Discrete-event simulation of mine equipment systems combined with a reliability assessment model based on genetic algorithms. *International Journal of Mining, Reclamation & Environment, 22*(1), 70-83.

Appendix I: Time and Motion Study

A1. Time Data

A1.1 Face Drilling

Drilling times were collected under “I”, “K” and “M” type headings. The profiles dimensions and descriptions can be referred to in Table 4-1. The other headings’ drilling times were based on the assumption:

Total drilling time = number of drill holes (N) x time required to drill one hole (Td) + time required to drill reaming holes (Tr).

The numbers of face drill holes for each heading were derived from the MS2 Feasibility Study’s drift design (AMEC Engineering, 2011). A few of these face hole numbers employed were based on the site’s actual practices which were slightly different from those of original design. For example, the “I” type 5mWx5.5mH site used 63 drill holes on its face rather than the 61 originally designed. For “K” type 6mWx7mH and “M” type face drilling, the site used 91 and 80 drill holes, respectively. Each face required two reaming holes which were shown as the lager pink circles on Figure A.1.1. After drilling the regular blast holes the drill bits needed to be replaced by a reamer. This time period has also been recorded in the time study. The drill length was planned at 5.2 meters but in the actual operation the average drill length was about 4.8 meters since 0.3 to 0.4 meters at the end of the drill holes was not be blasted. Therefore, 4.8 meters came to be recognized as the drill length used in simulations.

Drilling activity was performed mostly by the Atlas Copco M2C drill jumbo. Sandvik Axera 7, however, although occasionally involved in face drilling as well. Minimum, mean and maximum drilling times per hole are shown in Table A.1.1. The drilling time data follows a triangular distribution. The notation is TRI(1.5, 1.7, 1.7).

Figure A.1.1. Face Drill Plan Of the "I" Type 5mWx5.5mH Profile

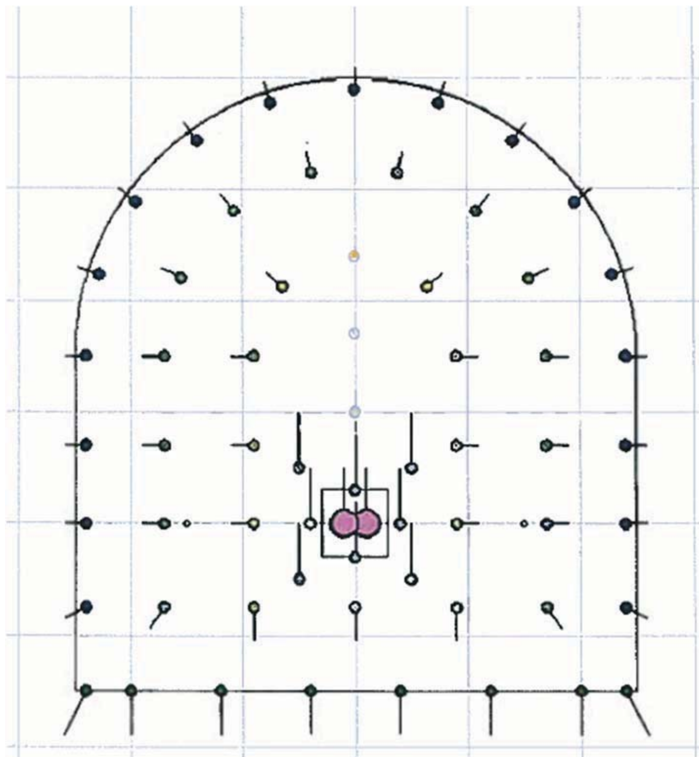


Table A.1.1. Unit Drilling and Reaming Time

Face Drill

Drilling time per hole (minute/hole)

Min	Avg	Max
1.5	1.7	1.9

Total reaming time (minute)

Min	Avg	Max
8.4	10.3	14.2

Table A.1.2. Face Drilling Times Used as Model Inputs

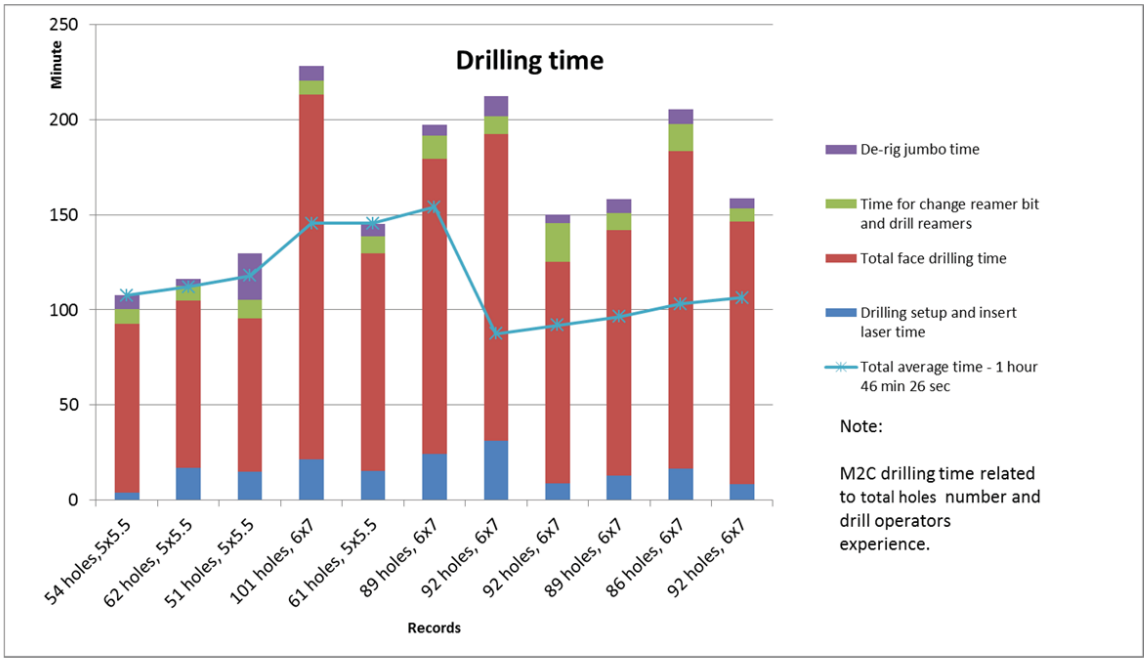
Drill Time Calculation (minute)				
Face	Min	Avg	Max	# of holes
Crusher1	512	551	587	324
L_D	83	95	109	50
L_E	100	114	130	61
L_F	92	106	121	56
L_H	100	114	130	61
L_I	103	117	134	63
L_J	122	140	159	76
L_K	145	165	187	91
L_M	128	146	166	80
L_O	188.4	214.3	242.2	120
L_P	248.4	282.3	318.2	160
L_Q	128.4	146.3	166.2	80
L_R	173.4	197.3	223.2	110
L_S	94	107	123	57
L_T	98	112	128	60
L_U	106	121	138	65
L_V	143	163	185	90
L_X	103	117	134	63
Top_of_conv	299	322	343	189
Transfer	397	427	455	251
(yellow)	assumptions			
(white)	site data based calculation			

*Yellow cells are assumptions since drill patterns are not available.

Table A.1.3. Drilling Activity Times Collected from the Site

Drill Preparation and Teardown (minute)											
1st Time	~~66~~	~~83~~	~~70~~	~~53~~							
2nd Time											
3rd Time	21.0	39.4	29.0	21.5	30.2	41.9	13.4	20.2	24.2	13.8	~~11.2~~
Drilling (minute/hole)											
1st Time	2.17	1.94	1.48								
2nd Time											
3rd Time	1.6	1.4	1.6	1.9	1.9	1.7	1.8	1.3	1.4	1.9	1.5
Reaming (minute/hole)											
1st Time	7.48	7.42	10.00								
2nd Time											
3rd Time	7.4	9.1	12.2	9.5	20.4	8.9	14.3	7.0			

Figure A.1.2. Breakdowns and Variances in Drilling Times on Different Profiles



A1.2 Charging

Charging activity was performed by two explosive loaders, Normet Charmec. Bulk emulsion explosives were used on-site. Operators stand on the charger's lift to manually charge the face. Bottom holes were often blocked by loose rocks or wet mud, and needed to be cleaned manually before charging could begin. If the jammed hole could not be cleaned, this hole would then be abandoned and a new hole needed to be re-drilled adjacent to the old. After connecting all wires and detonate cords, the operator would contact the shift boss for inspection. All such delays and waiting times such as cleaning jammed hole and re-drill time were included in the charging time study.

Total charging time includes the time required to clean the face, charge all of the holes, and tie the detonation cords. In order to scale the charging time and make it applicable to different profiles, charging time has been expressed in unit time, involving the number of minutes required to charge one hole. This was calculated by dividing the total charging time by the number of blast holes on the face. For those headings which did not have charging time data the following formula was employed to assume their charging times:

Total charging time = number of blast holes (N) x time required to charge one hole (Tc)

Charging times were collected from “I” 5mWx5.5mH, “M” 5.8mWx5.8mH, and “K” 6mWx7mH type profiles. The minimum, mean and maximum charging time per hole is shown in Table A.1.4., and is triangularly distributed TRI(1.1, 1.9, 1.5).

Table A.1.4. Unit Charging Time

Charging

Face Chargin time (min/hole)

Min	Avg	Max
1.1	1.5	1.9

Table A.1.5. Charging Times Used as Model Input

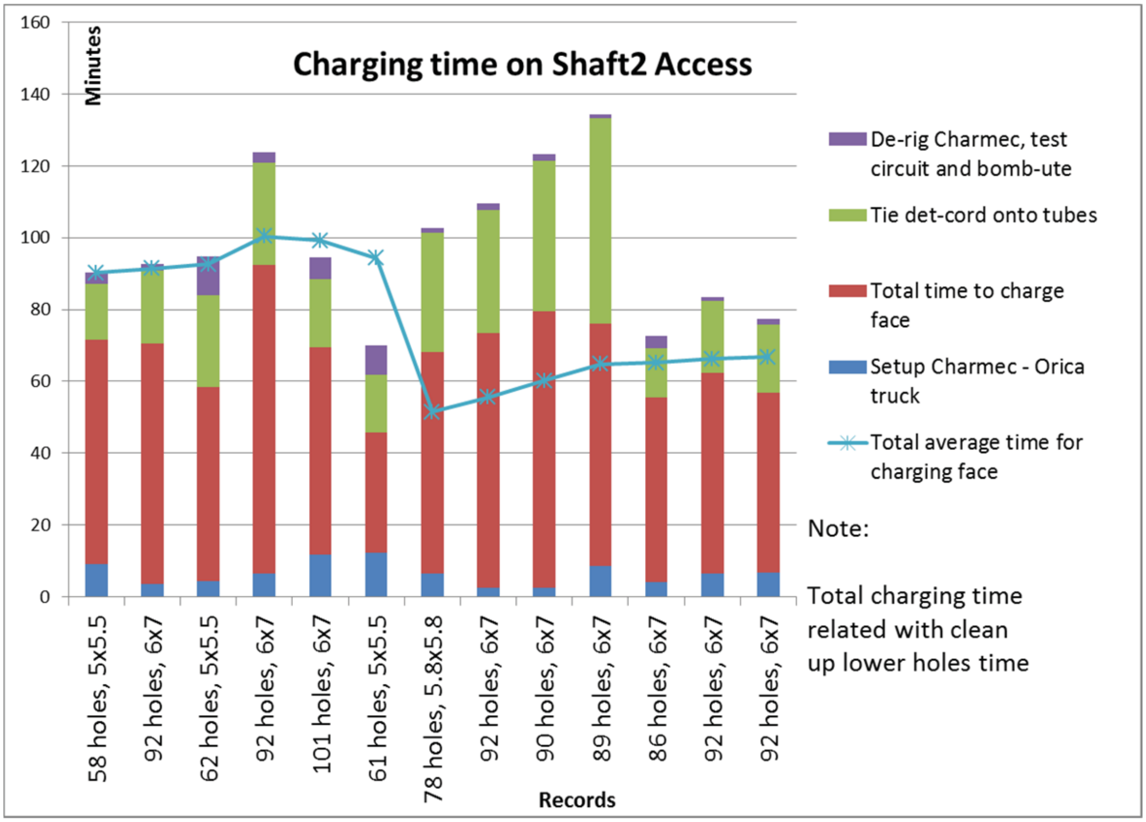
Face Chargin time (min/hole)

Face	Min	Avg	Max	# of holes
Crusher1	413	550	732	324
L_D	55	75	95	50
L_E	67	92	116	61
L_F	62	84	106	56
L_H	67	92	116	61
L_I	69	95	120	63
L_J	84	114	144	76
L_K	100	137	173	91
L_M	88	120	152	80
L_O	132	180	228	120
L_P	176	240	304	160
L_Q	88	120	152	80
L_R	121	165	209	110
L_S	63	86	108	57
L_T	66	90	114	60
L_U	72	98	124	65
L_V	99	135	171	90
L_X	69	95	120	63
Top_of_cc	184	245	326	144
Transfer	300	400	532	235
(yellow)	assumptions			
(white)	site data based calculation			

Table A.1.6. Charging Activity Times Collected from the Site

Charging Preparation and Teardown (minute)													
1st Time													
2nd Time	27	31	19	20	17	15							
3rd Time	12.2	5.4	15.3	9.3	17.9	20.8	7.7	4.6	4.5	9.7	7.7	7.5	8.4
Drilling (minute/hole)													
1st Time													
2nd Time													
3rd Time	1.3	1.4	0.9	1.5	2.0	1.3	1.8	2.0	2.1	1.1	1.3	1.2	1.6

Figure A.1.3. Breakdowns and Variances in Charging Times on Different Profiles



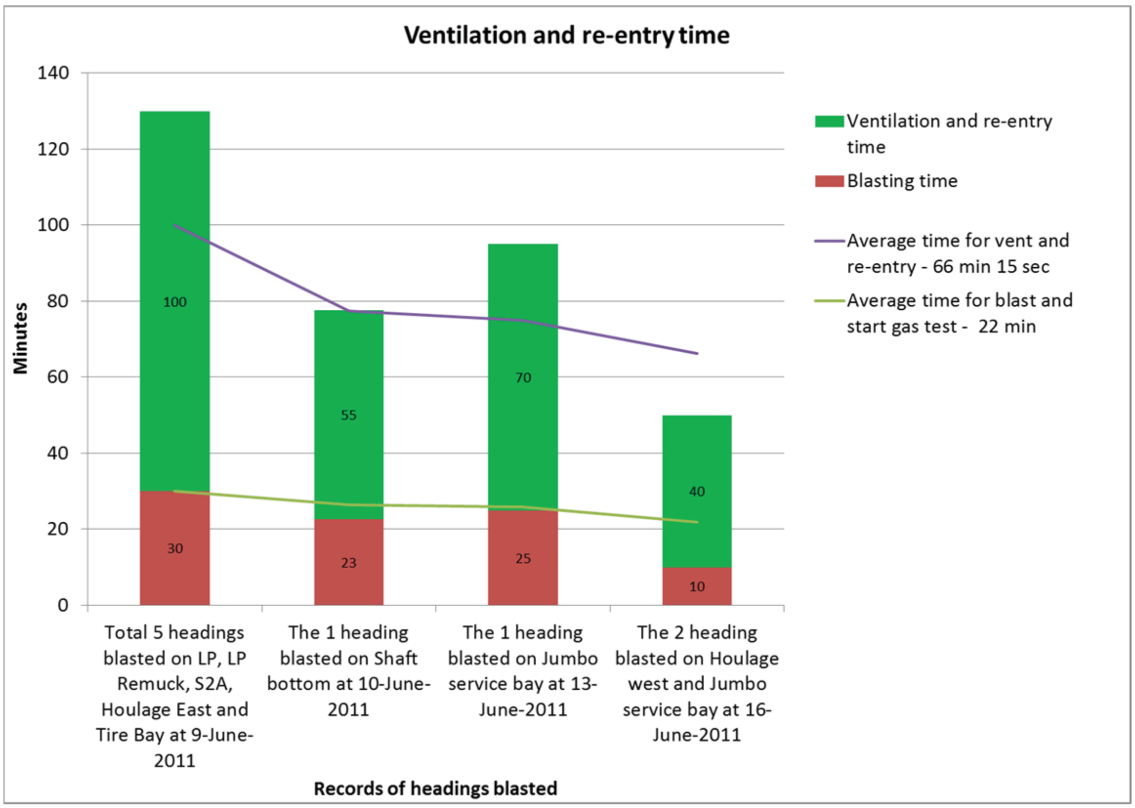
A1.3 Blasting

After the shift boss inspected the charged face, all personnel were required to return to the surface for shift changes. When the blast was initiated at 6:00 am and 6:00 pm, all underground work was required to cease. The first gas test was usually started approximately 1 hour after blasting. If the test results were good enough to allow work on certain headings then the next activity would be able to start in those areas. Conversely, if the CO content was still above limits, a second gas test would need to be performed after 20 to 30 minutes' time, and this process was repeated until the results met the requirements.

The ventilation re-entry time was related to the number of headings blasted. For example, if one or two headings were blasted the re-entry time would be about 60 minutes. However, if 5 headings were blasted simultaneously, re-entry would take more 100 minutes, because in the 5-heading-blasted case, only 4 headings would actually have been cleared after 100 minutes, and one more would still need to undergo the gas test.

Therefore, it was very difficult to accurately estimate the blasting reentry time for the simulation. The ventilation time was unrelated to the face profiles. Therefore, the mean value from 4 data points was required to represent the blasting ventilation reentry time. This was assumed to follow a triangular distribution of TRI(61, 87, 113).

Figure A.1.4. Blasting and Ventilation Re-entry Time



A1.4 Cleaning

Cleaning might to be performed before certain activities could take place. Some cases would require an LHD. The first type of cleaning was known as "face cleaning", and was utilized to clean the loose rocks from the floor following face drilling. The second type was called "cleaning", for installing meshes as required after scaling the face. An LHD was used to clean the floor of rocks during face support. This prepared the face for the next jumbo drill round. Other types of cleanings, such as cleaning the drilled face holes, would be completed manually. The time for such performance has already been included in the charging preparation and the actual charge processing times. For such non-equipment cleanings, no separate activities were created in the simulation model because these times would be included under other major activities.

The cleaning activity times were collected on “I” 5mWx5.5mH, “M” 5.8mWx5.8mH, and “K” 6mWx7mH type profiles. The face cleaning time was assumed to follow TRI(5.3, 22.2, 13.8). The cleaning time was assumed to follow TRI(8.5, 16.5, 12.5). The other profiles’ cleaning times were scaled based on face cross-sectional areas. Note that the face cleaning time was input as the total time in minutes, but the cleaning time was in minutes per meters advanced (minute/length in meters).

Table A.1.7. Face Cleaning Times as Model Input

Face Cleaning	Total Time (minute)			
	Min	Avg	Max	Face Area (m^2)
Crusher1	106	211	317	334
L_D	5	10	19	14
L_E	5	10	19	18
L_F	5	14	19	20
L_H	5	14	19	22
L_I	5	14	19	25
L_J	10	24	34	36
L_K	14	24	38	38
L_M	10	14	24	32
L_O	24	53	77	83
L_P	34	72	106	113
L_Q	10	19	29	33
L_R	19	38	58	62
L_S	10	19	29	32
L_T	5	14	19	22
L_U	14	24	38	44
L_V	10	14	24	28
Top_of_convey	38	72	110	114
Transfer	62	130	192	201
(yellow)	assumptions			
(white)	site data based calculation			

Table A.1.8. Cleaning Times as Model Input

Cleaning	Time per meter advance (minutes/length meter)			
	Min	Avg	Max	Face Area (m^2)
Crusher1	16	32	48	334
L_D	1	3	4	14
L_E	1	3	4	18
L_F	1	3	4	20
L_H	1	3	4	22
L_I	1	3	4	25
L_J	2	5	6	36
L_K	2	5	4	38
L_M	2	3	4	32
L_O	5	10	15	83
L_P	5	11	16	113
L_Q	2	4	6	33
L_R	4	8	12	62
L_S	2	4	6	32
L_T	1	3	4	22
L_U	2	5	6	44
L_V	2	3	4	28
Top_of_convey	8	16	24	114
Transfer	12	24	36	201
(yellow)	assumptions			
(white)	site data based calculation			

A1.5 Loading and Transport

Loading and transport activity times included the LHD's loading time, dumping time to trucks, and dumping time to loading bays; and the trucks' loading and dumping times. The loading and transport process at OT was investigated. After a round was blasted, an LHD came to the blasted face for muck. If a truck was available and near the face, it would drive to the face, and the LHD would load the truck. However, if no trucks were available or in the vicinity, the loaders would dump the muck into the nearest remuck. When the remuck was full, the loader and truck needed to empty it before it could collect the muck for the next round. The trucks always dumped the muck around the Shaft 1 loading pocket during the time study period. A 6 ton LHD was designated to work on the Shaft 1 loading pocket to move the development muck to the hoisting buckets in the Shaft 1.

The equipment's tramming time was calculated automatically by simulation model based on the distance over the speed. The mobile equipment's speeds decreased as the gradient increased, and increased if its bucket was empty. For example, an empty truck traveled from Shaft 1 to the Remuck #1. The distance was 300 meters (0 gradient). The speed for the empty and 0 gradient truck was 7 km/h. The model would calculate the travel time as 300 m/1000 km/7 km/h*3600 s/h = 154 seconds.

LHD's loading and dumping times were calculated based on the entire cycle time of loading-tramming-dumping, minus the tramming time. The average tramming speed of 2.2 m/s used in this calculation was based on the difference between two traveling distances over the difference between two cycle times. For example, the loading-tramming-dumping Cycle 1's tramming distance is 92 m, and it takes 145 seconds to complete this cycle. However, Cycle 2's tramming distance is 42 m, and it takes 101 seconds to complete this cycle. Therefore, the tramming speed would be (92-42)*2/(145-101) = 2.27 m/s.

Table A.1.9. Unit Loading and Dumping Time

Loading and Dumping

Loader				
	Min	Avg	Max	
Loading time	21	42	63	second/bucket
Dumping time	21	42	63	second/bucket

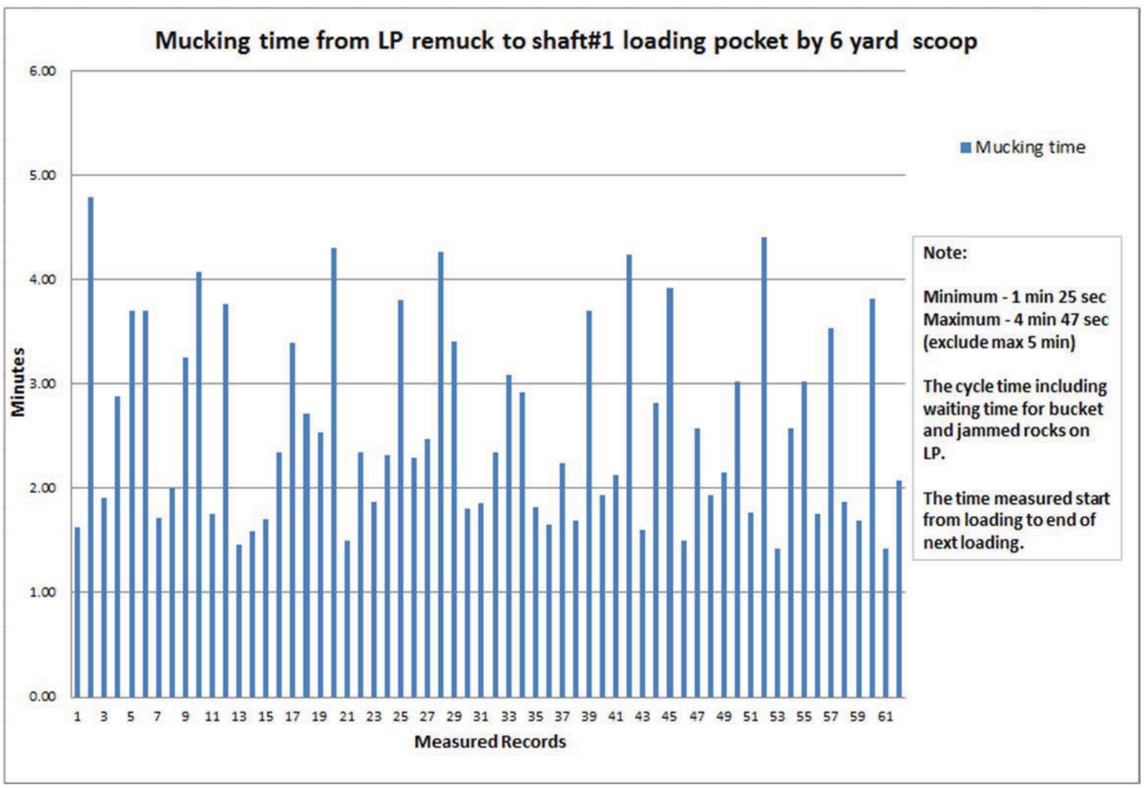
Truck				
	Min	Avg	Max	
Loading time	100	127	155	second/bucket
Dumping time	30	60	90	second/bucket

*The yellow cells are assumptions because no truck dumping time was collected.

Table A.1.10. Loading and Dumping Times as Model Input

	SimMine Inputs		
Loading and Transport Times	**Std**	**Mean**	**Units**
LHD Loading, tramming and dumping-LH517			
LH517 loading time	10.0	**41.5**	Second
LH517 tramming speed		**7.9**	km/hour
LH517 dumping time	0.0	**42.0**	Second
LHD Loading, tramming and dumping-Toro7			
Toro7 loading time			Second
Toro7 tramming speed			Meter/sec
Toro7 dumping time			Second
LHD Dumping to Truck-LH517			
Toro7			Second/bucket
LH517	21.0	**42.0**	Second/bucket

Figure A.1.5. Mucking Time Study for 6-yard Loader



A1.6 Shotcreting

The fiber reinforced shotcrete was used in the ground support of the OT underground development. The shotcrete was batched on the surface batch plant and transported by a slicken line to the underground mine. The transmixer (Spraymec) was used to load shotcrete from the slicken line near Shaft 1, and then delivered the shotcrete to the working faces. The shift boss ordered shotcrete from the surface batch plant before the transmixer's shotcrete tank was nearly empty, should it require additional loads. The loading time for a 5 m^3 transmixer was approximately 15 to 20 minutes. The tramming time of the transmixer was calculated automatically in the same model as the LHDs and trucks.

The Shotcreting time was expressed as minutes per cubic meter sprayed. This time was calculated by the duration of spraying 1 tank (5 m^3) of shotcrete, which was the time from the initiation of the spraying until the entire tank was finished. The spraying time was only a small proportion of the entire shotcrete cycle. The Waiting time for the agi-truck, however, could be much longer than the spraying time. The preparation and teardown times of the Spraymec were studied in detail as well.

The spraying time was assumed to follow TRI(3.9, 5.7, 4.8). The unit spraying time would be the same for all types of heading profiles because the same type of equipment (Spraymec) would be used for them. However, in the model, different profiles had their own inputs for the required amount of shotcrete. So the shotcreting time would be different for these headings. Usually, one round would require several loads of shotcrete. Based on the site data, the "I" 5mWx5.5mH profile needed 10 m^3 (2 loads), while "M" 5.8mWx5.8mH required 15 m^3 (3 loads), and "K" 6mWx7mH required 20 m^3 (4 loads) of shotcrete. The number of shotcrete loads needed on other types of profiles was assumed to be based on the face perimeter (floor excluded). The number of loads was then multiplied by 5 m 3/load to give the cubic meters needed per round.

After the profile was shotcreted, it would usually take up to 160 minutes for the shotcrete to set. However, some operators would not wait and would begin to bolt immediately after the bolter had been set up. The teardown time of Spraymec and setup time for bolting jumbo was often sufficient for the shotcrete to partially set.

Table A.1.11. Unit Shotcreting Time

Shotcreting time (minute/m^3)		
Min	Avg	Max
3.9	4.8	5.7

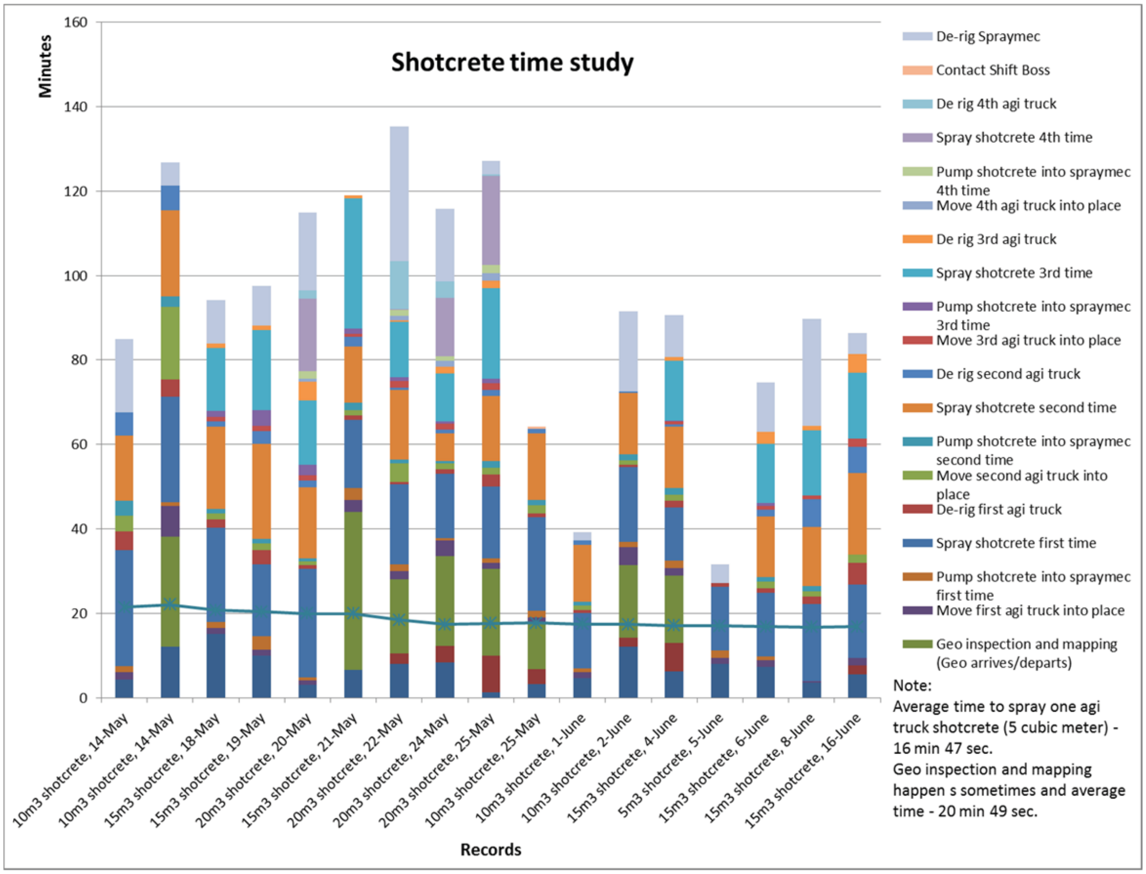
Table A.1.12. Shotcreting Times and Amount Requirements as Model Input

Shotcreting	Minute/m^3			m^3
Face	Min	Avg	Max	Shotcret/round
Crusher1	3.9	4.8	5.7	100.0
L_D	3.9	4.8	5.7	7.5
L_E	3.9	4.8	5.7	10
L_F	3.9	4.8	5.7	10
L_H	3.9	4.8	5.7	10
L_I	3.9	4.8	5.7	10
L_J	3.9	4.8	5.7	15
L_K	3.9	4.8	5.7	20
L_M	3.9	4.8	5.7	15
L_O	3.9	4.8	5.7	40
L_P	3.9	4.8	5.7	50
L_Q	3.9	4.8	5.7	15
L_R	3.9	4.8	5.7	40
L_S	3.9	4.8	5.7	20
L_T	3.9	4.8	5.7	10
L_U	3.9	4.8	5.7	10
L_V	3.9	4.8	5.7	15
L_X	3.9	4.8	5.7	12.5
Top_of_co	3.9	4.8	5.7	80.0
Transfer	3.9	4.8	5.7	80.0

Table A.1.13. Shotcreting Activity Time Collected from the Site

Shotcreting Preparation and Teardown (minute)																	
1st Time	36.0	29.0	33.0	26.0	~~4.0~~												
2nd Time	34.0	39.0	33.0	30.0	20.0	29.0	30.0	~~5.0~~									
3rd Time	19.5	21.3	31.0	25.4	39.9	25.5	16.3	12.4	19.0	28.8	10.5	21.8	17.6	~~3.2~~	~~6.7~~	~~6.6~~	~~4.4~~
Shotcreting (minute/m3)																	
1st Time																	
2nd Time	4.0	4.8	4.4	6.4	5.6	4.2	5.6	5.2	6.4	5.6	5.6	5.2	6.2				
3rd Time	6.3	4.6	5.0	3.8	3.3	4.7	3.5	3.8	3.7	4.1	4.9	5.2	4.7	4.7	3.3	4.1	

Figure A.1.6. Breakdowns and Variances in Shotcreting Times on Different Profiles



A1.7 Bolting

Bolting time was collected on “I”, “K” and “M” type profiles. It included bolt preparation, bolt drilling, resin pumping, and bolt inserting and tensioning and teardown. Time to install resin, and pin in and tension bolts was grouped together as bolt installation time in this time study. Thus, for example, the time for one hole would include the time required to drill it and install the bolts. Both DS310 and Axera 7 were involved in bolting activity and possessed different bolting times.

Bolt drilling and installation time were calculated as minutes per hole in order to scale and apply the activity time for headings that lacked site data. Two types of bolting holes were drilled at the OT. 3 m hole was used on “K” type profiles and 2.4 m hole was used on “I” and “M” type profiles. Axera 7’s bolt drilling time was assumed to follow TRI(1.65, 2.7, 2.0) for 2.4 m holes and TRI(1.75, 2.8, 2.1) for 3 m holes. DS310's bolt drilling time was assumed to follow TRI(2.8, 3.7, 3.1) for 2.4 m holes and TRI(3, 3.9, 3.3) for 3 m holes. Axera's bolt installing time was assumed to follow TRI(2.25, 4.2, 2.9), while DS310's bolt installing time was assumed to follow TRI(4.85, 9.2, 6.3). Note that the lower limit of each distribution was obtained by subtracting half the standard deviation from the mean value. The OT site suggested that it would not be possible to finish bolting a hole within the time of the mean value minus one standard deviation.

It can be seen that drilling time would increase (5%) were the hole's length increased from 2.4 to 3 m. The one boom DS310 bolting jumbo was much slower and less efficient than the two-boom Axera 7. In Table A.1.15. and Table A.1.16., the numbers of bolts and bolting hole length required in white cells were derived from the MS2 and MS3 heading profile designs. The ones in the yellow cells were assumptions.

Table A.1.14. Unit Bolting Time

Bolt Drilling				
2.4m hole	Min	Avg	Max	
Axera 7	1.65	2.00	2.70	min/hole
DS310	2.80	3.1	3.70	min/hole
3mhole	Min	Avg	Max	
Axera 7	1.75	2.10	2.80	min/hole
DS310	3.00	3.3	3.90	min/hole
Install Bolts				
2.4/3m ho	Min	Avg	Max	
Axera 7	2.25	2.90	4.20	min/hole
DS310	4.85	6.3	9.20	min/hole

Table A.1.15. Bolt Drilling Times and Bolts Amount Requirements as Model Input

Bolting Drilling

Axera 7 (minute/length meter)				# of bolts/m	
Face	Min	Avg	Max	Type 1	Type 2
Crusher1	248	300	405	150	150
L_D	17	20	27	10	10
L_E	20	24	32	12	12
L_F	15	18	24	9	9
L_H	14	17	22	8.3	8.3
L_I	17	20	28	10	12
L_J	23	28	38	14	14
L_K	15	18	24	8	17
L_M	26	32	43	16	16
L_O	50	60	81	30	30
L_P	58	70	95	35	35
L_Q	17	20	27	10	10
L_R	37	45	61	22.5	22.5
L_S	11	13	18	6	13
L_T	17	20	27	10	10
L_U	10	12	16	6	6
L_V	26	32	43	16	16
L_X	23	28	38	14	14
Top_of_co	145	176	238	88	88
Transfer	171	234	341	117	117

DS310 (minute/length meter)				# of bolts/m	
Face	Min	Avg	Max	Type 1	Type 2
Crusher1	450	495	585	150	150
L_D	30	33	39	10	10
L_E	36	40	47	12	12
L_F	27	30	35	9	9
L_H	25	27	32	8.3	8.3
L_I	31	34	40	10	12
L_J	42	46	55	14	14
L_K	27	29	35	8	17
L_M	48	53	62	16	16
L_O	90	99	117	30	30
L_P	105	116	137	35	35
L_Q	30	33	39	10	10
L_R	68	74	88	22.5	22.5
L_S	20	22	26	6	13
L_T	30	33	39	10	10
L_U	18	20	23	6	6
L_V	48	53	62	16	16
L_X	42	46	55	14	14
Top_of_co	264	290	343	88	88
Transfer	351	386	456	117	117

Table A.1.16. Bolt Installing Times and Bolts Amount Requirements as Model Input

Bolting

Axera 7 (minute/length meter)				# of bolts/m	
Face	Min	Avg	Max	Type 1	Type 2
Crusher1	338	435	630	150	150
L_D	23	29	42	10	10
L_E	27	35	50	12	12
L_F	20	26	38	9	9
L_H	19	24	35	8.3	8.3
L_I	23	30	43	10	12
L_J	32	41	59	14	14
L_K	20	26	37	8	17
L_M	36	46	67	16	16
L_O	68	87	126	30	30
L_P	79	102	147	35	35
L_Q	23	29	42	10	10
L_R	51	65	95	22.5	22.5
L_S	15	19	28	6	13
L_T	23	29	42	10	10
L_U	14	17	25	6	6
L_V	36	46	67	16	16
L_X	32	41	59	14	14
Top_of_co	198	255	370	88	88
Transfer	263	339	491	117	117

DS310 (minute/length meter)				# of bolts/m	
Face	Min	Avg	Max	Type 1	Type 2
Crusher1	728	945	1380	150	150
L_D	49	63	92	10	10
L_E	58	76	110	12	12
L_F	44	57	83	9	9
L_H	40	52	76	8.3	8.3
L_I	49	64	94	10	12
L_J	68	88	129	14	14
L_K	43	56	82	8	17
L_M	78	101	147	16	16
L_O	146	189	276	30	30
L_P	170	221	322	35	35
L_Q	49	63	92	10	10
L_R	109	142	207	22.5	22.5
L_S	32	42	62	6	13
L_T	49	63	92	10	10
L_U	29	38	55	6	6
L_V	78	101	147	16	16
L_X	68	88	129	14	14
Top_of_co	320	451	527	88	88
Transfer	425	598	700	117	117

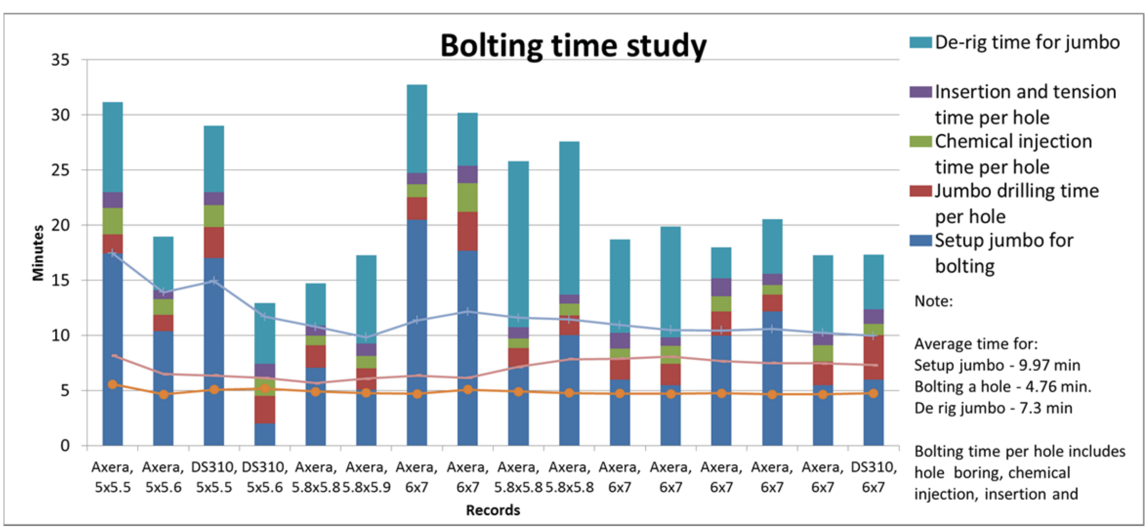
Table A.1.17. Bolting Activity Times Collected from the Site for Axera 7

Bolting Preparation and Teardown Axera 7	
1st Time	17.0 22.0 12.0 19.0 11.0 19.0 ~~9.0~~
2nd Time	26.0 ~~9.0~~
3rd Time	25.6 15.2 23.1 13.1 28.5 22.5 22.2 23.9 14.5 15.6 12.8 17.1 12.6 ~~7.5~~ ~~10.9~~ ~~11.0~~
Bolt Drilling (minute/2.4m hole) Axera 7	
1st Time	1.2 2.3 1.6 ~~1.1~~
2nd Time	
3rd Time	1.7 1.5 2.6 3.3
Bolt Drilling (minute/3m hole) Axera 7	
1st Time	1.6
2nd Time	1.6
3rd Time	2.0 1.9 1.9 2.2 1.5 2.1 4.0 3.6 2.0 2.0 1.7 1.8
Installing Bolt (minute/hole) Axera 7	
1st Time	5.1 2.3 3.7 6.6
2nd Time	2.4
3rd Time	2.2 4.2 2.4 2.4 3.0 1.9 2.5 3.9 2.3 1.8 2.2 1.9 1.8

Table A.1.18. Bolting Activity Times Collected from the Site for DS310

Bolting Preparation and Teardown Axera 7	
1st Time	17.0 22.0 12.0 19.0 11.0 19.0 ~~9.0~~
2nd Time	26.0 ~~9.0~~
3rd Time	25.6 15.2 23.1 13.1 28.5 22.5 22.2 23.9 14.5 15.6 12.8 17.1 12.6 ~~7.5~~ ~~10.9~~ ~~11.0~~
Bolt Drilling (minute/2.4m hole) Axera 7	
1st Time	1.2 2.3 1.6 ~~1.1~~
2nd Time	
3rd Time	1.7 1.5 2.6 3.3
Bolt Drilling (minute/3m hole) Axera 7	
1st Time	1.6
2nd Time	1.6
3rd Time	2.0 1.9 1.9 2.2 1.5 2.1 4.0 3.6 2.0 2.0 1.7 1.8
Installing Bolt (minute/hole) Axera 7	
1st Time	5.1 2.3 3.7 6.6
2nd Time	2.4
3rd Time	2.2 4.2 2.4 2.4 3.0 1.9 2.5 3.9 2.3 1.8 2.2 1.9 1.8
Bolt Drilling (minute/2.4m hole) DS310	
1st Time	3.0 3.4 2.7 3.8 2.6 3.0 4.6
2nd Time	
3rd Time	2.8 2.5
Installing Bolt (minute/hole) DS310	
1st Time	8.9 4.8 7.7 9.9 10.1 8.4 4.6
2nd Time	
3rd Time	3.1 2.9 2.4

Figure A.1.7. Breakdowns and Variances in Bolting Times on Different Profiles



A1.8 Face Support

The blasted face requires support for safety reasons. Face scaling (using the jumbo boom) and shotcreting, face meshing and the cleaning of scaled rocks were regarded as face support activities. Different ground conditions would need diverse approaches for face support. In good ground conditions, faces only needed to be scaled and the top regions shotcreted. Under normal ground conditions, the face needed to be entirely scaled and shotcreted. And for poor ground conditions, the face needed to be shotcreted and meshed. After scaling the face, if too many rocks were left on the floor, cleaning would need to be performed by an LHD. The cleaning was described in Section A1.4.

Face support times were collected from “I”, “M” and “K” profiles. The data points were so limited that it was not possible to determine the variation. The mean value could be used to represent the activity times. .

Table A.1.19. Face Supporting Time Collected From the Site

Face Support	5x5.5		5.8x5.8	6x7		
Scaling	17.0	17.0		17.0		
Screening	49.0	39.2	19.0	28.2		
Cleaning	12.5	20.1	14.5	52.0	18.0	6.3

A1.9 Cable Bolting

Cable bolting time included cable bolt preparation, cable bolt drilling, cement pumping, and cable bolt installation and teardown. This was performed by cable bolter (Cabletec). The time data was collected during the intersection cable bolting. The method for the cable bolting time analysis was similar to that of Section A1.7. It was expressed as minutes per hole. The total cable bolting cycle time for drifts and intersections was based on their cable bolting design patterns. Cable bolting activity consisted of regular drift and intersection cable bolting.

Drift cable bolting patterns were derived from the MS2 and MS3 heading profile designs. In Table A.1.21., the yellow cells were assumption 2 meter grid cable bolting patterns and the green cells were based on the MS2 and MS3 cable bolting designs.

The number of cable bolts required for each intersection was derived from the “I” type 5 m wide, “M” type 5.8 m wide and “K” type 6 m wide headings. For typical "T" intersections, where these headings intersected with 5 m wide remucks or cutouts, the intersections would require 7, 9 and 10 cables, respectively. Four-way intersections would require 9 central cables and 10 perimeter cables when 5 m wide drift intersected with 6 m wide drift. Thus, the average number of cables at intersections was 10 for 5 m wide headings and 15 for 5.8 m or 6 m wide headings.

6 m and 9 m long cables were used. However, only the cable bolting time was collected for 6 m long cables. The 9 m long cable bolts were assumed to have the same cable bolting time. They both followed TRI(29.8, 46.4, 38.1)

Table A.1.20. Unit Cable Bolting Time

	min/hole		
Cable Bolting Time	Min	Avg	Max
Drill	12.1	16.0	19.9
Pumping Cement	4.5	5.2	5.9
Install	13.2	16.9	20.6
Total bolting	29.8	38.1	46.4

Table A.1.21. Cable Bolting Times and Cable Amount Requirements as Model Input

Cable Bolting

Face	Min	Avg	Max	Type 1	Type 2	Weighted Avg	# of cables at intersection
Crusher1	626	800	941	17	17	17	40
L_D	12	15	19	0	2	0	0
L_E	105	134	163	2.4	8	4	0
L_F	113	145	176	2.5	9	4	0
L_H	30	38	38	0	0	0	10
L_I	30	38	38	0	0	0	10
L_J	89	114	127	1.5	1.5	1.5	15
L_K	80	103	113	0	6	1	15
L_M	164	210	243	4	4	4	15
L_O	161	206	236	3.6	3.6	3.6	18
L_P	161	206	236	3.6	3.6	3.6	18
L_Q	143	183	209	3.2	3.2	3.2	16
L_R	143	183	209	3.2	3.2	3.2	16
L_S	134	171	196	3	3	3	15
L_T	89	114	131	2	2	2	10
L_U	75	95	104	1	1	1	15
L_V	164	210	243	4	4	4	15
L_X	417	533	641	13	13	13	10
Top_of_conveyor	313	400	466	8	8	8	25
Transfer	417	533	625	11	11	11	30

Table A.1.22. Cable Bolt Requirements Assumptions for Profiles On Footprint Areas

Heading	Profile	of cables/roun		# of cables/m		Weighted Avg
		Type 1	Type 2	Type 1	Type 2	
Turnout	4.5mWx4.5mH	6	21	1.2	4.2	1.8
Extraction drive	4.5mWx4.5mH	15	50	3	10	4.4
Extraction and Turnout	4.5mWx4.5mH	12	40	2	8	3.5
Undercut drive	4.1mWx4.1mH	0	10	0	2	0.4
Extraction perimeter	5mWx5.5mH	65	65	13	13	13
Undercut perimeter	5mWx5.5mH	65	65	13	13	13

Table A.1.23. Cable Bolting Times Collected From the Site

Cable Bolting	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	29	30
Setup	15.2	13.0																				
Drill	13.3	15.5	20.5	16.7	16.7	12.6	18.5	17.7	20.0	15.5	11.5	10.8	10.7	11.0	14.2	14.8	13.4	20.5	20.3	25.5	~~7.0~~	~~4.5~~
Pumping Cement	6.1	5.8	4.1	4.9	5.2																	
Install	16.3	10.3	21.5	18.8	17.6																	
Idle	42.0																					

A1.10 Non-equipment Activities

After the face was blasted, dust would be a problem for operators and loaders when beginning mucking. Therefore, the muckpile and the road were first washed with water to settle the dust. The washing activity time was assumed to follow TRI(12.5, 34.5, 23.5).

Hydroscaling was used to scale the backs and walls which required the later applic ation of shotcrete. This scaling was completed with high pressure water. It was assumed to follow TRI(3.9, 18.5, 11.2). This time distribution was not used because no separate activity was create for its simulation. The mean value of the hydroscaling time, however, was included in the shotcreting's preparation and teardown time to represent this activity.

After the drift was hydroscaled, the geotechnical group would arrive to inspect the tunnel and map the geotechnical conditions before the shotcreting was performed. This activity's time was assumed to follow TRI(13.3, 28.3, 20.8).

The M2C drill jumbo required a grade line before the face could be drilled. Usually a laser grade line installed in the tunnels would be used by the M2C. If the laser guile line were not available, two surveyors would arrive at the face to draw the grade line. One of them would measure the grade line while the other would spray the lines in the tunnel. This activity's time was assumed to follow TRI(9.6, 20.4, 15).

Table A.1.24. Non-equipment Activity Times Collected from the Site

Wash Down Face (minute)	
1st Time	40.0
2nd Time	21.0 18.0 11.0 20.0
3rd Time	11.0 40.1 31.1 2.5 22.5 17.0 25.1 26.7 30.3 20.0 22.9 8.4 36.5 42.7
Hydroscaling (minute)	
1st Time	20.0 12.0 30.0 11.0 6.0
2nd Time	15.0 12.0 20.0 18.0 7.0 11.0
3rd Time	2.2 3.7 8.8 4.0 2.5 6.6
Survey Time for M2C Grade Line	
1st Time	
2nd Time	
3rd Time	22.1 22.3 14.8 17.5 11.7 8.0 7.2 16.2
Geotechnical inspection	
1st Time	
2nd Time	
3rd Time	10.8 17.2 37.3 17.5 21.2 20.6 16.0 26.0

A2. Activity Time Analysis

A2.1 Quality of Data

The quality of the collected site data has been improved from the first time collection to the third. After the data was received from the site, it was then carefully analyzed and verified. Requirements for additional information and recommendations on changes would be sent to the site for future collection. During the third time collection, the site made a formatted time study template for collecting each of the activity times. During the data preparation process, several discussions were conducted with the engineers who had actually collected the data. Communication was never stopped between the study team and the site engineers. Final processed time data to be used in the simulation was reviewed by site engineers. Therefore, the activity times are reliable.

A2.2 Activity Times Variance

Activity times are probability distributed. In order to simplify and calibrate the variances, triangular distribution has been selected to represent the data set. The notation for triangular distribution is TRI(a, c, b) where a is the lower limit, b is the upper limit and c is the mode (mean). The lower and upper limits were calculated based on the mean value plus or minus one standard deviation. So $a=b-\sigma$, $c=b+\sigma$. The lower limit of the bolting time, however, was only calculated by the mean value minus half the standard deviation. Therefore, the input data's probability density function should resemble Figure 3-1.

A2.3 Delays and Waiting Times

Delays and waiting times were studied and included in all activities. The time duration of each activity was recorded from the initiation of this activity and continued until its completion. For example, drilling started at 9:30 am and finished at 12:00 pm, so the total duration of the drilling would be 150 minutes. These 150 minutes included all delays, regardless of their reason, and the waiting time for resources.

Typical delays and waiting times for each activity have been identified and discussed with the site engineering department which had collected this time data. They were summarized and listed in the table below.

Table A.2.1. Delay and Wait Time Analysis

#	Activity	Descriptions
1	Face Drilling	Jammed drill holes: water and mud in bottom holes (50% probability). Pumping water: needed 1-2 hours. (in the shaft bottom 100% occurred. The main access drive also experienced water problems). Changed drill bits: needed 5 mins. Water pressure was not enough: two booms could not work together. The contractor's exploration drilling had to be stopped so tht more water pressure could be used for the face drilling (30 mins). Electrical problem: the power was down. The drill could not be pulled out and was stuck in a hole. One boom was broken.
2	Face cleaning	Cleaning the hole took most of the time since it was hard work.
3	Charging	Water was on the floor: Had to wait while pumping out the water (30 mins) The explosive loader's capacity was only sufficient for charging 1-2 faces per load. Went to the magazine, charged the tank and returned to the face. Ground conditions were bad: had to clean all the holes. Asked the shift boss to inspect them. When required, the holes had to re-drilled (this did not happen) or abandoned. Too much rock was on the floor after drilling: needed to call and wait for a scoop to clean it. The scoop should have been called by the drilling group but they sometimes forgot and it was left to the charging group.
4	Blasting	Multiple headings blasted: would need much time for a ventilation re-entry. Usually one heading took 15 mins. Blasting: all work ceased and no one was allowed underground. Waited in the shaft station until the fume was cleared.
5	Washing	The hosing time needed to be extended. Attached many hoses to obtain water. The LHD cleaned the fly rock from the road (very fast).
6	Loading and transport	Had to remove the water pump which was left behind by the washing operators (it should have been removed after the washing) Much time was required to empty the remuck. Long time to wait for the dump truck. Therefore, the LHD had to bring one scoop and dump to the shaft.
7	Geotechnical inspection	Data not available.
8	Shotcreting	The hose jammed (20-60 min); used a hammer to knock off the set shotcrete.

		Many hoses needed to be connected to obtain water. Not enough transmixers were available to deliver the shotcrete. Usually Spraymec needs 15 mins to wait for the shotcrete (if one transmixer were working, had to wait 30 mins). 20 mins to load a transmixer. The shift boss ordered shotcrete. Positioning the truck and attaching the hose for Spraymec. Scaling the walls and back. Small rocks could jam the hose. Performed shotcrete quality and slum tests. If the shotcrete were too wet, it would flow; if too dry, it would jam the hose. Washing the shotcrete machines after work. Slicken line jammed.
9	Meshing& installing split set	Axera 7 was used. 49 min were spent on S2A-VR3. If scaled faces and too much rock fell on the floor, needed to clean it before charging. Meshed face top, scaled and then charged. First charged top holes, then lower holes.
10	Bolting	DS310 only has 1 boom. Poor rock quality caused holes to jam. Re-drill needed on the hole or on nearby areas. Resin had been stored in a hot place and had expired; needed new resin. Water pressure down: waited water for 30 mins Checked oil level and added more oil for 5 min. Had only one water pipe for shotcrete and bolt but both of them needed water. Rock fallen on the floor required cleaning by scoop.
11	Strapping	Data not available.
12	Cable bolting	Maintenance performed a training check on the power level and the engine. On the date when the data was collected, Cabletec had only been used for training and testing.
13	Face scaling	Axera 7 scaled the face and meshed (45 mins), rocks needed to be cleaned.
14	Survey	Surveyors brought the wrong survey equipment; called the surface and waited for the correct items. Brought the wrong design file (5-10%). If manual spraying required, needed to call lifting equipment. One operator was sketching while the other was watching

A2.4 The Basis of Activity Time Calculations and Assumptions

The activity time assumptions for face drilling and charging were based on the number of drill holes on a face and time required to complete one hole. For bolting and cable bolting, the assumptions were based on the bolting pattern and the unit times for the completion of one hole. All assumptions are listed on Table A.2.2. Shotcrete quantity requires that different varieties of headings are based on the site data of “I”, “M” and “K” type headings. The details of other profiles which did not have site data were assumed by their face perimeters (floors excluded). The average shotcrete consumption per meter of face perimeter for a 75 mm thickness shotcrete application was calculated and then this number was used to scale other profiles by simply multiplying the face perimeters.

The activity time assumptions for drilling and charging were based on the formula:

- Total drilling time = the number of drill holes on the face x the time required to drill one hole + the time required to the drill reamer holes
- Total charging time = the number of drill holes on the face x time required to charge one hole

The activity time assumptions for boltings and cable boltings were based on the formula:

- Total bolt drilling time = the number of drill holes per meter x the time required to drill one hole. For example, on a 1.2 x 1.2 grid bolt pattern, the “I” type heading, the bolt drilling time using Axera 7 is 2 mins/hole * 12 holes/ring/1.2 m ring spacing = 20 minutes/meters advanced.
- Total bolt installing time = the number of drill holes per meter x the time required to install one hole. For example, on a 1.2 x 1.2 grid bolt pattern, the “I” type heading, the bolt drilling time using Axera 7 is 2.9 min/hole * 12 holes/ring/1.2 m ring spacing = 30 minutes/meters advanced.
- Total cable bolting time = the number of cables per meter x the time required to install one cable. For example, on a 2 x 2 grid cable, using the bolt pattern of “D” type heading, the total cable bolting time using Cabletec is 38.1 min/hole * 4holes/ring/2 m ring spacing = 76.2 minutes/meters advanced

Table A.2.2. The Number of Face Holes, Bolt and Cable Bolts on Different Profiles

Face Profile	# of face holes	# of bolt/round		# of bolt/m		# of cable/round		# of cable/m	
		Type 1	Type 2	Type 1	Type 2	Type 1	Type 2	Type 1	Type 2
D	50	50	50	10	10	0.0	10.0	0	2
E	61	60	60	12	12	13	55	2.5	11
F	56	60	60	12	12	13	45	2.5	9
H	61	42	42	8	8	0	0	0	0
I	63	50	60	10	12	0	0	0	0
J	76	70	70	14	14	7.5	7.5	1.5	1.5
K	91	40	85	8	17	0	30	0	6
M	80	80	80	16	16	20	20	4	4
O	120	150	150	30	30	0	18	0	18
P	160	175	175	35	35	0	18	0	18
Q	80	50	50	10	10	0	16	0	16
R	110	113	113	23	23	0	16	0	16
S	57	30	65	6	13	15	15	3	3
T	60	50	50	10	10	10	10	2	2
U	95	30	30	6	6	5	5	1	1
V	75	80	80	16	16	20	20	4	4
X	63	70	70	14	14	65	65	13	13

Assumption

Site Data and calculation based on MS3 or MS2 design

Table A.2.3. Shotcrete Amount Requirements on Different Profiles

Face Profile	Peri. (m)	1st layer shotcrete m^3/ round	1st layer shotcrete m^3/m	2nd layer shotcrete m^3/ round	2nd layer shotcrete m^3/m
D	10.3	7.5	1.5	7.5	1.5
E	11.6	8	2	10	2
F	11.0	8	2	10	2
H	12.8	10	2	10	2
I	13.8	10	2	10	2
J	17.1	15	3	15	3
K	17.4	20	4	20	4
M	15.2	15	3	15	3
O	23.8	40	8	40	8
P	28.9	50	10	50	10
Q	15.7	15	3	15	3
R	22.6	40	8	40	8
S	14.0	20	4	20	4
T	15.3	10	2	10	2
U	14.4	20	4	10	2
V	17.0	15	3	15	3
X	13.8	10.0	2.0	12.5	2.5

Assumption

Site Data and calculation based on MS3 or MS2 design

A3. Equipment Maintenance and Downtime Study

The equipment maintenance schedule was received from the site's maintenance group. The planned maintenance schedule and shift by shift records were provided from January to June, 2011 for the entire underground mobile equipment fleet. The time length for all services was recorded, including for equipment washing, oil and battery changing, replacing parts and repairs.

Because not all of the underground mobile equipment would be used in the development simulation model, only the major development equipment's data was studied. The major development equipment include: the drill jumbo, LHD, underground truck, bolter, charger, cable bolter, shotcreter and transmixer. The maintenance was broken down by corrective maintenance (short–term, planned maintenance), preventative maintenance and breakdown. However the simulation model only requires the input for preventative maintenance and availability, so the corrective maintenance was included in the breakdown catalogue.

A3.1 Preventive Maintenance

The Site provided a planned schedule for the preventative maintenance program, as illustrated in Table A.3.1. (all numbers are in hours). However, after being reconciled with the daily maintenance records, some of the planned maintenance times were modified to those of the actual times performed (Table A.3.2.).

Table A.3.1. Planned Underground Mobile Fleet Preventative Maintenance

#	Equip. Ref.	Equipment name	Weekly/Bi weekly	250 HR	500 HR	1000 HR	2000 HR
1	**UL-0001**	UNDERGROUND LOADER #1 TORO 7		8	12	12	12
2	**UL-0002**	UNDERGROUND LOADER #2 TORO 6		8	12	12	12
3	**UL-0003**	UNDERGROUND LOADER #3 LH-517	6	8	12	12	12
4	**JU-0001**	JUMBO DRILL #1	10	12	12	12	12
5	**CMJU-0001**	JUMBO #1 COMPRESSOR			4	4	4
6	**BMJU-0001L**	JUMBO DRILL #1 LEFT BOOM"			4	4	4
7	**BMJU-0001R**	JUMBO DRILL #1 RIGHT BOOM			4	4	4
8	**JU-0002**	JUMBO DRILL #2	10	12	12	12	12
9	**CMJU-0002**	JUMBO #2 COMPRESSOR			4	4	4
10	**BMJU-0002L**	JUMBO BOOMER LEFT BOOM			4	4	4
11	**BMJU-0002R**	JUMBO BOOMER RIGHT BOOM			4	4	4
12	**JB-0001**	JUMBO BOLTER #1	10	12	12	12	12
13	**CMJB-0001**	JUMBO BOLTER #1 COMPRESSOR			4	4	4
14	**DR-78062026**	BOLTER ROCK DRILL			4	4	4
	JB-0002	CABLETEC	10	12	12	12	12
	CMJB-0002	CABLETEC COMPRESSOR			4	4	4
	BMJB-0002	CABLETEC BOOM			4	4	4
15	**UT-0001**	UNDERGROUND TRUCK	6	8	12	12	12
16	**UT-0002**	UNDERGROUND TRUCK TH-550	6	8	12	12	12
17	**US-0001**	UNDERGROUND SPRAYMEC	6	10	12	12	12
18	**CMUS-0001**	SPRAYMEC COMPRESSOR			3	3	3
19	**PMUS-0001**	SPRAYMEC CONCRETE PUMP			3	3	3
20	**US-0002**	UNDERGROUND SPRAYMEC	6	10	12	12	12
21	**CMUS-0002**	SPRAYMEC COMPRESSOR			3	3	3
22	**PMUS-0002**	SPRAYMEC CONCRETE PUMP			3	3	3
23	**UM-0001**	UNDERGROUND MIXER	6	8	12	12	12
24	**UM-0002**	UNDERGROUND MIXER	6	8	12	12	12
25	**UM-0003**	UNDERGROUND MIXER	6	8	12	12	12
26	**UP-0001**	UNDERGROUND SCISSOR LIFT	6	8	12	12	12
27	**UE-0001**	UNDERGROUND EMULSION LOADER	6	8	12	12	12

Table A.3.2. Corrected Underground Mobile Fleet Preventative Maintenance Schedule

#	Equip. Ref.	Equipment name	Weekly	Biweekly	Recorded	Monthly Ins.	250 HR	Recorded	500 HR	1000 HR	2000 HR
1	UL-0001	UNDERGROUND LOADER #1 TORO 7					8	10,12	12	12	12
2	UL-0002	UNDERGROUND LOADER #2 TORO 6					8	8	12	12	12
3	UL-0003	UNDERGROUND LOADER #3 LH-517		6	0		8	12,8,12,7	12	12	12
4	JU-0001	JUMBO DRILL #1	10		8,12,7		12		12	12	12
5	CMJU-0001	JUMBO #1 COMPRESSOR							4	4	4
6	BMJU-0001L	JUMBO DRILL #1 LEFT BOOM"							4	4	4
7	BMJU-0001R	JUMBO DRILL #1 RIGHT BOOM							4	4	4
8	JU-0002	JUMBO DRILL #2	10		8,12,10		12		12	12	12
9	CMJU-0002	JUMBO #2 COMPRESSOR							4	4	4
10	BMJU-0002L	JUMBO BOOMER LEFT BOOM							4	4	4
11	BMJU-0002R	JUMBO BOOMER RIGHT BOOM							4	4	4
12	JB-0001	JUMBO BOLTER #1		10	12,7		12		12	12	12
13	CMJB-0001	JUMBO BOLTER #1 COMPRESSOR							4	4	4
14	DR-78062026	BOLTER ROCK DRILL							4	4	4
	JB-0002	CABLETEC	10		8,12,5,7		12		12	12	12
	CMJB-0002	CABLETEC COMPRESSOR							4	4	4
	BMJB-0002	CABLETEC BOOM							4	4	4
15	UT-0001	UNDERGROUND TRUCK EJC530		6	8,6,7,12	8	8	10,8,6	12	12	12
16	UT-0002	UNDERGROUND TRUCK TH-550		6	0		8		12	12	12
17	US-0001	UNDERGROUND SPRAYMEC		6	8,12,6		10		12	12	12
18	CMUS-0001	SPRAYMEC COMPRESSOR							3	3	3
19	PMUS-0001	SPRAYMEC CONCRETE PUMP							3	3	3
20	US-0002	UNDERGROUND SPRAYMEC		6	8,10		10		12	12	12
21	CMUS-0002	SPRAYMEC COMPRESSOR							3	3	3
22	PMUS-0002	SPRAYMEC CONCRETE PUMP							3	3	3
23	UM-0001	UNDERGROUND MIXER		6	8,7,9,10		8	12	12	12	12
24	UM-0002	UNDERGROUND MIXER		6	8,6,5,12		8	12,10,8,6	12	12	12
25	UM-0003	UNDERGROUND MIXER		6	8		8	10,8	12	12	12
27	UE-0001	UNDERGROUND EMULSION LOADER		6	6,8,5		8		12	12	12

A3.2 Equipment Availability

Equipment availability is defined as downtime/(uptime+downtime)%, where downtime is the time in which the equipment is broken down, and uptime is the time in which the equipment is available for work. The site provided 6 months of shift-by-shift maintenance data which recorded everything for maintaining and repairing the underground fleet. This data was analyzed and the preventive maintenance times of all equipment were removed from the record sheet. Thus, the times left on the sheet would be breakdown and corrective maintenance times. Their sum was used as the downtime in the simulations. The uptime would then be the total available shift hours (9.5 hours) minus the downtime.

The number of failures was also counted during this 6-month period. The mean time between failures (MTBF) is the uptime/number of failures, and the mean time to repair (MTTR) is the downtime/number of failures. In the simulation model, the input for equipment availability was calculated as the downtime/(operating time+ downtime)%; therefore, in the model, when equipment was not working it would never have broken down. In the site data, however, the uptime was merely the calendar time (9.5 hours shift * number of shifts), and the actual operating hours for each piece of equipment were not recorded. Therefore, based on the current and long-term forecast equipment availability performance of 80% to 85% and other operating mines' maintenance records for similar equipment, a conservative 80% availability was selected and used for all equipment in the simulation.

Table A.3.3. MTBF and MTTR Equipment

*All time in hours	Downtime	Uptime	Failures	MTBF	MTTR
JU-0001 Axera & Jumbo	578.5	2632.5	243	10.83	2.38
JU-0002 Atlas Copco M2C	444	2767	196	14.12	2.27
JB-0001 Robolter	409	2802	185	15.15	2.21
JB-0002 Cabletec	142.5	3068.5	48	63.93	2.97
US-0001 Spraymec	164.5	3046.5	59	51.64	2.79
US-0002 Spraymec	431.5	2779.5	168	16.54	2.57
UM-0001 Transmixer	101	3110	33	94.24	3.06
UM-0002 Transmixer	132.5	3078.5	58	53.08	2.28
UM-0003 Transmixer	154.5	3056.5	64	47.76	2.41
UL-0001 Toro 7	679.5	2531.5	187	13.54	3.63
UL-0002 Toro 6	184	3027	111	27.27	1.66
UL-0003 Toro LH 517	446.5	2764.5	179	15.44	2.49
UT-0001 EJC 530	384.5	2826.5	163	17.34	2.36
UT-0002 TH 550	472	2739	93	29.45	5.08
UP-0001 Utilift	145.5	3065.5	64	47.90	2.27
UE-0001 Charmec	103	3108	57	54.53	1.81
TH-0002 TeleHandler	732	2479	136	18.23	5.38
FO-0003 Station Forklift	87.5	3123.5	53	58.93	1.65
LO-5022 Cat Skid Steer	148	3063	72	42.54	2.06
EX-0005 Hitachi Mini Ex	50	3161	25	126.44	2.00
DZ-0006 Cat D3K	341.5	2869.5	94	30.53	3.63
LV-0015 Toyota Truck	134.5	3076.5	81	37.98	1.66
LV-0125 Toyota Truck	166.5	3044.5	75	40.59	2.22
LV-0146 Toyota Truck	513.5	1215.5	67	18.14	7.66

Appendix II: Simulation Tests

Test 1. Base Case Shaft 2 Access Dual Headings and Single Heading Rate

Crew number 2 was designated to work on Mine Area 31, Shaft 2 access (S2A) drives. This area was circled in Figure A.4.1 using yellow lines. Other possible work areas were disabled in the simulation. The blasting time was set to shift end blasting. This crew group includes the standard set of development equipment. The Toro 7 loader's tramming capacity was changed to 14.5 tons in order to calibrate the development rates with other development crew groups. The mine shut down period (December 1, 2011 to May 24, 2012) was removed from the model so that the crew could work continuously during the experimental period.

This mine area includes two primary headings (S2A drives), and some remucks and crosscuts. The two primary headings are both high priority and the remucks and crosscuts are low priority. The average rate was 5.9 meters per day per development crew. Therefore, the result was 2.95 meters per day per heading.

Figure A.4.1. Tested Dual Headings of S2A Drives

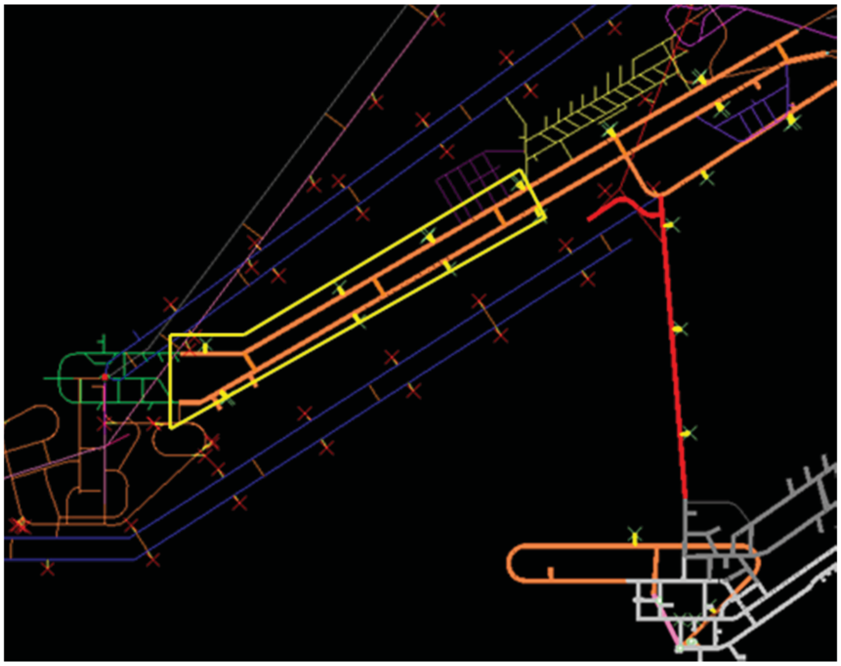


Table A.4.1. Equipment Utilization and Cycle Times on S2A Drives

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft_2_access_high_1 (1)	40.9 %	261		7/26/2011	10/15/2011	53.6	22.5
Shaft_2_access_high_2 (31)	49.1 %	234		10/15/2011	1/12/2012	41.9	21.5
Shaft_2_access_high_3 (31)	45.4 %	118		1/12/2012	3/4/2012	48.5	22.4
Shaft_2_access_high_4 (31)	44.0 %	254		3/4/2012	5/30/2012	52.1	22.3
Shaft_2_access_high_5 (4)	47.1 %	185		8/1/2012	10/13/2012	429.9	22.0
Shaft_2_access_high_6 (4)	47.3 %	140		10/13/2012	12/9/2012	45.9	21.8
Shaft_2_access_low_1 (1)	42.0 %	200		8/17/2011	10/9/2011	50.0	20.8
Shaft_2_access_low_2 (31)	53.8 %	61		10/9/2011	11/1/2011	38.6	20.4
Shaft_2_access_low_3 (31)	49.8 %	242		11/1/2011	2/4/2012	42.3	21.1
Shaft_2_access_low_4 (31)	52.4 %	173		2/4/2012	4/11/2012	40.0	22.2
Shaft_2_access_low_5 (31)	38.6 %	93		4/11/2012	5/25/2012	55.3	21.8

Table A.4.2. Dual Heading Development Rates in Mine Area 31

	Working days	Distance (m)	Rate (m/d)
Nov-11	30	188	6.3
Dec-11	31	170	5.5
Jan-12	31	163	5.3
Feb-12	29	182	6.3
Mar-12	31	168	5.4
Apr-12	30	201	6.7
Total		1072.0	5.9

Figure A.4.2. Monthly Development Meters and Available Headings of Mine Area 31

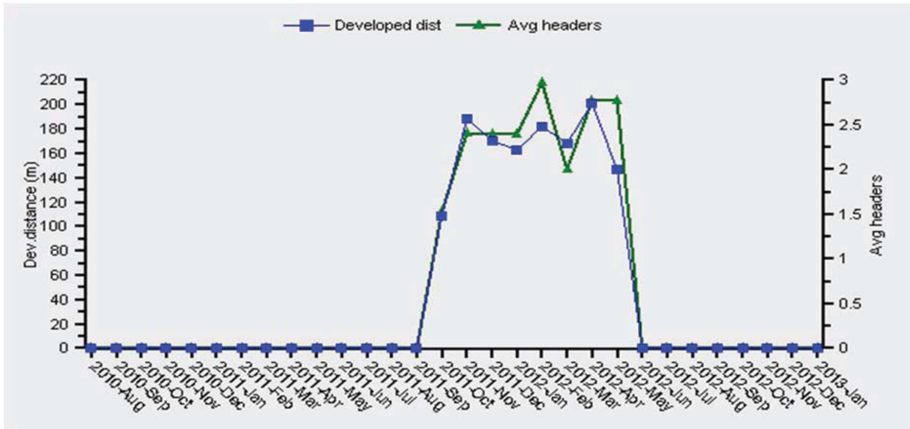


Table A.4.3. Equipment Utilization and Cycle Times on S2A

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft_2_access_high_1 (1)	40.9 %	261		7/26/2011	10/15/2011	53.6	22.5
Shaft_2_access_high_2 (31)	49.1 %	234		10/15/2011	1/12/2012	41.9	21.5
Shaft_2_access_high_3 (31)	45.4 %	118		1/12/2012	3/4/2012	48.5	22.4
Shaft_2_access_high_4 (31)	44.0 %	254		3/4/2012	5/30/2012	52.1	22.3
Shaft_2_access_high_5 (4)	47.1 %	185		8/1/2012	10/13/2012	429.9	22.0
Shaft_2_access_high_6 (4)	47.3 %	140		10/13/2012	12/9/2012	45.9	21.8
Shaft_2_access_low_1 (1)	42.0 %	200		8/17/2011	10/9/2011	50.0	20.8
Shaft_2_access_low_2 (31)	53.8 %	61		10/9/2011	11/1/2011	38.6	20.4
Shaft_2_access_low_3 (31)	49.8 %	242		11/1/2011	2/4/2012	42.3	21.1
Shaft_2_access_low_4 (31)	52.4 %	173		2/4/2012	4/11/2012	40.0	22.2
Shaft_2_access_low_5 (31)	38.6 %	93		4/11/2012	5/25/2012	55.3	21.8

Single heading development rates were tested in Mine Area 30, Shaft 2 access (S2A) top ramp. This area is circled in yellow lines in Figure A.4.3. Crew number 2 was designated to work only on this ramp after Mine Area 31 was finished. The single heading rate was experimented on in the same model as the dual heading test. This area included a single heading ramp and remucks. The development rate was 3.4 meters per day per heading.

Figure A.4.3. Tested Single Heading S2A Top Ramp

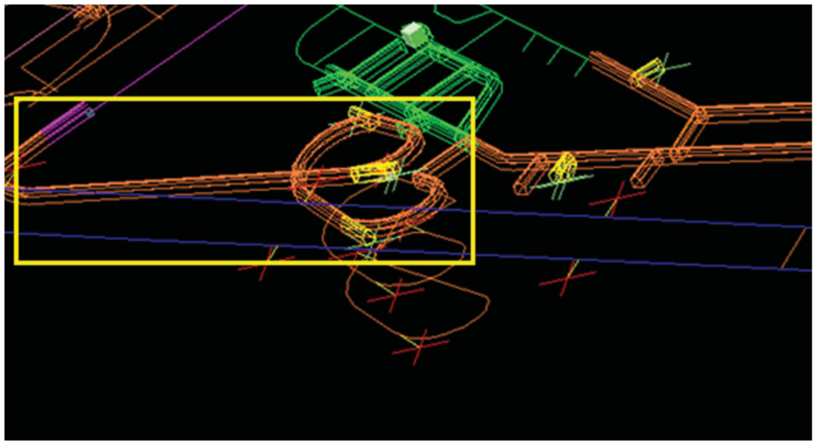


Table A.4.4. Single Heading Development Rates in Mine Area 30

	Working days	Distance (m)	Rate (m/d)
Jun-12	28	96	3.4
Jul-12	28	98	3.5
Aug-12	28	97	3.5
Sep-12	27	95	3.5
Oct-12	29	91	3.1
Total		381.0	3.4

Figure A.4.4. Monthly Development Meters and Available Headings of Mine Area 30

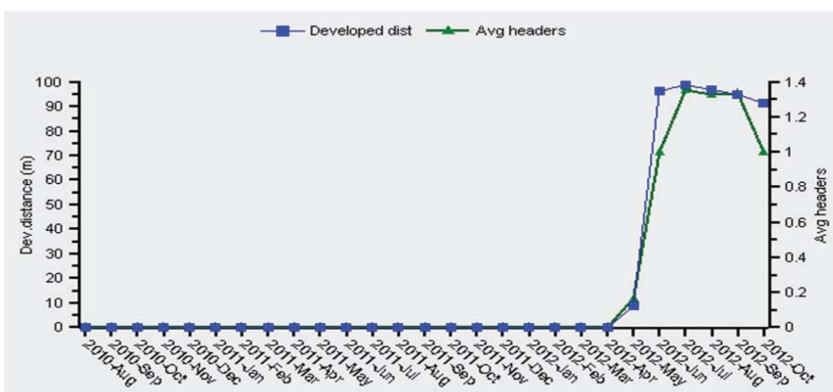


Table A.4.5. Equipment Utilization and Cycle Times on S2A Top Ramp

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_ramp_top_1 (30)	52.0 %	137		5/27/2012	7/17/2012	45.2	22.0
Shaft2_ramp_top_2 (30)	48.5 %	108		7/17/2012	8/24/2012	40.8	20.2
Shaft2_ramp_top_3 (30)	53.5 %	192	42	8/24/2012		39.4	21.5

Test 2. S2A Drives Middle Shift Blasting Rate

This test was based on the same model as that of Test 1. Therefore, it used the same designated development crew (number 2). However, the blasting times were set to both shift end and middle shift blasting. The blasting times were at 0:00, 6:00, 12:00, and 18:00 each day, and these times were applied to all mine areas.

Both dual headings (S2A drives) and single heading (S2A top ramp) were experimented on for these blasting times. The average development rate was 6.4 meters per day per development crew for S2A drives of dual-heading case. For the single-heading S2A top ramp, the rate was 3.5 meters per day. The development rates were increased by 8.5% for dual headings and 3% for single headings development as compared with those of the base case Test 1.

Table A.4.6. The Mid-shift Dual Headings Development Rates of S2A Drives

	Working days	Distance (m)	Rate (m/d)
Oct-11	29	197	6.8
Nov-11	30	190	6.3
Dec-11	31	189	6.1
Jan-12	31	184	5.9
Feb-12	29	177	6.1
Mar-12	31	227	7.3
Total		1164.0	6.4

Figure A.4.5. Monthly Development Meters and Available Headings of Mine Area 31 with Mid-shift Blasting

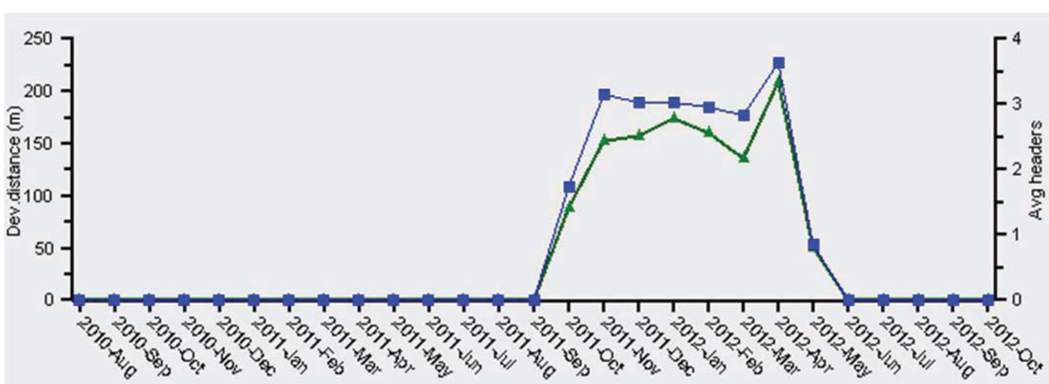


Table A.4.7. The Mid-shift Blasting Single Heading Development Rates of S2A Top Ramp

	Working days	Distance (m)	Rate (m/d)
Jun-12	28	113.0	4.0
Jul-12	28	92.0	3.3
Aug-12	28	91.0	3.3
Sep-12	27	104.0	3.9
Oct-12	29	91.0	3.1
Total			3.5

Figure A.4.6. Monthly Development Meters and Available Headings of Mine Area 30 with Mid-shift Blasting

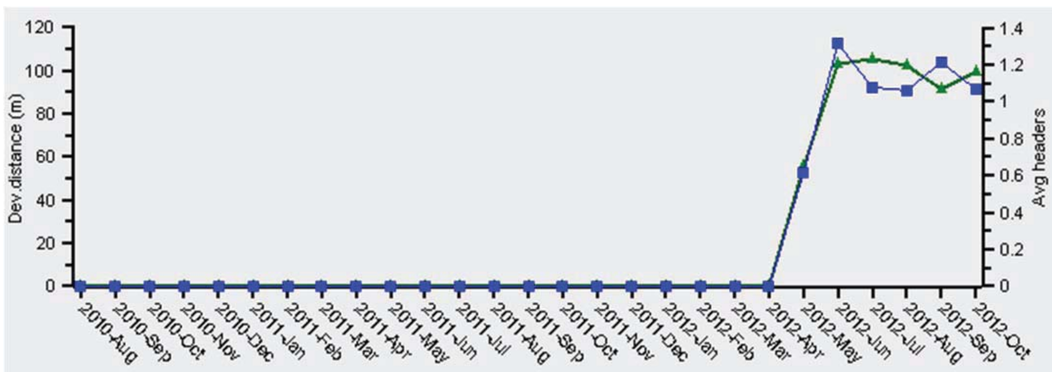


Table A.4.8. Mid-shift Blasting Equipment Utilization and Cycle Times

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft_2_access_high_2 (31)	51.4 %	234		10/14/2011	1/4/2012	39.2	20.6
Shaft_2_access_high_3 (31)	50.8 %	118		1/4/2012	2/16/2012	39.7	20.8
Shaft_2_access_high_4 (31)	46.6 %	254		2/16/2012	5/12/2012	44.4	21.4
Shaft_2_access_high_5 (4)	43.0 %	185		8/1/2012	10/10/2012	430.0	21.3
Shaft_2_access_high_6 (4)	49.7 %	54	58	10/10/2012		43.8	22.4
Shaft_2_access_low_1 (1)	39.8 %	200		8/10/2011	10/11/2011	55.6	21.5
Shaft_2_access_low_2 (31)	46.7 %	61		10/11/2011	11/6/2011	47.0	20.6
Shaft_2_access_low_3 (31)	52.2 %	242		11/6/2011	1/31/2012	38.6	20.7
Shaft_2_access_low_4 (31)	58.4 %	173		1/31/2012	4/4/2012	36.8	22.5
Shaft_2_access_low_5 (31)	46.1 %	93		4/4/2012	5/12/2012	48.7	23.4

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_ramp_top_1 (30)	55.1 %	137		5/12/2012	6/28/2012	37.1	21.1
Shaft2_ramp_top_2 (30)	50.7 %	108		6/28/2012	8/4/2012	39.9	20.7
Shaft2_ramp_top_3 (30)	47.5 %	240	28	8/4/2012		42.4	23.2

Test 3. S2A Drives with Two Secondary Headings

In this test, office drifts were developed while the S2A drives advanced to the office area (Mine Area 36, circled in Figure A.4.9. with yellow lines). Two office drifts were scheduled to open at one time as two secondary development headings for the same crew as Test 1. S2A drives, crosscuts and remucks were given higher priority. Office drifts had the same heading profile as S2A drives (5mWx5.5mH, 5 m wide by 5.5 m high, “I” type).

The results were selected from December, 2011 to March, 2012 since only during this time were both of the two primary and secondary headings developing. The rate of the primary headings was 4.9 meters per day and that of the secondary headings was 3.6 meters per day. Total development rates for these crews were 8.6 meters per day, increasing by 45.8% from the base case Test 1. This test demonstrated that, should more headings opened, the development rate would increase significantly.

Table A.4.9. Primary S2A with Two Secondary Office Drifts Development Rate

		Dual Heading Area 31		Secondary Heading Area 36		
	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
Dec-11	31	161	5.2	128	4.1	9.3
Jan-12	31	144	4.6	125	4.0	8.7
Feb-12	29	152	5.2	91	3.1	8.4
Mar-12	31	145	4.7	101	3.3	7.9
Total		602.0	4.9	445.0	3.6	8.6

Figure A.4.7. Monthly Development Meters and Available Headings of Mine Area 31

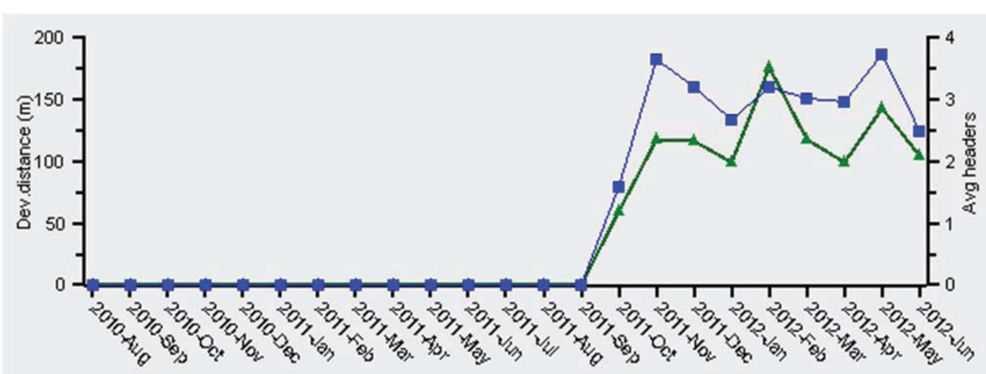


Figure A.4.8. Monthly Development Meters and Available Headings of Mine Area 36

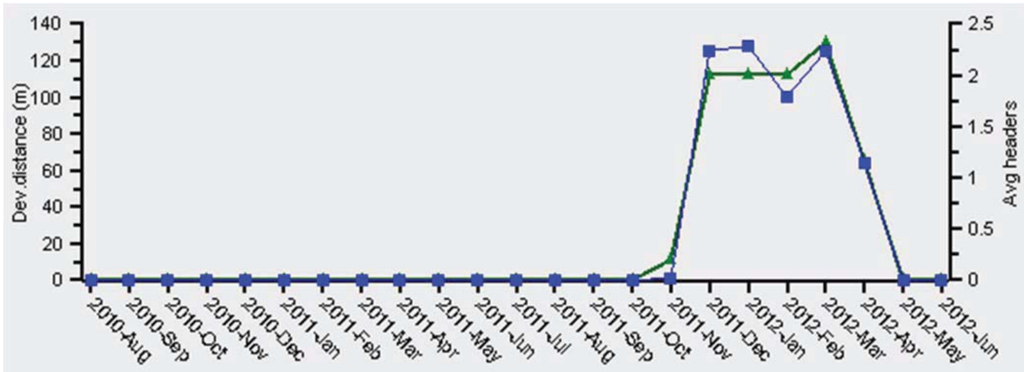
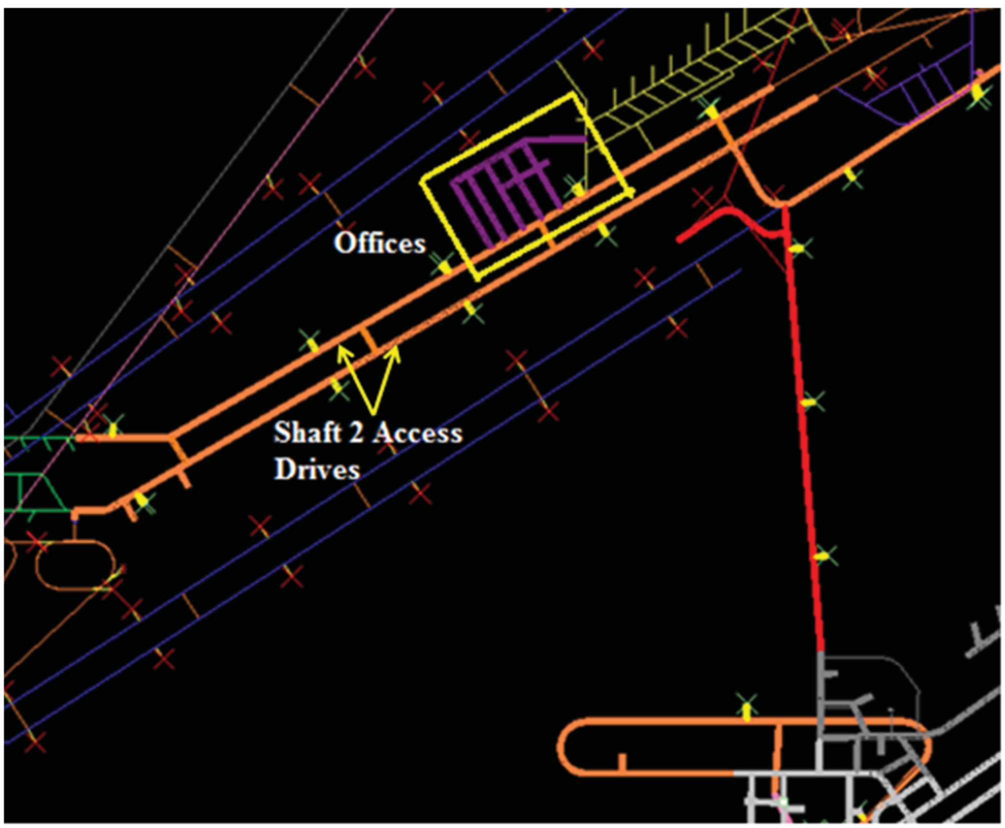


Figure A.4.9. Dual Heading (S2A drives) with Two Secondary Headings (Offices) Open



Test 4. Main Conveyor and Service Drift

Crew number 1 was designated to work on Mine Area 8. Mine Area 8 included two primary headings, the main conveyor drift (MCD) and the service drift (SD), and some remucks and crosscuts. Other possible work areas were disabled during the simulation. This crew includes the typical set of equipment. The mine shut down period (December 1, 2011 to May 24, 2012) was removed from the model.

The two headings are of the same high priority and the remucks and crosscuts are of low priority. The service and conveyor drifts were modified to “I” type profiles (5mWx5.5mH) in order to make them comparable with the previous test on S2A areas.

In the simulation test, the true dual heading development situation began in August, 2012, since before this time the average heading was above 3 since the transfer station provided additional headings. The average development rate was 5.7 meters per day per development crew. Therefore, the result was 2.85 meters per day per heading. From this test, it can be seen that, with the development moved farther away from Shaft 1, the development rate decreased slightly by 3.4%.

Table A.4.10. Dual Headings Development Rates of MCD and SD

	Working days	Distance (m)	Rate (m/d)
Mar-12	31	154	5.0
Apr-12	30	170	5.7
May-12	29	144	5.0
Jun-12	28	192	6.9
Jul-12	28	184	6.6
Aug-12	28	222	7.9
Sep-12	27	134	5.0
Oct-12	29	149	5.1
Nov-12	30	155	5.2
Dec-12	31	180	5.8
Jan-13	31	160	5.2
Feb-13	28	165	5.9
Mar-13	31	166	5.4
Apr-13	30	153	5.1
Total		2328.0	5.7

Figure A.4.10. Monthly Development Meters and Available Headings of Mine Area 8

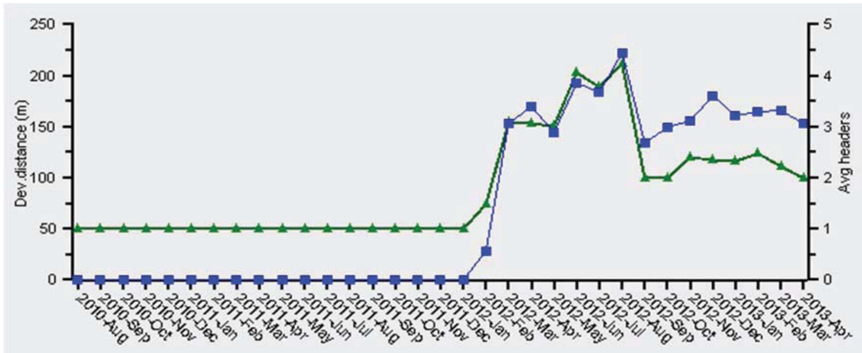
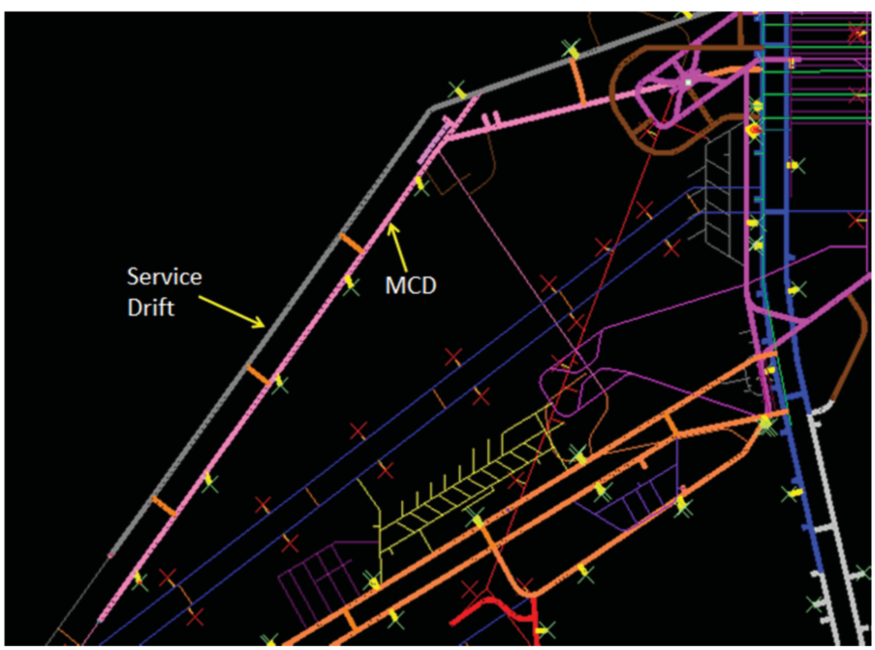


Table A.4.11. Equipment Utilization and Cycle Times on MCD and SD

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	44.8 %	372		3/22/2012	8/12/2012	48.4	23.0
Conveyor_drive_2 (8)	38.5 %	206		8/1/2010	10/2/2012	1,183.8	22.6
Conveyor_drive_3 (8)	51.6 %	159		10/4/2012	12/3/2012	45.0	22.4
Conveyor_drive_4 (8)	42.7 %	163		12/3/2012	2/8/2013	54.3	21.1
Conveyor_drive_5 (8)	49.8 %	155		2/8/2013	4/9/2013	44.3	21.9
Conveyor_drive_6 (8)	53.3 %	53	7	4/9/2013		45.5	24.7

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	48.1 %	156		2/23/2012	4/28/2012	46.5	22.8
Service_drive_2 (8)	38.5 %	149		4/28/2012	7/8/2012	64.5	23.8
Service_drive_3 (8)	44.8 %	233		7/8/2012	10/21/2012	51.3	22.5
Service_drive_4 (8)	46.9 %	232		10/21/2012	1/25/2013	47.5	22.6
Service_drive_5 (8)	48.6 %	170		1/25/2013	4/8/2013	47.3	22.6
Service_drive_6 (8)	53.9 %	58	23	4/8/2013		43.8	23.8

Figure A.4.11. Tested Dual Headings MCD and SD



Test 5. MCD and SD with Two Secondary Headings

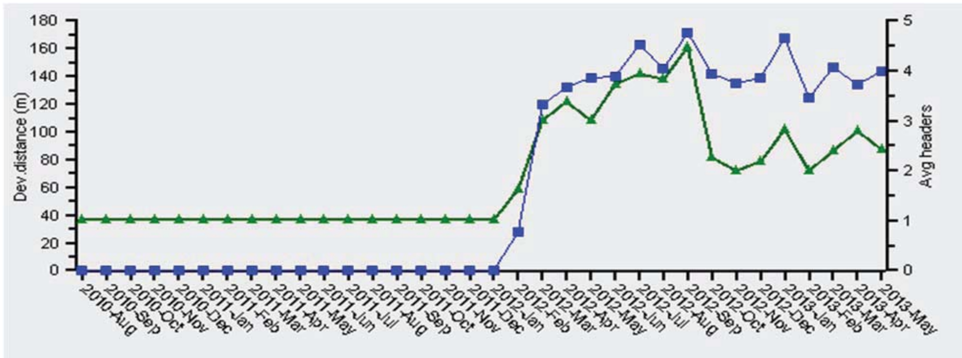
Shaft 2 intake drifts (S2ID) were linked as start-to-start triggers with two primary headings of Test 4, MCD and SD. These two headings were scheduled to be low priority secondary headings for the same development crew (#1). MCD, SD, crosscuts and remucks were given higher priority. All of the primary and secondary headings were changed to “I” type profiles (5mWx5.5mH).

The results were selected from March, 2012 to May, 2013 because, during this time, all of the four primary and secondary headings were developing and MCD and SD were in a true two-headings advancing situation. The development rate of the primary headings was 4.9 meters per day and that of the secondary headings was 3.5 meters per day. The total development rate was 8.4 meters per day per development crew, which increased by 42.3% from the base case. This test demonstrated that, when development moved farther away from Shaft No.1, the development rate decreased slightly by 2.4%.

Table A.4.12. Primary MCD and SD with Two Secondary S2ID Development Rates

	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
Mar-12	31	120	3.9	120	3.9	7.7
Apr-12	30	132	4.4	106	3.5	7.9
May-12	29	139	4.8	105	3.6	8.4
Jun-12	28	139	5.0	110	3.9	8.9
Jul-12	28	163	5.8	100	3.6	9.4
Aug-12	28	146	5.2	96	3.4	8.6
Sep-12	27	172	6.4	77	2.9	9.2
Oct-12	29	142	4.9	120	4.1	9.0
Nov-12	30	135	4.5	111	3.7	8.2
Dec-12	31	139	4.5	120	3.9	8.4
Jan-13	31	167	5.4	106	3.4	8.8
Feb-13	28	125	4.5	96	3.4	7.9
Mar-13	31	146	4.7	96	3.1	7.8
Apr-13	30	134	4.5	86	2.9	7.3
May-13	29	144	5.0	101	3.5	8.4
Total		2143.0	4.9	1550.0	3.5	8.4

Figure A.4.12. Monthly Development Meters and Available Headings of Mine Area 8 and 37



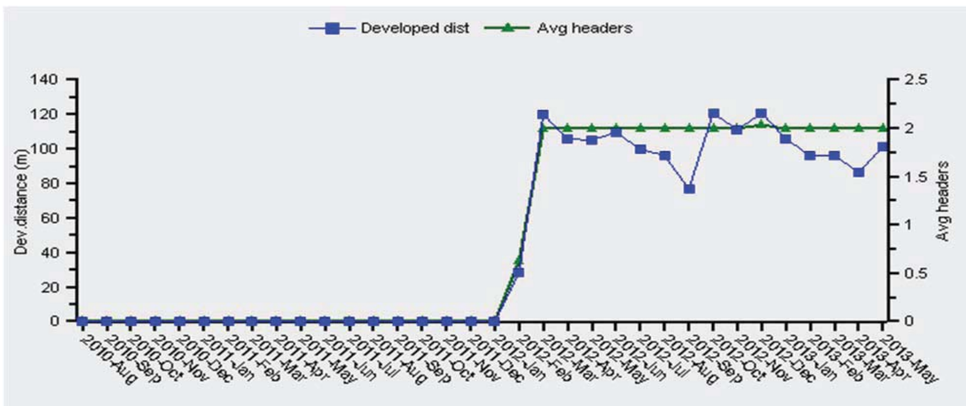


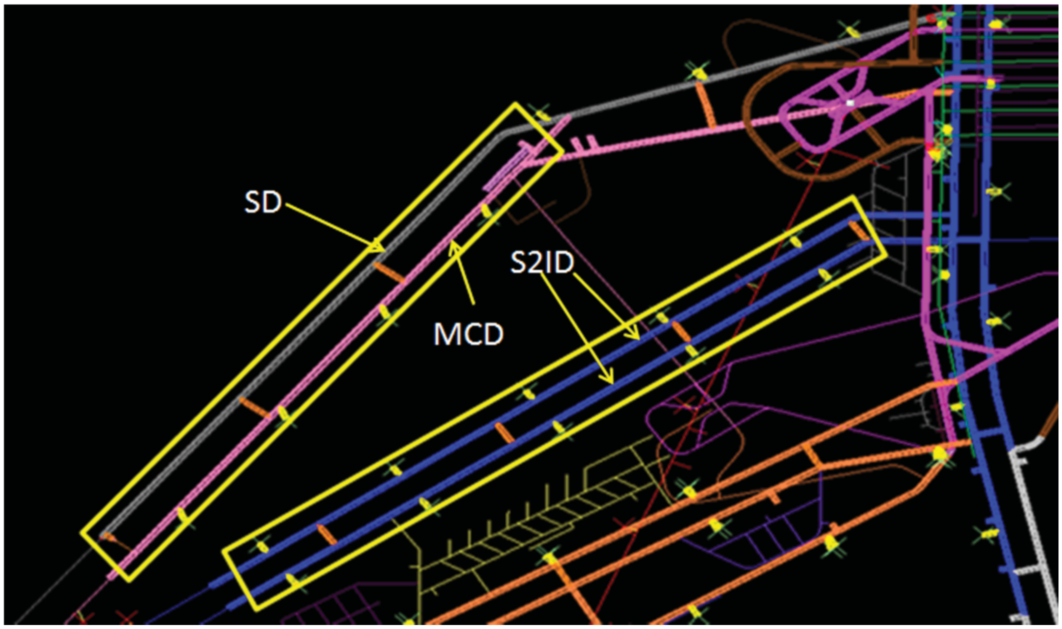
Table A.4.13. Equipment Utilization and Cycle Times on MCD and SD and S2ID

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	36.5 %	372		3/31/2012	9/25/2012	62.1	23.4
Conveyor_drive_2 (8)	33.0 %	206		8/1/2010	11/18/2012	1,248.4	22.1
Conveyor_drive_3 (8)	44.3 %	159		11/18/2012	1/26/2013	49.3	21.8
Conveyor_drive_4 (8)	38.7 %	163		1/26/2013	4/13/2013	53.3	21.3
Conveyor_drive_5 (8)	33.8 %	88	42	4/13/2013		63.9	21.6

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	39.4 %	156		2/21/2012	5/11/2012	55.9	23.0
Service_drive_2 (8)	39.3 %	149		5/11/2012	7/21/2012	54.3	21.9
Service_drive_3 (8)	33.5 %	233		7/21/2012	11/23/2012	63.8	21.9
Service_drive_4 (8)	43.7 %	232		11/23/2012	3/13/2013	53.5	24.2
Service_drive_5 (8)	41.3 %	160	10	3/13/2013		54.3	23.3

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_main_vent_drive_1 (37)	35.2 %	210		2/21/2012	6/14/2012	57.6	21.9
Shaft2_main_vent_drive_10 (37)	28.4 %	154	29	2/17/2013		74.0	21.4
Shaft2_main_vent_drive_2 (37)	31.6 %	191		2/22/2012	6/11/2012	62.7	21.1
Shaft2_main_vent_drive_3 (37)	30.5 %	175		6/14/2012	10/9/2012	75.2	22.2
Shaft2_main_vent_drive_4 (37)	32.4 %	172		6/11/2012	10/4/2012	70.3	21.4
Shaft2_main_vent_drive_5 (37)	35.9 %	149		10/9/2012	12/23/2012	57.1	21.2
Shaft2_main_vent_drive_6 (37)	39.5 %	155		10/4/2012	12/19/2012	56.0	22.9
Shaft2_main_vent_drive_7 (37)	35.3 %	163		12/23/2012	3/25/2013	61.7	22.4
Shaft2_main_vent_drive_8 (37)	30.7 %	100		12/19/2012	2/15/2013	62.5	21.3
Shaft2_main_vent_drive_9 (37)	32.5 %	109	13	3/25/2013		70.4	22.8

Figure A.4.13. Tested Dual Heading MCD and SD with S2ID as Secondary Headings



Test 6. Four Headings Limited to Two Concurrent Activities

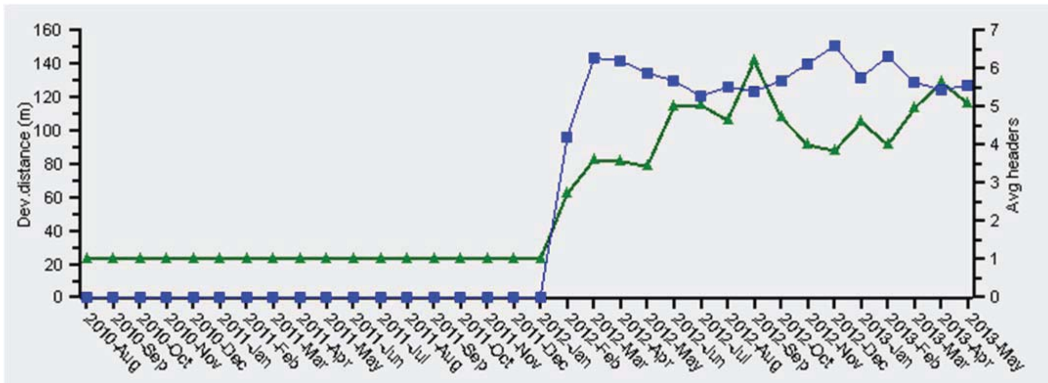
This test was slightly modified from Test 5 but the concurrent activities were limited to two for the selected Mine Areas 8 (MCD and SD) and 37 (S2ID). For example, when the two available faces are on drilling and mucking activity performance in Mine Area 8, no other activities are allowed. All other setups of this test were the same as those of Test 5.

The results were also selected from September, 2012 to April, 2013. The development rate of primary headings was 4.5 meters per day and for secondary headings was 3.7 meters per day. The total development was 8.2 meters per day, decreasing by 2.3% from Test 5. This test demonstrated that if the maximum concurrent activities are constrained, the development rate will drop slightly. Each test mine area, however, had only two available headings and usually on each heading only one activity took place. Therefore, the test showed a very slight decrease in the development rate.

Table A.4.14. Four Headings Limited to Two Concurrent Activities

		Dual Heading Area 8		Secondary Heading Area 37		
	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
Sep-12	27	124	4.6	115.0	4.3	8.9
Oct-12	29	129	4.4	101.0	3.5	7.9
Nov-12	30	140	4.7	139	4.6	9.3
Dec-12	31	151	4.9	116.0	3.7	8.6
Jan-13	31	131	4.2	120.0	3.9	8.1
Feb-13	28	144	5.1	106.0	3.8	8.9
Mar-13	31	129	4.2	96.0	3.1	7.3
Apr-13	30	124	4.1	73.0	2.4	6.6
Total		1072.0	4.5	866.0	3.7	8.2

Figure A.4.14. Monthly Development Meters and Available Headings of Mine Area 8 and 37



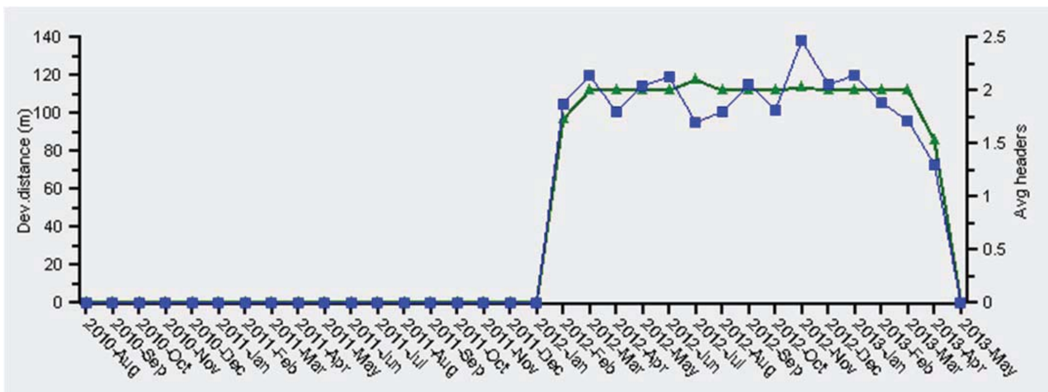


Table A.4.15. Equipment Utilization and Cycle Times on MCD and SD

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	38.8 %	372		3/13/2012	8/29/2012	55.1	24.4
Conveyor_drive_2 (8)	32.2 %	206		8/16/2012	11/7/2012	1,268.3	24.0
Conveyor_drive_3 (8)	45.4 %	159		11/8/2012	1/13/2013	46.2	21.5
Conveyor_drive_4 (8)	43.6 %	163		1/30/2013	4/10/2013	56.2	21.7
Conveyor_drive_5 (8)	44.9 %	63	2	5/3/2013		93.4	23.2

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	47.2 %	156		2/5/2012	4/12/2012	47.2	23.1
Service_drive_2 (8)	42.8 %	149		4/18/2012	6/27/2012	52.9	22.2
Service_drive_3 (8)	31.4 %	233		7/11/2012	11/24/2012	68.0	21.6
Service_drive_4 (8)	47.1 %	232		12/7/2012	3/16/2013	56.6	22.8
Service_drive_5 (8)	50.8 %	155	14	3/20/2013		56.3	25.6

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_main_vent_drive_1 (37)	36.1 %	210		2/5/2012	5/22/2012	58.6	21.5
Shaft2_main_vent_drive_10 (37)	31.4 %	183		1/5/2013	4/26/2013	65.0	21.1
Shaft2_main_vent_drive_2 (37)	34.6 %	191		2/5/2012	5/19/2012	62.0	21.8
Shaft2_main_vent_drive_3 (37)	28.8 %	175		5/22/2012	9/7/2012	89.0	21.5
Shaft2_main_vent_drive_4 (37)	34.0 %	172		5/19/2012	8/25/2012	62.3	21.9
Shaft2_main_vent_drive_5 (37)	38.8 %	149		9/8/2012	11/19/2012	55.9	21.5
Shaft2_main_vent_drive_6 (37)	33.8 %	155		8/25/2012	11/18/2012	60.4	21.1
Shaft2_main_vent_drive_7 (37)	39.1 %	163		11/19/2012	2/7/2013	54.2	21.9
Shaft2_main_vent_drive_8 (37)	36.3 %	100		11/18/2012	1/5/2013	54.8	21.1
Shaft2_main_vent_drive_9 (37)	32.1 %	122		2/7/2013	4/22/2013	66.1	21.8

Test 7. Shaft 2 Station All Headings Open

The objective for this test was to determine the maximum development rate which one crew could achieve. The Shaft 2 station (S2S) was developed after the S2A drives had been completed. The S2S drifts were of the "I" type profile (5mWx5.5mH).

All headings in S2S were available for Crew # 2 in this test. This crew was designated to work on Mine Area 13 (S2S). From simulation time June to August 2012 S2S had an average of 4 to 5.5 headings open each month. Therefore, the results from these three months were collected. The average rate was 8.4 meters per day. This rate was similar to those of Test 3 (S2A drives with 2 secondary headings) which had about 4.5 headings open on average during its development. Therefore, the maximum development rate one development crew could achieve was 8.5 m/d with 5 headings or more available in the OT case.

Table A.4.16. Shaft 2 Station Multiple Headings Development Rates

	Working days	Distance (m)	Rate (m/d)
Jun-12	28	224	8.0
Jul-12	28	244	8.7
Aug-12	28	237	8.5
Total	84	705.0	8.4

Figure A.4.15. Monthly Development Meters and Available Headings of Mine Area 13

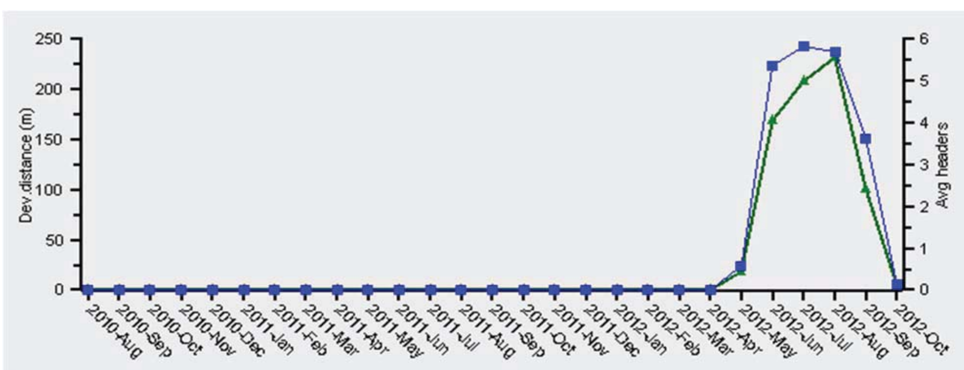


Table A.4.17. Equipment Utilization and Cycle Times at the Shaft 2 Station

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_station_2 (13)	33.4 %	885		5/27/2012	10/4/2012	64.9	21.3

Figure A.4.16. Tested Shaft 2 Station with Multiple Headings Available

Test 8. Undercut and Extraction Drive

Two development crews were assigned to footprint areas in the PPD. One designated crew was working at the extraction level (Mine Area 10) and the other one was at the undercut level (Mine Area 9). Each crew had a complete set of development equipment. The development mucks were dumped at the closest orepass located on the perimeter drives. Four extraction drives and four undercut drives were tested in this model. The drives all started to advance simultaneously from west to east.

All of these drives used campaigned cable bolting and a second layer of shotcrete ground support performed in each 50 meter development interval. In the model, all the footprint drifts were broken into 50 m segments, and between each two segments, a short drift was created (no development meters or muck were generated) to simply represent the cable bolting and second layer of shotcrete activity. For example, the footprint headings advanced 50 m (about 10 rounds) on drilling, charging, blasting, mucking, shotcreting, bolting, meshing and strapping (where needed). The heading then ceased and a very long round of cable bolting and a second layer of shotcrete were applied as campaign practice on this 50 m drift. All ground support regimes were derived from the MS3 drift design. 80% Type 1 (good ground), and 20% Type 2 (poor ground) was assumed. Due to lesser ground support design of the perimeter drifts, a 100% extraction level Type 2 ground support style was assumed.

The rates were measured by the drift length divided by the actual number of working days. Thus the average development rates for one crew were the sum of these rates for four drives. For example, the extraction drive average rate was the sum of the development rates of the extraction drive 1, 2, 3, and 4; extraction drive 1's drift length is 323 m and it was completed in 210 days, so its development rate was 1.47 m/d. The development rates were all tested with four available headings. The undercut drive rate was 8.45 m/d; the extraction drive rate was 5.7 m/d; and the perimeter drift was 2.13 m/d. It has been noticed, however, that the ground support requirements assumption for the rims of the 100% Type 2 extraction level pattern might have been overestimated, and therefore a faster rate 3 m/d would be suggested.

Table A.4.18. Shaft 2 Station Multiple Headings Development Rate

	Start	End	Working days	Distance (m)	Rate (m/d)
Extraction Drive 1	05/01/2014	21/08/2014	219	323	1.47
Extraction Drive 2	05/01/2014	16/08/2014	215	323	1.50
Extraction Drive 3	05/01/2014	02/09/2014	232	325	1.40
Extraction Drive 4	05/01/2014	18/09/2014	245	325	1.33
Extraction Drive Avg					**5.70**
Undercut Drive 1	02/01/2014	29/04/2014	120	256	2.13
Undercut Drive 2	02/01/2014	02/05/2014	123	256	2.08
Undercut Drive 3	02/01/2014	27/04/2014	118	256	2.17
Undercut Drive 4	02/01/2014	03/05/2014	124	256	2.06
Undercut Drive Avg					**8.45**
Perimeter Drift 1	01/01/2014	15/09/2015	595	323	0.54
Perimeter Drift 2	01/01/2014	10/08/2015	563	323	0.57
Perimeter Drift 3	01/01/2014	19/11/2015	659	325	0.49
Perimeter Drift 4	01/01/2014	18/10/2015	627	325	0.52
Perimeter Drift Avg					**2.13**

Figure A.4.17. Tested Undercut, Extraction Drives and Perimeter Drifts

Test 9. Shaft 1 Workshop Massive Excavation

Input

Three runs were completed in the Shaft 1 workshop (S1W) with six bays. The first run had one bay developing at a time, the second had two bays developing at a time, and the third had three bays developing at a time. One bay needed be developed from two cuts, the first being the top cut and the second the bottom cut.

The profile of the top cut was 7.0mWx5mH, and the cross sectional area was about 31.5 m2. The bottom cut was 7.0mWx3.7mH, and the cross sectional area was about 25.9 m^2. 90 face drill holes were assumed for the top cut which was similar to the 6mWx7mH exhaust drift (of 91 holes). 60 face drill holes were assumed for the bottom cut which had a similar cross sectional area to the 5mWx5.5mH heading (of 61 holes). Ground support was assumed to be of sandwiching type shotcreting (50 mm shotcrete in the first layer, a layer of mesh, and 50 mm of shotcrete in the second layer), a 1 m x 1 m grid resin rebar, and 2 m x 2 m grid cable bolting. Details of ground support were listed in the table below.

Table A.4.19. General Assumptions for Face Drill Hole and Support Patterns

7.0mWx8.7mH	Dimension	# of Face holes	# of Cables/m	# of Bolts/m	Shotcrete/m
Top cut	7.0mWx5mH	90	4	16	3 m^3
Bottom cut	7.0mWx3.7mH	65	1	6	2 m^3

Output

Mass excavation rates varied depending on the number of opened bays open. 3 bays were kept open at a time, i.e. developing bays 1, 2 and 3 first and, after they were finished, bays 4, 5 and 6 were then opened. The excavation rate in this run could achieve 131 m^3 per day. Some reduction factor, however, should be applied to reflect overestimations on the outcome. This was since the model did not take into consideration the muck handling bottleneck in Shaft 1. In addition, the model did not include interactions between other development crews and mine areas. One crew (including the loader, jumbo, bolter, cable bolter, charger, and shotcrete vehicle) was designated to work on these 6 bays. In reality, however, some of the equipment might be shared and relocated to other development headings.

Therefore, a 25% reduction factor was applied to the 3 open case bays which was similar to the model reduction factor for the lateral development rates. For Runs #1 and #2, the results were not reduced by 25% since this ratio was not applicable if only one or two bays were open at a time. Three bays' available conditions might not always be simultaneously achievable, but it would be very realistic to simultaneously achieve one or two bays' available conditions for one development crew. In conclusion, 100 m^3/d is suggested for the mass excavation rate of 7.0mWx8.7mH bays with two cuts. This was derived from 3 bays' open run being de-rated by a 25% model reduction factor.

Table A.4.20. Shaft 1 Workshop Massive Excavation Rates

	Bay Volume (m^3)	Modeled Exc. Rates (m^3/d)	Rad. Factors	Rec. Rates (m^3/d)	Remarks
Run #1	2755	53	0%	53	1 bay open
Run #2	2755	96	12.5%	80	2 bay open
Run #3	2755	131	25%	100	3 bay open

Figure A.4.18. Tested Shaft 1 Workshop Bays Massive Excavation

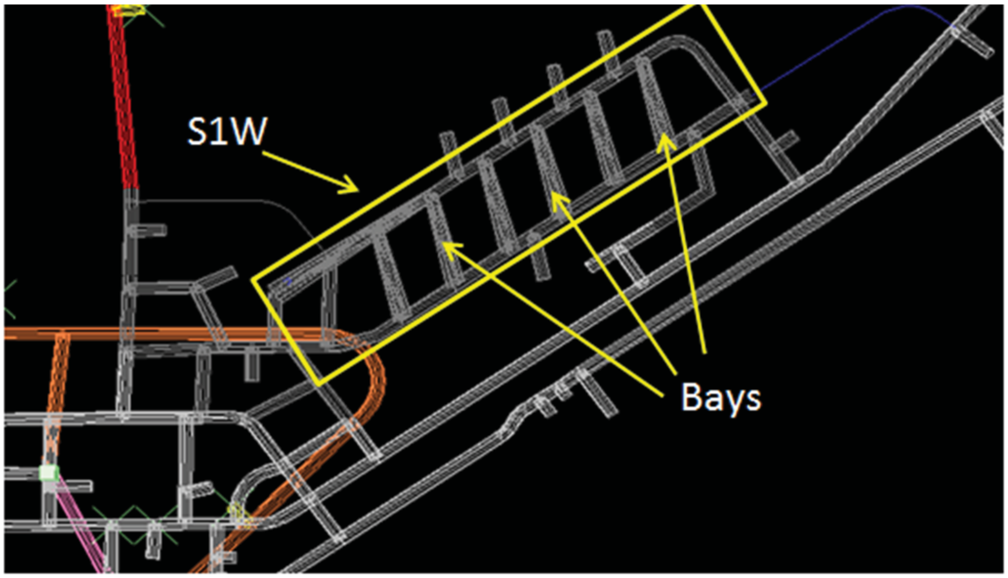


Table A.4.21. Mass Excavation Run 1: 1 Bay Open

Run #1	Start	Finish	Working Days	Bay Volume (m3)	Exc. Rate (m3/d)
Bay 1	02/03/2011	25/04/2011	53	2755	52
Bay 2	25/04/2011	21/06/2011	54	2755	51
Bay 3	21/06/2011	15/08/2011	50	2755	55
Bay 4	15/08/2011	15/10/2011	53	2755	52
Bay 5	15/10/2011	05/12/2011	51	2755	54
Bay 6	05/12/2011	26/01/2012	52	2755	53
				Avg Exc. Rate	**53**

Table A.4.22. Mass Excavation Run 2: 2 Bay Open

Run #2	Start	Finish	Working Days	Bay Volume (m3)	Exc. Rate (m3/d)
Bay 1	11/03/2011	07/05/2011	56	2755	49
Bay 2	11/03/2011	09/05/2011	58	2755	48
Bay 3	09/05/2011	01/07/2011	53	2755	52
Bay 4	09/05/2011	18/07/2011	67	2755	41
Bay 5	18/07/2011	14/09/2011	53	2755	52
Bay 6	18/07/2011	19/09/2011	58	2755	48
				Avg Exc. Rate	**96**

Table A.4.23. Mass Excavation Run 3: 3 Bay Open

Run #3	Start	Finish	Working Days	Bay Volume (m3)	Exc. Rate (m3/d)
Bay 1	08/04/2011	17/06/2011	67	2755	41
Bay 2	08/04/2011	20/06/2011	70	2755	39
Bay 3	08/04/2011	10/06/2011	60	2755	46
Bay 4	20/06/2011	01/09/2011	68	2755	41
Bay 5	20/06/2011	19/08/2011	55	2755	50
Bay 6	20/06/2011	25/08/2011	61	2755	45
				Avg Exc. Rate	**131**

Test 10. Jumbo with Face Drilling and Bolting Functions

The M2C drill jumbo and Boltec bolter of Crew #2 were removed during this test. A new multi-functional machine was added to the crew to perform both face drilling and bolting. All other input was the same as that for Test 1.

The Development rate of the S2A drives of dual headings dropped to 5.3 meters per day, which was an 8.6% decrease from the base cast of Test 1. For the single heading S2A top ramp, the development rate decreased slightly by 5.9% to 3.2 meters per day. It could be interpreted from this test that the development rate would drop if one piece of equipment were used to perform both drilling and bolting rather than separate pieces of equipment; with the number of available headings increased, the development rate would be further affected by this type of practice.

Table A.4.24. Dual Heading Development Rates with Jumbo Drilling and Bolting

	Working days	Distance (m)	Rate (m/d)
Nov-11	30	169	5.6
Dec-11	31	161	5.2
Jan-12	31	139	4.5
Feb-12	29	160	5.5
Mar-12	31	170	5.5
Apr-12	30	158	5.3
May-12	29	168	5.8
Total	211	1125.0	5.3

Figure A.4.19. Monthly Development Meters and Available Headings of Mine Area 31

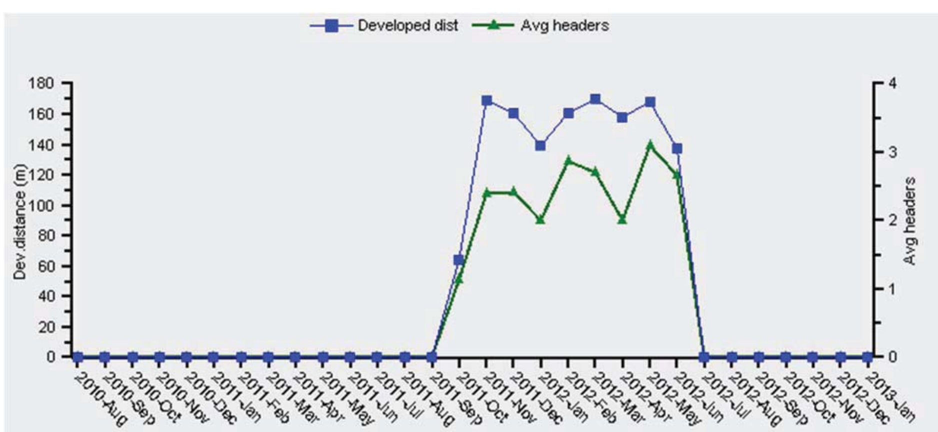


Table A.4.25. Equipment Utilization and Cycle Times on S2A Drives

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft_2_access_high_2 (31)	42.2 %	234		10/20/2011	1/31/2012	44.9	21.1
Shaft_2_access_high_3 (31)	45.0 %	118		1/31/2012	3/21/2012	46.0	21.5
Shaft_2_access_high_4 (31)	39.3 %	254		3/22/2012	6/27/2012	53.7	21.8
Shaft_2_access_high_5 (4)	39.0 %	185		8/1/2012	10/19/2012	438.5	21.7
Shaft_2_access_high_6 (4)	46.8 %	140		10/19/2012	12/20/2012	50.7	21.4
Shaft_2_access_low_1 (1)	39.8 %	200		8/16/2011	10/16/2011	56.5	21.2
Shaft_2_access_low_2 (31)	43.8 %	61		10/16/2011	11/12/2011	45.5	20.9
Shaft_2_access_low_3 (31)	42.2 %	242		11/12/2011	2/28/2012	45.7	20.3
Shaft_2_access_low_4 (31)	53.2 %	173		2/28/2012	5/10/2012	40.3	22.8
Shaft_2_access_low_5 (31)	35.7 %	93		5/10/2012	6/27/2012	55.3	22.0

Table A.4.26. Single Heading Development Rates with Jumbo Drilling and Bolting

	Working days	Distance (m)	Rate (m/d)
Jul-12	28	77	2.8
Aug-12	28	94	3.4
Sep-12	27	92	3.4
Oct-12	29	104	3.6
Nov-12	30	91	3.0
Dec-12	31	101	3.3
Total	211	559	3.2

Figure A.4.20. Monthly Development Meters and Available Headings of Mine Area 30

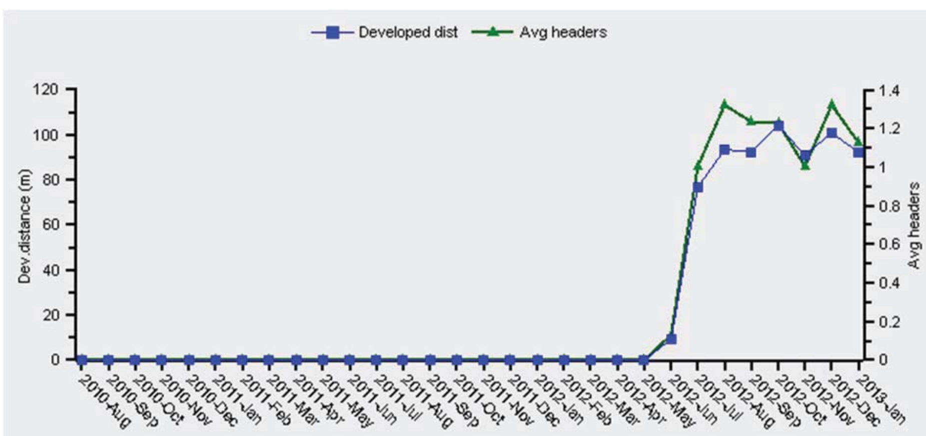


Table A.4.27. Equipment Utilization and Cycle Times on S2A Top Ramp

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_ramp_top_1 (30)	44.0 %	137		6/27/2012	8/24/2012	46.0	21.5
Shaft2_ramp_top_2 (30)	48.3 %	108		8/24/2012	10/1/2012	42.2	20.4
Shaft2_ramp_top_3 (30)	55.2 %	265		10/4/2012	1/5/2013	41.6	23.5

Test 11. 4.8m and 4m Drill Lengths Using Jumbo Drilling and Bolting

The drill length of multi-functional machines (face drilling and bolting) in Crew #2 was changed from Test 10's 4.8 m to 4 m. All other inputs were the same as those of the base case. The drilling and charging time required reduction since the drill length had decreased by 16.7%. However, the drilling time included the preparation, drilling and teardown times; and the drilling time could be further broken down to boom the positioning, boring, and delay times. Therefore, the influence on the total drilling time by reducing 0.8 meter drill length was minimal. Thus, the input of the drilling and charging times remained the same as those of the base case.

The development rate of the S2A drives dropped slightly to 5.08 meters per day, which was a 4.7% decrease from that of Test 10. For the single heading S2A top ramp, the development rate decreased by 6.25% to 3 meters per day as compared with that of Test 10. The results demonstrated that if the drill length were decreased by 16.7% to 4 m, the development rate would drop by 4.7% and 6.25%, respectively, for the dual and single headings. The shortened drill round's length would slow the single heading rate more than it would the dual headings.

Table A.4.28. Dual Heading Development Rates with a 4m Drill Length and Jumbo Bolting

	Working days	Distance (m)	Rate (m/d)
Nov-11	30	171	5.7
Dec-11	31	159	5.1
Jan-12	31	136	4.4
Feb-12	29	143	4.9
Mar-12	31	153	4.9
Apr-12	30	140	4.7
May-12	29	152	5.2
Jun-12	28	161	5.8
Total		1215.0	5.08

Figure A.4.21. Monthly Development Meters and Available Headings of Mine Area 31

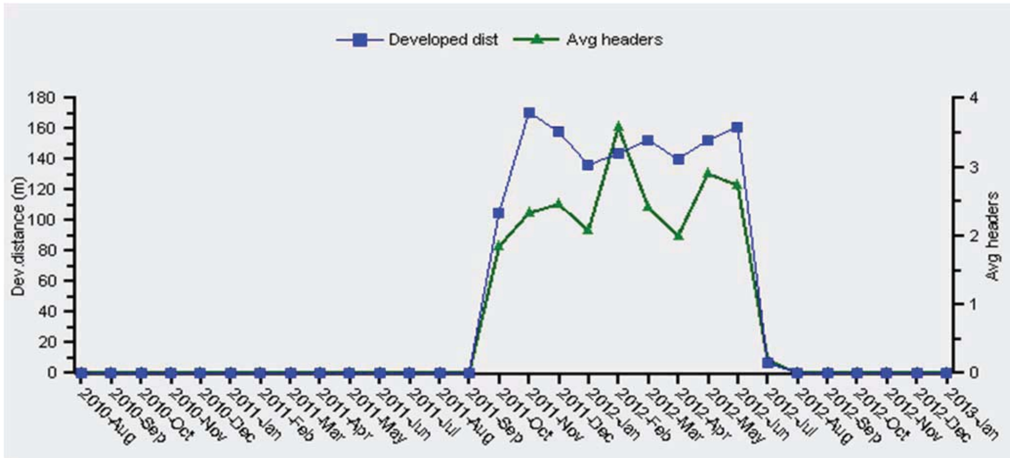


Table A.4.29. Equipment Utilization and Cycle Time on S2A Drives

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft_2_access_high_1 (1)	33.8 %	261		7/9/2011	10/8/2011	62.7	21.1
Shaft_2_access_high_2 (31)	40.8 %	234		10/8/2011	1/24/2012	42.0	17.9
Shaft_2_access_high_3 (31)	43.6 %	118		1/24/2012	3/20/2012	37.6	18.9
Shaft_2_access_high_4 (31)	35.4 %	254		3/20/2012	7/7/2012	47.7	18.0
Shaft_2_access_high_5 (4)	43.1 %	185		8/1/2012	10/15/2012	438.7	21.5
Shaft_2_access_high_6 (4)	46.9 %	140		10/15/2012	12/14/2012	42.9	19.9
Shaft_2_access_low_1 (1)	36.7 %	200		7/30/2011	10/6/2011	59.4	20.5
Shaft_2_access_low_2 (31)	52.0 %	61		10/6/2011	11/5/2011	39.3	17.9
Shaft_2_access_low_3 (31)	48.5 %	242		11/5/2011	2/17/2012	33.3	18.0
Shaft_2_access_low_4 (31)	41.6 %	173		2/17/2012	5/14/2012	45.2	19.9
Shaft_2_access_low_5 (31)	41.6 %	93		5/14/2012	6/28/2012	42.2	19.0
Shaft_2_access_low_6 (4)	56.4 %	363		8/1/2012	12/20/2012	137.4	21.0

Table A.4.30. Single Heading Development Rates with 4m Drill Length and Jumbo Bolting

	Working days	Distance (m)	Rate (m/d)
Jul-12	28	70.0	2.5
Aug-12	28	87	3.1
Sep-12	27	74	2.7
Oct-12	29	93	3.2
Nov-12	30	107	3.6
Dec-12	31	90	2.9
Total		521	3.01

Figure A.4.22. Monthly Development Meters and Available Headings of Mine Area 30

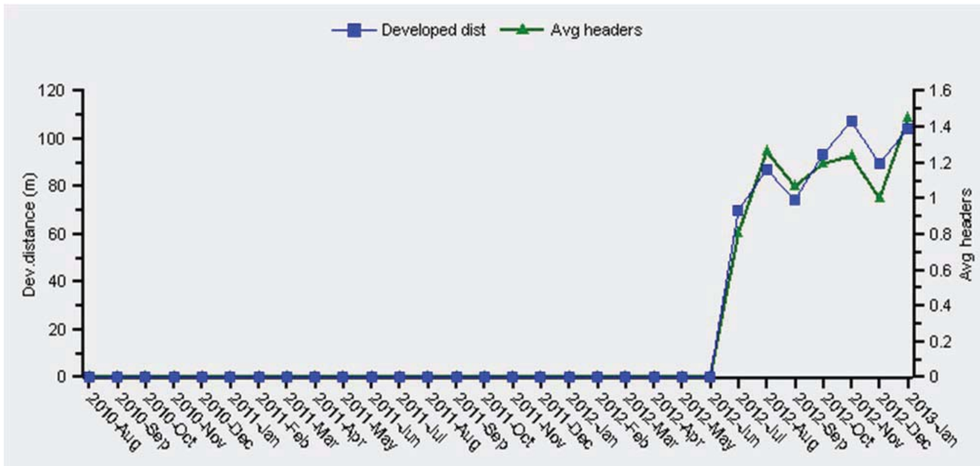


Table A.4.31. Equipment Utilization and Cycle Times on S2A Top Ramp

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_ramp_top_1 (30)	51.6 %	137		7/7/2012	8/30/2012	34.5	17.9
Shaft2_ramp_top_2 (30)	44.9 %	108		8/30/2012	10/14/2012	40.0	17.8
Shaft2_ramp_top_3 (30)	53.1 %	265		10/14/2012	1/17/2013	33.7	18.2

Test 12. Haulage Drives

The haulage drives' profile was 5.8mHx5.8mW by the MS2 design but later it was modified to 6.0mHx6.1mW in the MS3. This test was used to determine the development rates on new MS3 haulage drives. Most haulage drives were located in the footprint area, although about 200 m were outside that area (mine asbuilt and crusher drives). However, the tested haulage drives were all in the footprint area and followed the footprint ground support regime, which required extensive cable bolting and two-layer shotcreting. One development crew with an independent cable bolter was designated to work in the tested mine area. Four haulage drives were open (two in the north, two in the south). Cable bolting and the second layer of shotcrete were campaigned at 50 m intervals. The development rate of the haulage drives using the same footprint campaigned support regime was 4.17 m/d.

Figure A.4.23. Tested Haulage Drives

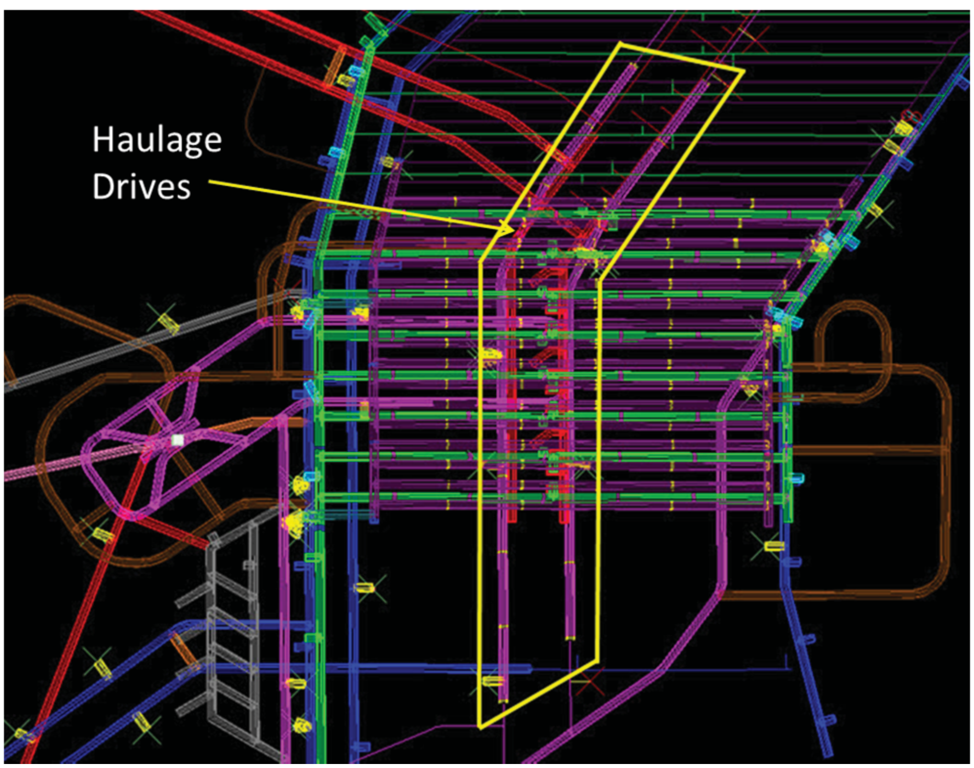


Table A.4.32. Durations and Development Rates on Haulage Drives

	Start	End	Working days	Distance (m)	Rate (m/d)
Haulage_test_1	01/01/2015	02/08/2015	207	215	1.04
Haulage_test_2	01/01/2015	24/07/2015	198	215	1.09
Haulage_test_3	01/01/2015	18/08/2015	220	230	1.05
Haulage_test_4	01/01/2015	12/08/2015	214	215	1.00
Haulage Drive Avg					**4.17**

Table A.4.33. Equipment Utilization and Cycle Times of Campaigned Support

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
CS_haulage_1 - 1 (88)	42.5 %	50		2/2/2015	2/19/2015	403.2	
CS_haulage_11 - 1 (88)	38.7 %	50		2/3/2015	2/24/2015	507.9	
CS_haulage_12 - 1 (88)	33.2 %	50		2/6/2015	3/3/2015	598.1	
CS_haulage_13 - 1 (88)	45.6 %	50		3/28/2015	4/13/2015	384.3	
CS_haulage_15 - 1 (88)	75.5 %	50		6/2/2015	6/14/2015	289.7	
CS_haulage_16 - 1 (88)	48.7 %	50		6/2/2015	6/21/2015	450.1	
CS_haulage_17 - 1 (88)	58.5 %	50		8/3/2015	8/18/2015	353.2	
CS_haulage_18 - 1 (88)	54.1 %	50		7/27/2015	8/12/2015	390.4	
CS_haulage_2 - 1 (88)	70.5 %	50		2/1/2015	2/13/2015	278.9	
CS_haulage_3 - 1 (88)	46.7 %	50		4/1/2015	4/19/2015	427.7	
CS_haulage_4 - 1 (88)	61.1 %	50		3/25/2015	4/7/2015	313.2	
CS_haulage_5 - 1 (88)	56.9 %	50		5/22/2015	6/6/2015	367.0	
CS_haulage_6 - 1 (88)	78.1 %	50		5/20/2015	5/31/2015	277.7	
CS_haulage_7 - 1 (88)	63.3 %	50		7/17/2015	8/2/2015	366.8	
CS_haulage_8 - 1 (88)	60.9 %	50		7/10/2015	7/24/2015	337.6	
CS_haulge_14 - 1 (88)	81.1 %	50		4/15/2015	4/26/2015	266.9	

Table A.4.34. Equipment Utilization and Cycle Times of Haulage Drives with In-cycle Support

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Haulage_test_1 (14)	36.3 %	50		1/1/2015	2/2/2015	432.4	26.0
Haulage_test_11 (14)	36.3 %	50		1/1/2015	2/3/2015	312.0	24.7
Haulage_test_12 (14)	35.2 %	51		1/2/2015	2/6/2015	319.5	26.8
Haulage_test_13 (14)	32.4 %	50		2/24/2015	3/28/2015	78.0	24.9
Haulage_test_14 (14)	28.1 %	50		3/3/2015	4/15/2015	100.3	25.9
Haulage_test_15 - 1 (14)	24.2 %	50		4/13/2015	6/2/2015	107.9	26.1
Haulage_test_16 (14)	27.1 %	50		4/26/2015	6/2/2015	90.9	25.0
Haulage_test_17 (14)	20.8 %	50		6/14/2015	8/3/2015	133.1	26.1
Haulage_test_18 - 1 (14)	30.3 %	50		6/21/2015	7/27/2015	80.5	25.7
Haulage_test_2 (14)	38.1 %	50		1/1/2015	2/1/2015	308.8	26.0
Haulage_test_3 (14)	31.6 %	50		2/19/2015	4/1/2015	83.0	25.3
Haulage_test_4 (14)	28.0 %	50		2/13/2015	3/25/2015	106.2	24.8
Haulage_test_5 (14)	32.4 %	50		4/19/2015	5/22/2015	78.5	25.8
Haulage_test_6 (14)	33.7 %	50		4/7/2015	5/20/2015	77.9	25.0
Haulage_test_7 (14)	24.3 %	50		6/8/2015	7/17/2015	133.7	25.8
Haulage_test_8 - 1 (14)	28.0 %	50		5/31/2015	7/10/2015	94.4	26.7

Test 13. Conveyor Drift off the Footprint

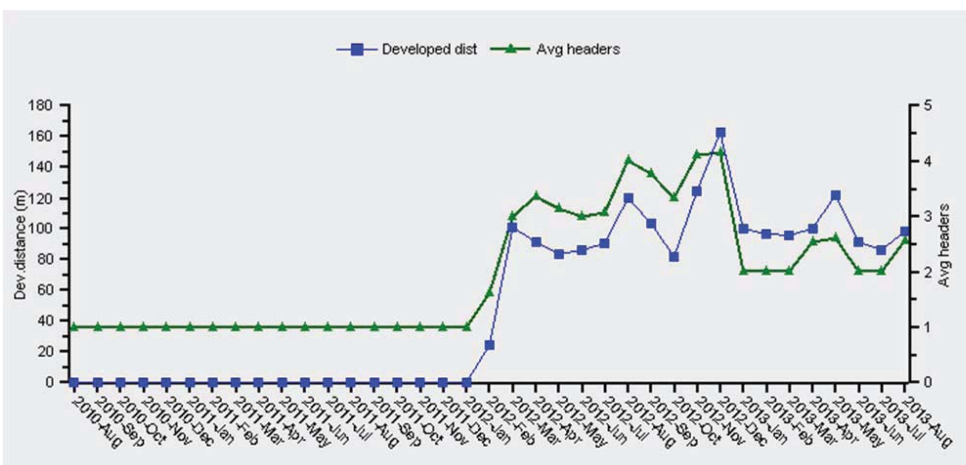
The development rate of conveyor type drifts was tested on the two primary headings, MCD and SD (Mine Area 8) and two secondary headings, S2ID (Mine Area 37). The primary headings were set to high priority and the secondary headings to low priority. These four headings were modified to follow the “S” type profile (6.8mWx5.5mH) as conveyer drifts. Conveyor drifts needed regular in-cycle cable bolting for both Types 1 and 2 ground conditions. The four headings were initiated at the same time and linked as start-to-start triggers. All of these headings were assumed to be 90% Type 1 and 10% Type 2 ground support based on the MS3 drift design. One development crew was assigned to work on these mine areas (8 and 37), but it shared one cable bolter with another development crew.

The development rate of the two primary headings was 3.5 m/d, and that of the secondary headings was 2.54 m/d. Therefore, the total development rate was 6.04 m/d per crew. Compared with the development rate of 8.4 m/d in Test 5, the rate had decreased by 28% as the drift’s dimensions increase from “I” type to “S” type and more ground supports were required.

Table A.4.35. Development Rate of Tested Conveyor Drifts

		Dual Heading Area 8		Secondary Heading Area 37		
	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
Mar-12	31	101	3.3	77	2.5	5.7
Apr-12	30	91	3.0	72	2.4	5.4
May-12	29	83	2.9	77	2.7	5.5
Jun-12	28	86	3.1	62	2.2	5.3
Jul-12	28	91	3.3	86	3.1	6.3
Aug-12	28	120	4.3	67	2.4	6.7
Sep-12	27	103	3.8	72	2.7	6.5
Oct-12	29	82	2.8	53	1.8	4.7
Nov-12	30	124	4.1	67	2.2	6.4
Dec-12	31	163	5.3	77	2.5	7.7
Jan-13	31	100	3.2	91	2.9	6.2
Feb-13	28	96	3.4	76	2.7	6.1
Mar-13	31	96	3.1	92	3.0	6.1
Apr-13	30	100	3.3	82	2.7	6.1
May-13	29	122.0	4.2	72	2.5	6.7
Jun-13	28	91	3.3	63	2.3	5.5
Jul-13	28	86	3.1	77	2.8	5.8
Aug-13	28	98	3.5	67	2.4	5.9
Total		1833	3.50	1330.0	2.54	6.04

Figure A.4.24. Monthly Development Meters and Available Headings of Mine Area 8 and 37



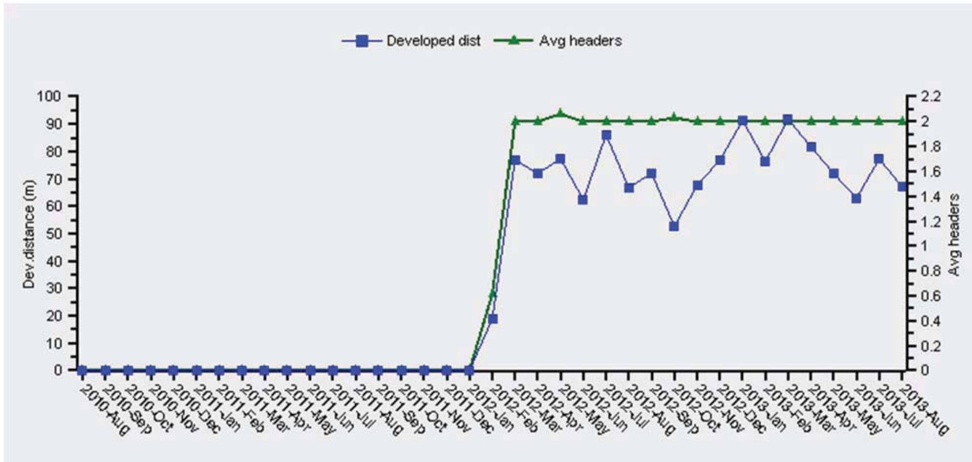


Table A.4.36. Equipment Utilization and Cycle Times on Tested Conveyor Drifts

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	39.5 %	369		4/3/2012	12/12/2012	78.5	33.6
Conveyor_drive_2 (8)	35.9 %	204		8/1/2010	2/24/2013	1,381.8	32.8
Conveyor_drive_3 (8)	41.9 %	159		2/24/2013	6/12/2013	76.5	32.8
Conveyor_drive_4 (8)	36.9 %	108	54	6/12/2013		90.5	32.0
Conveyor_drive_8 (11)	40.9 %	25	152	8/23/2013		70.4	29.6

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	40.5 %	156		2/21/2012	6/11/2012	79.8	32.7
Service_drive_2 (8)	42.7 %	149		6/11/2012	9/26/2012	78.4	33.1
Service_drive_3 (8)	41.7 %	233		9/26/2012	3/5/2013	81.0	33.4
Service_drive_4 (8)	44.5 %	232		3/5/2013	8/1/2013	73.3	32.8
Service_drive_5 (8)	42.0 %	45	5	8/1/2013		69.0	34.0

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_main_vent_drive_1 (37)	35.8 %	210		2/21/2012	8/5/2012	90.0	32.2
Shaft2_main_vent_drive_10 (37)	26.8 %	34	26	7/25/2013		122.8	33.1
Shaft2_main_vent_drive_2 (37)	33.1 %	191		2/22/2012	7/29/2012	94.5	31.4
Shaft2_main_vent_drive_3 (37)	30.8 %	175		8/5/2012	1/4/2013	111.2	31.4
Shaft2_main_vent_drive_4 (37)	36.1 %	172		7/29/2012	1/9/2013	95.1	32.3
Shaft2_main_vent_drive_5 (37)	40.6 %	149		1/4/2013	4/22/2013	80.9	33.4
Shaft2_main_vent_drive_6 (37)	42.1 %	155		1/9/2013	4/29/2013	82.5	34.3
Shaft2_main_vent_drive_7 (37)	34.2 %	163	1	4/22/2013		93.4	32.5
Shaft2_main_vent_drive_8 (37)	33.7 %	100		4/29/2013	7/25/2013	94.9	34.9

Test 14. Exhaust Drift off Footprint

The development rate of the exhaust drift was tested on the MCD and SD (primary headings) and two S2ID (secondary headings). All of these headings were modified to those of the exhaust type drift, “K” profile (6.0mWx7.0mH). These four headings began to be excavated simultaneously by one development crew; however, the group shared a cable bolter with another crew. The headings were assumed to follow 90% Type 1 and 10% Type 2 ground support based the MS3 drift design.

Development rate of the two primary headings was 3.83 m/d and that of the two secondary headings was 2.52 m/d. Therefore, the total rate was 6.23 m/d per crew. This rate was about 3.1% higher than the “S” type conveyer drift, even though the heading dimension was greater for the “K” type heading. This resulted from the in-cycle cable bolting on the “S” type heading, which slowed its development rates.

Table A.4.37. Development Rate of Tested Exhaust Drifts

		Dual Heading Area 8		Secondary Heading Area 37		
	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
Mar-12	31	101	3.3	77	2.5	5.7
Apr-12	30	108	3.6	72	2.4	6.0
May-12	29	105	3.6	87	3.0	6.6
Jun-12	28	95	3.4	86	3.1	6.5
Jul-12	28	135	4.8	76	2.7	7.5
Aug-12	28	114	4.1	62	2.2	6.3
Sep-12	27	115	4.3	62	2.3	6.6
Oct-12	29	145	5.0	61	2.1	7.1
Nov-12	30	153	5.1	72	2.4	7.5
Dec-12	31	106	3.4	101	3.3	6.7
Jan-13	31	106	3.4	91	2.9	6.4
Feb-13	28	96	3.4	72	2.6	6.0
Mar-13	31	142	4.6	72	2.3	6.9
Apr-13	30	101	3.4	82	2.7	6.1
May-13	29	96.0	3.3	58	2.0	5.3
Jun-13	28	79	2.8	53	1.9	4.7
Jul-13	28	105	3.8	67	2.4	6.1
Aug-13	28	106	3.8	72	2.6	6.4
Total		2008	3.83	1323.0	2.52	6.36

Figure A.4.25. Monthly Development Meters and Available Headings of Mine Area 8 and 37

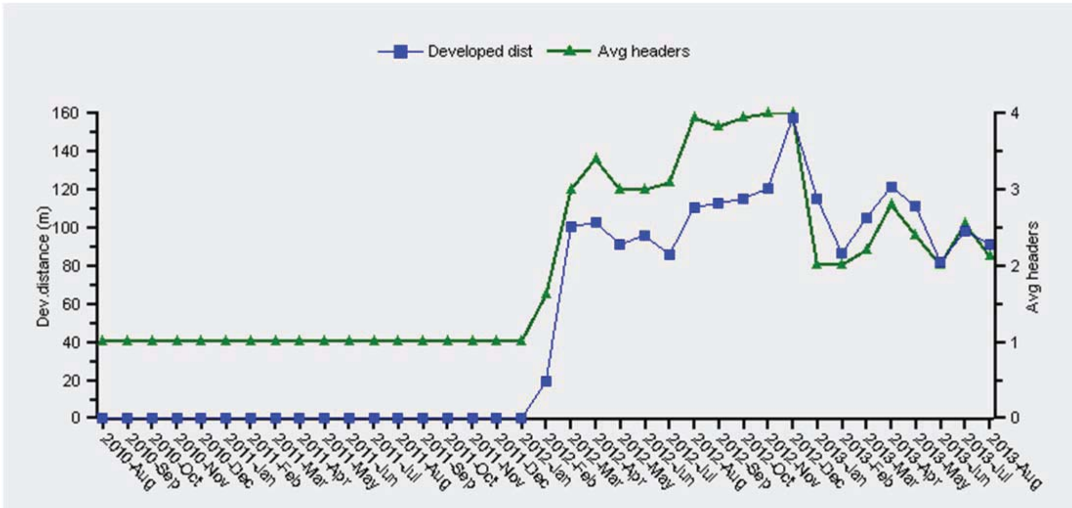


Table A.4.38. Equipment Utilization and Cycle Times on Tested Exhaust Drifts

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	39.5 %	369		4/3/2012	12/12/2012	78.5	33.6
Conveyor_drive_2 (8)	35.9 %	204		8/1/2010	2/24/2013	1,381.8	32.8
Conveyor_drive_3 (8)	41.9 %	159		2/24/2013	6/12/2013	76.5	32.8
Conveyor_drive_4 (8)	36.9 %	108	54	6/12/2013		90.5	32.0
Conveyor_drive_8 (11)	40.9 %	25	152	8/23/2013		70.4	29.6

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	40.5 %	156		2/21/2012	6/11/2012	79.8	32.7
Service_drive_2 (8)	42.7 %	149		6/11/2012	9/26/2012	78.4	33.1
Service_drive_3 (8)	41.7 %	233		9/26/2012	3/5/2013	81.0	33.4
Service_drive_4 (8)	44.5 %	232		3/5/2013	8/1/2013	73.3	32.8
Service_drive_5 (8)	42.0 %	45	5	8/1/2013		69.0	34.0

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_main_vent_drive_1 (37)	35.8 %	210		2/21/2012	8/5/2012	90.0	32.2
Shaft2_main_vent_drive_10 (37)	26.8 %	34	26	7/25/2013		122.8	33.1
Shaft2_main_vent_drive_2 (37)	33.1 %	191		2/22/2012	7/29/2012	94.5	31.4
Shaft2_main_vent_drive_3 (37)	30.8 %	175		8/5/2012	1/4/2013	111.2	31.4
Shaft2_main_vent_drive_4 (37)	36.1 %	172		7/29/2012	1/9/2013	95.1	32.3
Shaft2_main_vent_drive_5 (37)	40.6 %	149		1/4/2013	4/22/2013	80.9	33.4
Shaft2_main_vent_drive_6 (37)	42.1 %	155		1/9/2013	4/29/2013	82.5	34.3
Shaft2_main_vent_drive_7 (37)	34.2 %	163	1	4/22/2013		93.4	32.5
Shaft2_main_vent_drive_8 (37)	33.7 %	100		4/29/2013	7/25/2013	94.9	34.9

Test 15. Conveyor Drift on Cave Zone

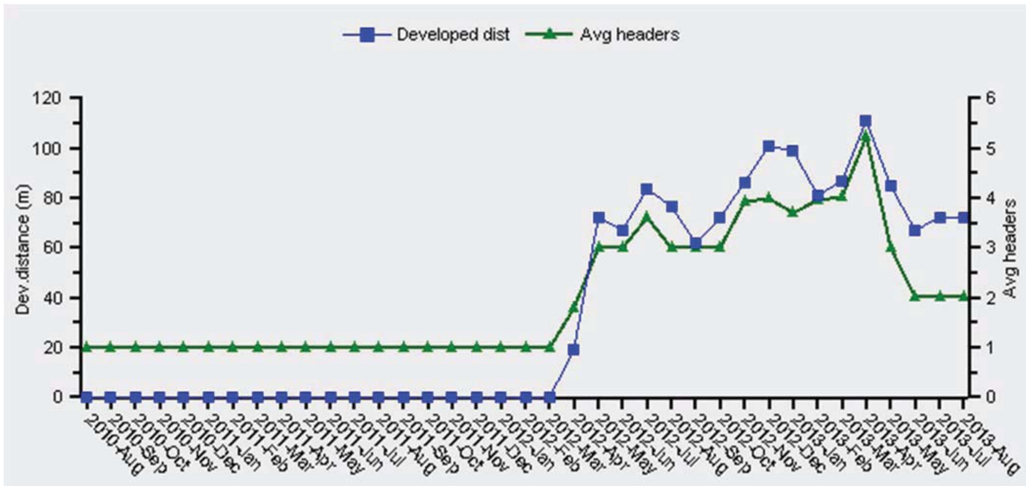
Some portions of the conveyor drifts close to the footprint are were located in the Cave Zone. This test assumed that the conveyor drift in the Cave Zone would need 100% Type 2 ground support of "S" drift design. The development rate was tested on the same drifts as those of Test 13. All four headings were modified to "S" type profiles (6.8mWx5.5mH). Type 2 ground support on the conveyor drift needed more rock bolts and an additional layer of fiber shotcrete as compared with Type 1. One development crew was assigned to work on these headings, although it shared a cable bolter with another development crew.

The development rate of the two primary headings was 2.8 m/d and of the two secondary headings was 1.84 m/d. The total rate was 4.64 m/d per crew. Compared with the development rate of 6.04 m/d in Test 13, the rate decreased by 23% since more ground supports were employed for that type than for Type 2.

Table A.4.39. Development Rates of Tested Conveyor Drift on Cave Zone

		Dual Heading Area 8		Secondary Heading Area 37		
	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
May-12	29	72	2.5	62	2.1	4.6
Jun-12	28	67	2.4	58	2.1	4.5
Jul-12	28	84	3.0	57	2.0	5.0
Aug-12	28	77	2.8	58	2.1	4.8
Sep-12	27	62	2.3	53	2.0	4.3
Oct-12	29	72	2.5	62	2.1	4.6
Nov-12	30	86	2.9	66	2.2	5.1
Dec-12	31	101	3.3	53	1.7	5.0
Jan-13	31	99	3.2	58	1.9	5.1
Feb-13	28	81	2.9	48	1.7	4.6
Mar-13	31	87	2.8	48	1.5	4.4
Apr-13	30	111	3.7	24	0.8	4.5
May-13	29	85	2.9	38	1.3	4.2
Jun-13	28	67	2.4	63	2.3	4.6
Jul-13	28	72	2.6	53	1.9	4.5
Aug-13	28	72	2.6	53	1.9	4.5
Total		1295	2.80	854.0	1.84	4.64

Figure A.4.26. Monthly Development Meters and Available Headings of Mine Area 8 and 37



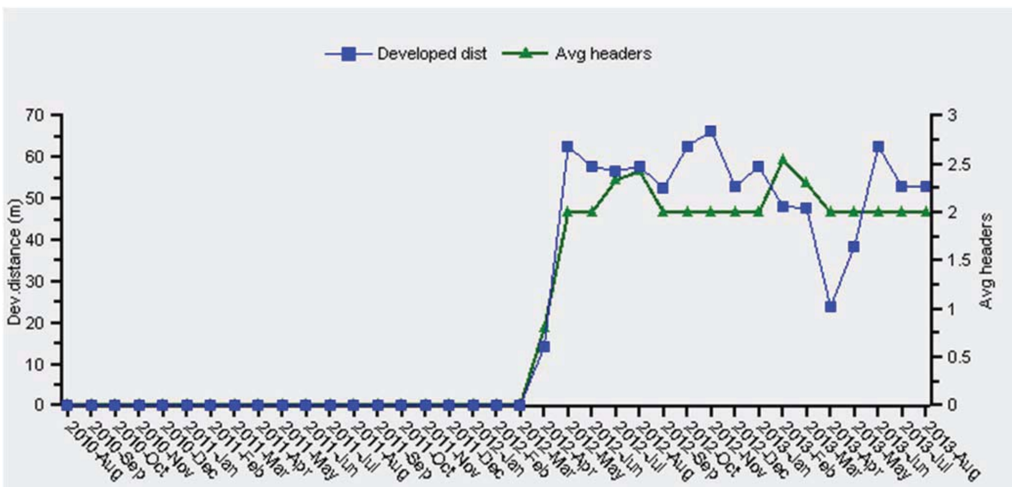


Table A.4.40. Equipment Utilization and Cycle Time in Tested Conveyor Drifts on Cave Zone

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	35.6 %	369		6/23/2012	4/16/2013	122.0	43.1
Conveyor_drive_2 (8)	33.2 %	205		8/1/2010	8/10/2013	1,612.5	43.2
Conveyor_drive_3 (8)	48.9 %	32	20	8/10/2013		84.7	41.5

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	40.4 %	156		4/19/2012	9/8/2012	103.2	41.9
Service_drive_2 (8)	43.6 %	149		9/8/2012	1/12/2013	94.3	40.9
Service_drive_3 (8)	40.6 %	233		1/12/2013	8/16/2013	106.6	43.2
Service_drive_4 (8)	38.4 %	18	64	8/16/2013		110.8	40.7

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_main_vent_drive_1 (37)	32.2 %	210		4/20/2012	11/20/2012	128.8	41.6
Shaft2_main_vent_drive_2 (37)	34.2 %	191		4/20/2012	11/6/2012	119.5	39.8
Shaft2_main_vent_drive_3 (37)	24.7 %	175		11/20/2012	7/2/2013	189.4	42.5
Shaft2_main_vent_drive_4 (37)	32.4 %	172		11/6/2012	6/20/2013	131.4	41.0
Shaft2_main_vent_drive_5 (37)	30.7 %	52	16	7/2/2013		140.0	43.9
Shaft2_main_vent_drive_6 (37)	32.7 %	67	2	6/20/2013		127.5	41.5

Test 16. Exhaust Drift on Cave Zone

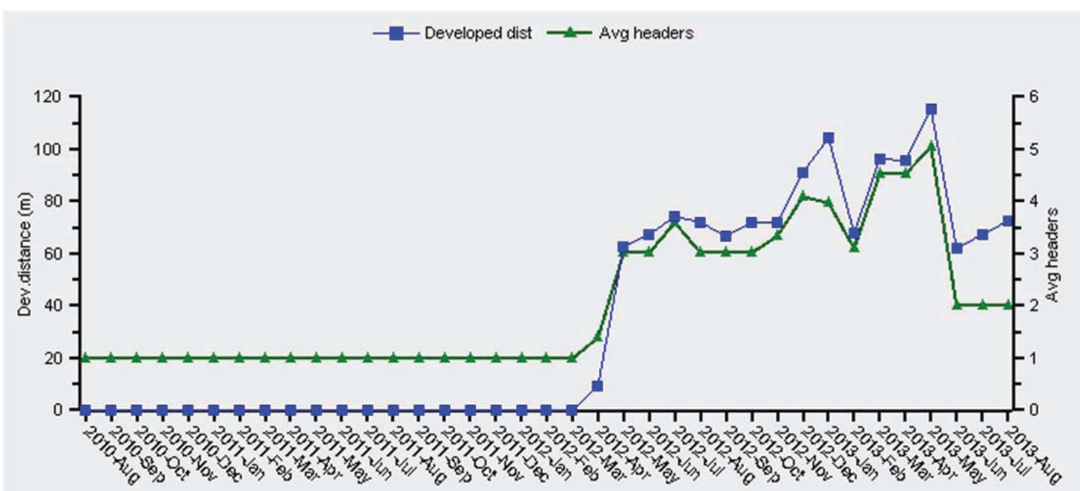
Some portions of exhaust drifts close to the footprint are were located in the Cave Zone. This test assumed that the exhaust drift in the Cave Zone would use 100% Type 2 ground support. The development rate was tested under the same headings as Test 14. All these headings were modified to the “K” type profile (6.0mWx7.0mH). Type 2 ground support of the exhaust drift required cable bolts, extra rock bolts and an additional layer of fiber shotcrete as compared with Type 1. One development crew was assigned to work on these headings, although it shared a cable bolter with another development crew.

The Development rate of the two primary headings was 2.4 m/d and that of the secondary headings was 1.63 m/d. The total rate was 4.02 m/d per crew. This rate had decreased by 35.5% as compared with that of Test 14 because the requirements of extra ground support of Type 1 as compared with Type 2. It also dropped 13.4% from that of Test 15 because its profile was larger and ground support was more intensive than that of the conveyor drift.

Table A.4.41. Development Rates of Tested Exhaust Drift on Cave Zone

		Dual Heading Area 8		Secondary Heading Area 37		
	Working days	Distance (m)	Rate (m/d)	Distance (m)	Rate (m/d)	Total
May-12	29	62	2.1	53	1.8	4.0
Jun-12	28	67	2.4	62	2.2	4.6
Jul-12	28	74	2.6	47	1.7	4.3
Aug-12	28	72	2.6	58	2.1	4.6
Sep-12	27	67	2.5	57	2.1	4.6
Oct-12	29	72	2.5	53	1.8	4.3
Nov-12	30	72	2.4	57	1.9	4.3
Dec-12	31	91	2.9	48	1.5	4.5
Jan-13	31	104	3.4	53	1.7	5.1
Feb-13	28	68	2.4	58	2.1	4.5
Mar-13	31	96	3.1	52	1.7	4.8
Apr-13	30	96	3.2	53	1.8	5.0
May-13	29	115	4.0	38	1.3	5.3
Jun-13	28	62	2.2	48	1.7	3.9
Jul-13	28	67	2.4	62	2.2	4.6
Aug-13	28	72	2.6	53	1.9	4.5
Total		1257	2.40	852.0	1.63	4.02

Figure A.4.27. Monthly Development Meters and Available Headings of Mine Area 8 and 37



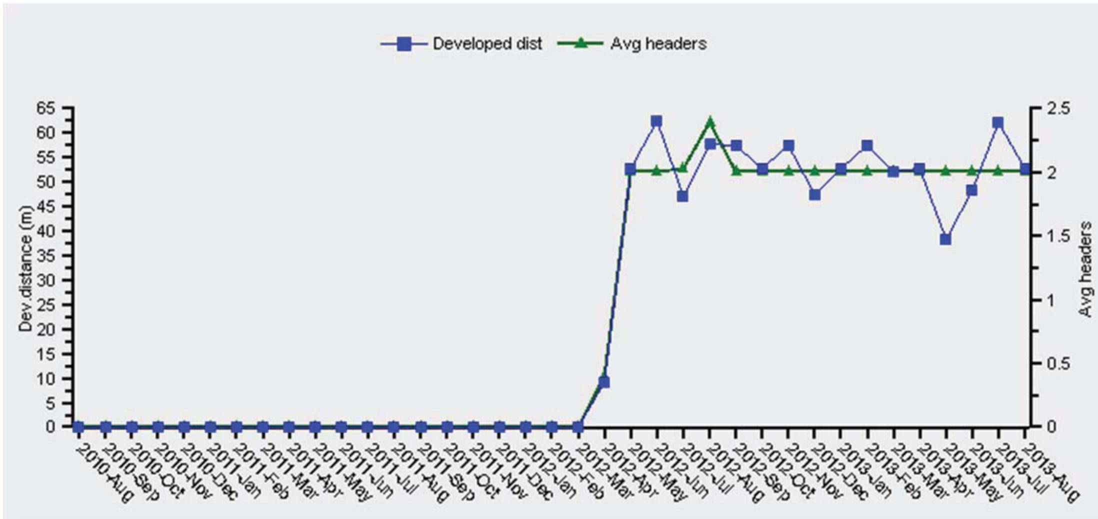


Table A.4.42. Equipment Utilization and Cycle Times on Tested Exhaust Drift on Cave Zone

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Conveyor_drive_1 (8)	40.0 %	370		7/2/2012	5/6/2013	116.1	46.9
Conveyor_drive_2 (8)	34.8 %	205		8/1/2010	8/29/2013	1,651.9	48.0
Conveyor_drive_3 (8)	48.6 %	2	50	8/29/2013		0.0	0.0

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Service_drive_1 (8)	43.2 %	156		4/25/2012	9/19/2012	106.3	46.2
Service_drive_2 (8)	44.9 %	149		9/19/2012	1/27/2013	98.8	43.8
Service_drive_3 (8)	41.6 %	232	1	1/27/2013		108.8	44.9

Sections	Utilization (%)	Completed (m)	Undone (m)	Start date	End date	Gross cycle time (h)	Cycle time (h)
Shaft2_main_vent_drive_1 (37)	33.8 %	210		4/25/2012	12/5/2012	129.6	44.6
Shaft2_main_vent_drive_2 (37)	35.3 %	191		4/25/2012	11/28/2012	129.4	45.8
Shaft2_main_vent_drive_3 (37)	32.2 %	175		12/5/2012	7/14/2013	143.9	45.2
Shaft2_main_vent_drive_4 (37)	34.0 %	172		11/28/2012	6/20/2013	130.5	45.2
Shaft2_main_vent_drive_5 (37)	36.1 %	46	21	7/14/2013		122.7	44.3
Shaft2_main_vent_drive_6 (37)	35.0 %	67	2	6/20/2013		131.4	46.2

Test 17. Exhaust Drift on Footprint

Exhaust drifts in the footprint area were different from those off the footprint, and they were designed as following the 5.0mWx5.5mH profile. The exhaust drifts (of ventilation level) were located at the center of the footprint. Due to poor ground conditions and high stress in those areas, more extensive ground support was needed for exhaust drift on footprint than that of the regular 5.0mWx5.5mH headings. To simplify the test model, the four extraction drives in the footprint area were modified to the 5.0mWx5.5mH profile and the ground support regime was altered to follow that of the footprint haulage and extraction drives. The assumptions were: for Type 1 ground support double layers of shotcrete with mesh, 1 m x 1 m grid rock bolts, and 2 m x 2 m grid cable bolts; and for Type 2, the additional mesh straps and cable bolts were changed to a 1 m x 1 m grid. This test assumed that exhaust in the footprint used 80% Type 1 and 20% Type 2 ground support. One development crew was assigned to work on these four made-up exhaust drives with one independent cable bolter.

The total development rate of the four headings was 5.26 m/ per development crew. Compared with the development rate of 5.7 m/d in Test 8 extraction drives, the rate had decreased by 8.4% as the heading profile increased from 4.5mWx4.5mH to 5.0mWx5.5mH.

Table A.4.43. Development Rates of Tested Exhaust Drift in the Footprint Area

	Start	End	Working days	Distance (m)	Rate (m/d)
Exhaust_test_1	03/01/2014	16/09/2014	243	323	1.33
Exhaust_test_2	03/01/2014	22/09/2014	249	323	1.30
Exhaust_test_3	03/01/2014	03/09/2014	233	325	1.39
Exhaust_test_4	03/01/2014	06/10/2014	263	325	1.24
Exhaust Drift Avg					**5.26**

Test 18. Milestone 3 (MS3) Tradeoff Study

This test was experimented on in the MS3 model. The simulation period was from January 1, 2012 to November 30, 2014. The mine shutdown period was scheduled to be from March 8, 2012 to the end of August, 2012. This model was validated in the nine-week period from January 1 to March 4, 2012 by comparing the actual development meters with the OT site. The results were more closely correlated with those of the planned meters from the OT site. The ramp-up and equipment buildup of the development crews were based on MS3 assumptions and ventilation models. The muck handling system was modified to follow that of the MS3 mine design. In this tradeoff test, different scenarios of fleet build-up were experimented on to examine the influences of additional resources on development performances.

Three scenarios were tested and compared with the base case assumptions of the MS3 mine design and schedules.

- Adding a truck to the trucking fleet after the mine's shutdown (on March 8, 2012)
- Adding a separate cable bolter to Crews 4 and 6 (Crews 3,4,5,6 will have independent cable bolters)
- Adding a truck to the trucking fleet and adding two cable bolters to Crews 4 and 6, respectively

The results were selected from June, 2013 to December, 2013 when the additional development equipment displayed the greatest improvement in development rates. Generally, over the selected simulation period, (June 1, 2013 to December 31, 2013), the development rate of Scenario 1 increased by 7%, Scenario 2 by 6%, and Scenario 3 by 13% as compared with the base case.

The utilization of trucks was plotted by the simulation models (Figure A.4.29. and Figure A.4.30.). The utilization in the model was defined by work time (the yellow bar) plus travel time (the purple line) divided by the total time. In the base case test, from November, 2013 to May, 2014 the utilization of Truck TH550-01 was very high, ranging from 60% to 85%. In Scenario 1, however, by adding an extra truck, the utilization of Truck TH550-01 was reduced by approximately 15% as compared with the base case, which dropped its utilization to a reasonable level. .

Table A.4.44. Monthly Development Meters of MS3 Equipment Fleet Tradeoff Tests

m/d Date	SimMine Base Case	+1 Truck*	+2 Cabletec*	+1 Truck and 2 Cabletec
Jun-13	358	403	358	378
Jul-13	414	461	414	442
Aug-13	496	490	505	510
Sep-13	494	501	517	554
Oct-13	642	747	726	803
Nov-13	822	902	883	989
Dec-13	972	988	1,044	1,054
Total	4198	4492	4447	4730
Improve %		7%	6%	13%

Figure A.4.28. Cumulative Development Meters of MS3 Equipment Fleet Tradeoff Tests

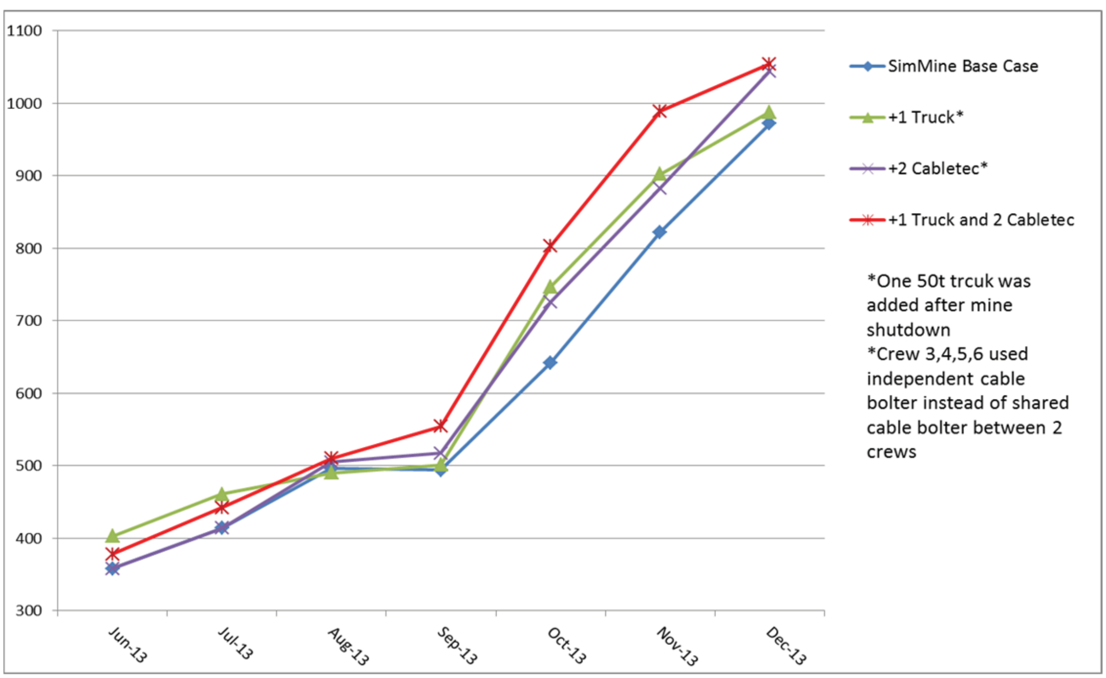


Figure A.4.29. Base Case Simulated Utilization of Truck TH550-01

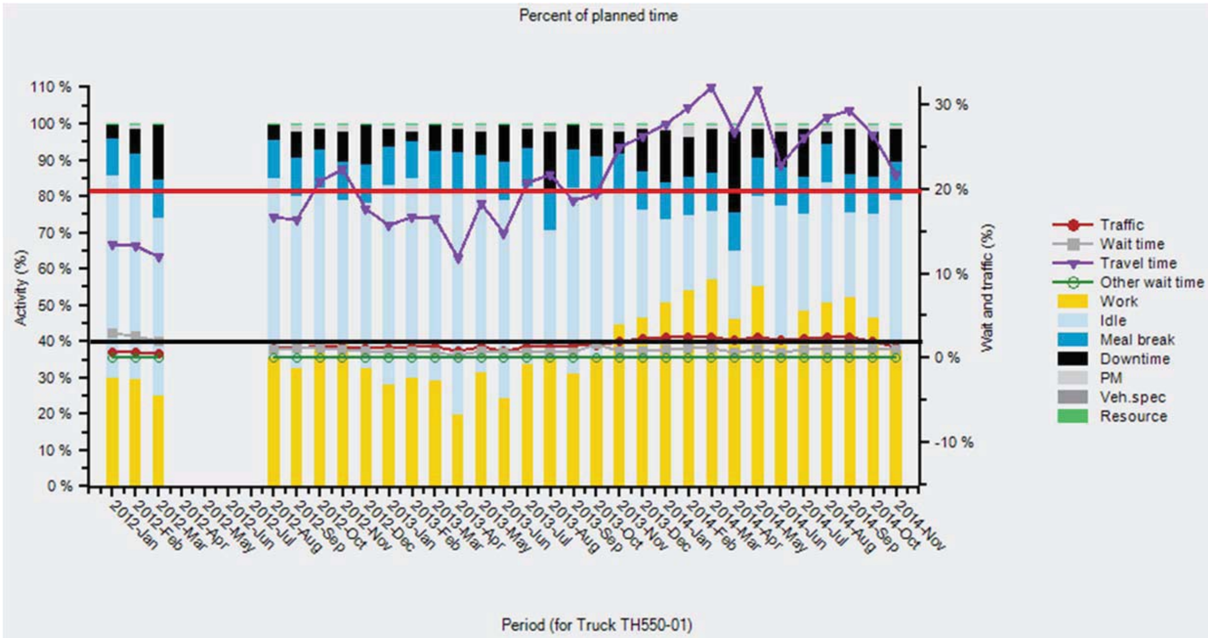


Figure A.4.30. Scenario 1 Simulated Utilization of Truck TH550-01