MODELLING THE IMPACT OF CHANGE IN SYSTEMS AND TECHNOLOGY IN A SURFACE MINING ENVIRONMENT WITH SYSTEM DYNAMICS

by

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DECLERATION OF ORIGINALITY

I hereby declare that this project is my own unaided work and I have referenced all the sources I have used. It is being submitted in fulfilment of the requirements for the degree Doctor of Philosophy at the University of Pretoria, Pretoria. It has not been submitted before for any degree or examination at any other university. This document represents my own opinion and interpretation of information received from research. I thus accept the rules of assessment of the University and the consequences of transgressing them.

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EXECUTIVE SUMMARY

Methods for estimating the impact of disruptive technologies in a surface mining environment are explored that will enable mining managers to justify the introduced technology. This was dependent on the estimation of time-based productivity and costs. Each sub process in mining operations works toward a long-term goal of minimized costs and maximized production. These sub processes are intricately dependent on each other. Due to high inter-dependency of these processes the impact is not easily captured for the total mine with traditional spread sheet modelling. The dynamic nature of such a mining environment required a dynamic modelling approach.

The analysis required all relevant criteria that can be obtained in a mine to be defined and was represented either as raw data inputs or as statistical parameters describing the behaviour of such data. The dynamic modelling heavily depended on these statistical parameters, since using raw data would further complicate any predictive analysis and modelling. In addition, cyclic mining processes and dependencies required a high-level description of the “as is” situations.

The research outcomes were dependent on a good understanding of variability of the processes that are normally due to the limitations of the mining machinery, the quality of the work impacted by people interacting with such machinery, mine structure and the sequence of the unit processes. The simulation also depended on many attributes, constants, auxiliaries and tables of data to describe behaviour from typical mining operations in similar geotechnical environments. These are defined adequately to model the mining environment.

The steps followed in the thesis are based on a top-to-bottom-to-top dynamic systems modelling method (DSM) as discussed in Chapter 2.The system dynamics simulation (SD) tool used is called Vensim, a widely used SD modelling tool. The justification of the method as well as ways of formulating the model structure required understanding of the system dynamics modelling and quantification techniques and is discussed in detail in chapter three. Most of the inputs used in the creation of the model comes from mining engineering literature of the typical input parameters and user experience based inputs. It required review of a large number of publications including system dynamics modelling based mining literature. This was discussed in detail in chapter four. The technique of testing the validity of the model is explained in chapter 5. The model testing methods were implemented from start to finish as the system dynamics model was built from the simple to the complex. The total mining value chain of a typical surface mine was discussed in detail to determine all quantifiable endogenous or exogenous variables. The causalities and relationship are

summarized to form the conceptual model initially, then the model is refined using a specific case study identified. The identified case study was also used to test the model as discussed in chapter six with appropriate testing methods described in chapter five. Chapter 6 also explains steps taken towards creating the larger total mine model by modelling chunks of sub-processes which are drilling, blasting, loading, hauling and crushing. Finally, application of mine specific data and discussion of the results are discussed in chapters seven and eight.

The thesis has shown that it is possible to represent the total mine value chain in a simulation modelling environment using system dynamics tools effectively at the abstract level to find answers to specific mining problems. The created model is generic enough so that it may apply to ` size and shape of the surface mining environment. The model created provides answers to the research questions in sufficient detail in terms of financials and efficiencies as the model is able to provide a quantified answer. The approach to quantification of impact of drill automation is based on the elimination of drilling deviation. The main assumption is that automation will lessen the variation due to resource limitations related to low quality drilling and blasting parameters and also drilling deviation related inefficiencies, by achieving the targeted or planned quantities of drilling required to achieve annual tons of ore production. The benefit of reducing the drill deviation by about 10% is calculated as 610 million Rands per year for the simulated case study which has a simulated mining cost of 16.1 billion Rands per year excluding labour and overheads. This amount easily justifies the automation of all the drill rigs in a mine. There could be other hidden benefits due to less variation in the mining environment due to on-time drilling being achieved which is beyond the scope of this thesis.

In conclusion, a novel parametric system dynamics model helps mining engineers to get quick answers to systematic changes made to mine excavation related parameters and the overall impact of these changes to the cost of mining as well as any efficiencies. The drilling quality is a significant parameter in any mine. The consequences of not following the designed drilling parameters lead to many complications. The research which resulted in a detailed system dynamics simulation model can give quick answers to mining engineers in terms of expected fragmentation levels, changes to resource requirements in terms of drilling, blasting, loading, hauling and their related costs. The technological changes which force the mine to change the planning of the mine environment on a daily basis also causes disruptions to the other mining processes. Therefore, SD modelling helps the management to have a deep understanding of the interactions of key parameters.

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LIST OF ABREVIATIONS

DSM	Dynamic systems modelling
SD	System Dynamics
VBM	Value Based Management
NPV	Net Present Value
NCV	Net Cash Value
PP	Payback period
IRR	Internal Rate of Return
MTTR	Mean Time to Repair
MTBF	Mean Time Before Failure
ROM	Run Of Mine
OEE	Overall Equipment Effectiveness
SSM	Soft Systems Methodology
HTP	Holistic Thinking Perspectives
IS	Information systems
FFD	Functional Flow Diagram
FD	Flow Dictionary
CLD	Causal loop diagram
QFD	Quality Function Matrix
OEE	Overall Operational Efficiency
LCC	Life Cycle Costs
GET	Ground Engaging Tools
GPS	Global Positioning Systems
TCO	Total cost of ownership
IRP	Instantaneous rate of drill penetration
DOH	Direct operating hours
PSD	Particle size distribution
MPI	Mine productivity index
TPM	Total productive management
OEM	Original equipment manufacturer
LP	Linear programming
FMS	Fleet management systems
LHD	Load haul dump machine
LTI	Lost time injury
WBS	Work breakdown structure
OJE	Overall job efficiency
ADT	Average daily tonnage
LCM	Loose cubic meters
BCM	Bank cubic meters

Traditionally the mining industry does not regard itself as being an operator of a system. It seems to regard itself as being the operator of equipment"

Chapter 1

1 Introduction

1.1. Background information

The mining process once established, becomes a complex business to plan, measure, control, and report. The complexity and cyclic nature of the mines are further exacerbated due to the size of the mine, fleet sizes and geology. To manage such a mine, various systems are in place to plan, measure, control, and report. The quality control of these processes is often very cumbersome and an oversight.

Numerous optimization studies are normally carried out during the life of a mine. New equipment, technology and systems may need to be introduced into an existing system from time to time. Data generated is often limited due to resource limitation and the size of the mine. The formal communication or data flow between processes may be interrupted due to introduction and adoption of new systems or technology. This is probably one of the main draw backs of adopting better systems and technologies due to limited trust in the change and/or uncertainty of the level of disruption in the rest of the system. Often the adoption of modern technologies is halted halfway due to short-sightedness and lack of evidence that it will change the bottom line.

Introduction of modern technology, changing existing systems and interventions of other processes have an impact on downstream processes in mining due to the cyclic nature of mining. It is possible that the impact of any of these changes are either over-estimated or under-estimated. Common experience suggests that the quantification of benefits or lack thereof could be difficult. Often decisions must be made without quantifying the overall impact, therefore adoption of modern technology or systems are slow in the mining industry, and the managers are reluctant to make decisions. Typically, spreadsheet type of applications are used in such studies. Due to high inter-dependency of mining processes the impact is not easily captured for the total mine with traditional spread sheet modelling. This may result in the result being skewed due to departmentalized isolated studies that does deviate away from the reality. Therefore, conventional approach may not be sufficient, That is why system dynamics is needed.

The initial research (Anonymous, 2015) that there are some approaches towards quantification, but they are rather departmentalized in terms of cost benefit analysis (CBA).

The measurement as we know is often generalized and biased based on the researchers approach and knowledge and most often it is hidden behind statistically interpreted summaries of the researcher's analysis and site specific. The dynamic nature of such a mining environment required a dynamic modelling approach. In big organizations and mines it is understandable to check customary practice of other mines for design and planning of operations. But once a mine is up and going those initial assumptions and design parameters should fall away and the mine starts building their own criteria and they have a better idea of what works for that specific operation. The established large mines however may be reluctant to change what is working for them; therefore, there is understandably resistance to change when changes are introduced to the system, if things go wrong, they go wrong in a big way due to the mine size. In addition, mines' undeniably complex nature makes it hard to see the incremental impact on the rest.

Considering the current survival mode of the mining industry, automation could be a new step towards a profitable environment. Technology has advanced considerably in the past 40 years and raises hope for a more efficient mining. As the mines start adopting recent technology, measuring how efficient and effective in terms of productivity and value this modern technology will bring is somehow not catching up with the speed of technology being injected into the industry. This is true for mining companies as well as technology providers.

Kolomela mine, one of the Kumba iron ore mines, has been identified as the case study where the drilling machines have been fitted with automated systems. Kolomela has fitted the existing drill rigs with Flanders automation systems that can be remotely operated either automatically or semi-automatically. The return on investment or value calculation of this system and the impact in changing market conditions becomes a biased decision if the model used to calculate the value is not flexible and adaptable to changing conditions. Therefore, a value-based model is needed to justify the adopted technology.

Anglo American reports on their public web site (Pienaar, 2016) that the benefits of an automation project are:

- "23% gain of direct operating hours
- 18% gain in drill rate (the actual time to drill a hole and to move, setup and start the next hole)

- 19% reduction in drilling cost, and
- 70% less injuries and fatalities”

How these claims were measured is unknown to this author. These and possibly more benefits could be quantified, and the results validated with high confidence. Having said that, no mine will go through a substantial change without seeing enough proof and confidence in the new technology will bring to the mine. The process of adoption of complex technology into a complex organization may require a formal approach for tracking the effect of the change. This research will attempt to analyse the mining cycle with the objective of tracking the potential changes in the system and where the disruption is most felt in the whole mining cycle. In order to avoid long and tedious trial runs to see the consequential effects, a mine manager should have an option to test the outcome with a quick to setup tool that can mimic the production environment with easily obtainable realistic detail for answers.

“77% of mining specialists think automation is high priority, and 40 % say now is more important than ever” (Somarin, 2014). Mining companies are excited but cautious about operational and financial benefits from automation technologies whether fully automated or semi-automated. Benefits listed by Somarin are: Consistent continuous operations, reduced infrastructure, and improved communications by promoters of these technologies. These benefits are not necessarily measured immediately, especially if the technology adopted is recent in a mining environment. Even though there are these claims of benefits, it is difficult to quantify some of them for future predictions. Once the technology adopted is fully implemented and observed for a period then the potential changes can be realized due to step changes in the performance curves. “The mining industry is primed for automation. It is capital intensive, buys expensive equipment and pays relatively well.” Says Jim Urquhart (2017) in computer world. If the technology is expensive, the value of it needs to be justified and the overall impact of it should be calculated. This is a challenge in the mining industry. Value Based Management (VBM) in mining companies is probably a popular term; however, it is often difficult to show the value of an injected process, innovation, or technology, due to the nature of mining. There is always a considerable risk attached to such changes due to highly variable qualities, efficiencies, and high interdependencies of sub-processes which are hard to visualize.

1.1. Problem statement

The rate of adoption of a recent technology is subject to its profitability and the degree of risk and uncertainty associated with it, and is highly influenced by the capital requirement, mining processes and standards, and the culture of the mine.

Technological change will have an impact on all organizations but is understood and observed differently by various stakeholders. The measurement of the effect of change is often not systematic and sometimes isolated on a specific area in the organization.

The value calculation of this system and the impact in changing market conditions becomes a biased decision if the model used to calculate the value is not flexible and adaptable to changing conditions. Therefore, a heuristic model may be needed to justify the adopted technology.

1.1. Rationale

The mining industry in the past 100 years or so has advanced with several approaches to mine planning, design, scheduling and optimization and is known to be slow to adopt to the recent technology (Anonymous, 2021). The mining value chain starts with locating a deposit that has an estimated economical value. Mine planners will evaluate the mine and determine profitability. It takes about 5 to 10 years for a mine to start full production and start making money to pay for the capital and return on investment. In addition to delayed return on investment, it is a high-risk business due to input variability and fluctuating demand to the commodities.

Common sense based on the history of technological developments would make one believe that automation of certain processes in the mining environment will improve productivity, efficiencies, increase value and therefore profitability. The question however that needs to be asked is how much value does the automation bring to the mining industry?

The dynamics in the mining industry has many attributes and they are often fluctuating over time with changing qualities and environment. How proactive could a mining engineer be to mitigate the negative impacts of these fluctuations? This requires some degree of predictive modelling of the system and measuring the impact of implemented changes. The conception of the idea that automation of drilling may bring some order to the fluctuating efficiencies in the mining environment led to a mining house to adopt automated drilling. The impact of this change has been estimated to some degree. The impact of automation on the overall mining efficiencies has not been quantified realistically speaking and is reasonably difficult to measure due to many other changes that are affecting the revenue. Since isolation of the change from the rest of the system is almost impossible, the remaining option for mining engineers is to mimic the existing processes in a simulated environment and measure the impact.

Business processes need to be self-assessed, audited and sometimes re-engineered due to disruptive changes. Process assessment in a large organization may be cumbersome and difficult to visualize of the relational impact.

Therefore, the intention is to gather as much information as possible in a known mining environment and model the causal relationship. After several communications with industry experts (Prof. Tinus Pretorius, Prof Leon Pretorius, Prof Johan Joubert) it has been decided that this question may require a “System dynamics” approach.

1.1 Research question

Based on the previous discussion the following research question is addressed in this thesis:

“*How to quantify the effects of a recent technological change in an open pit mine which has cyclically connected sub processes?”*

1.2 Research objectives

The main issue raised in the research question is that a simple change in one parameter leads to multitude of changes in the sub systems that are not necessarily linearly connected but is cyclic therefore may result in exponential changes to the results. Intuitively a solution based on a simulated environment or more so of a gaming like environment may provide an answer to the multitude of questions that may arise in the researchers mind. The traditional methods of quantification are spreadsheet based tabulations of parameters that are linked to each other, which presents a working platform that is hard to visualize the causalities in a spreadsheet environment. This is more pronounced when the number of parameters is in the range of hundreds. Descriptive visuals are much more easier to interact with than tabulated spreadsheet solutions.

Following questions are some of the questions that could be answered with a realistic quick and intuitive modelling and should be considered for designing the conceptual model.

Drilling and Blasting:

1. How much impact do the changes in cycle times of drilling have on the effectiveness and efficiencies of the operation?

2. By how much does drilling accuracy have an impact on oversize and undersize?

3. By how much drilling accuracy affect the fragmentation therefore revenue?

4. In what ways drilling delays affect the flow of other processes?

5. By how much drilling and blasting costs are affected by speed and drilling accuracy?

6. By how much does schedule pressure affect quality therefore delays.?

Loading and Hauling:

1. To what degree the cycle time of loading and hauling reduced due to effective drilling?

2. How does delay in loading rates affect revenues?

3. How much fuel is saved by reduced cycle time?

4. What is the effect of fine/coarse fragmentation on revenues?

Therefore, the objective of this research is to establish the causal relationship of the value chain and model the dynamic behaviour of a surface mine for prediction of impact of a disruptive change. At the end, all is linked to the revenue generated.

Dynamic models depend upon an element of time, and which also brings about fluctuations in exchange rates, fuel prices and dynamic instabilities in a dynamic world. Resource variability is also another reality in mining that brings dynamic changes and fluctuations in the mining environment.

Primarily this study will create a novel system dynamics model that can be applied to mining that will simulate cause and effect relationships and the output will be comparable of before and after scenarios due to disruptive technologies injected into the existing system. The objective is not to replicate the uniqueness of an existing project but to simulate sufficient elements that can simulate the specific case study as well as other similar changes in the mining environment.

The model created should be able to predict the impact of changes introduced to mines during its life without having to go through large data generated by the mines that is often not visual and difficult to interpret the relationships and policies between various measurement fields. The model should be generic enough so that it can be adopted to a typical large open pit mine.

The intention of the model is to provide answers to specific questions geared towards mining value specifically to the effects of autonomous drilling which is the technological change that is introduced to the drill rigs in the selected case study presented and discussed in this thesis.

1.3 Scope and Limitations

The literature study will firstly focus on an open pit mine environment and review typical modelling solutions and test their suitability as a solution to the research question to determine the impact of change. There will also be references similar in nature to the case study. The mines are interested in the economic value of a decision as well as to what degree the benefits are realized and quantified. Concluding on a decision made and implemented in a mine environment is not easy as mentioned by Roumpos and Akylas in 2004 (as cited by Sontamino, 2014). Roumpos states that for a complex environment such as a mine, a tool is required for the decision-making process which is in high demand. The same can be said for evaluating the results of such a decision once it is implemented. Typical criteria used for evaluating economic decisions are Net Present Value (NPV), Net Cash Value (NCV), Payback period (PP) and Internal Rate of Return (IRR) (Sontamino, 2014) within the system dynamics modelling environment.,. The author is in the opinion that future similar projects can be evaluated with system dynamics based modelling with similar financial measures for this thesis.

The more the mining value chain gets complex the more data is created than it was before. It is not practical to model a mine with every small detail. There must be boundaries or summaries of processes or data that is concise but representative.

The dynamic flow of mining processes change based on production needs as the mine progresses towards its final boundaries. During the life of a big mine, processes are invaded with increased technological interventions which is fitted into the existing environment on a needs basis. All this complexity is managed at different departments with different tools. Sometimes data is not used to give feedback to the other systems that are affected by the processes upstream or downstream. This also adds to the resistance to change behaviour. Each process has multiple variables and parameters that are often interdependent. The processes also have different evaluation parameters. Roumpos and Akylas stated that due to interdependency of the evaluation parameters they have adopted a systems approach (2004). The same has been mentioned by Sontamino (2014) that a mining system (in his case a coal mine) needs a tool that can connect key variables and can calculate or simulate quickly and flexibly.

Limitations and boundaries will be established further along this research. However, it is essential at this point to select the right tool that will meet the objectives.

1.4 Review of Existing Tools for Meeting Research Objectives

Firstly, why system dynamics but not a traditional Excel modelling approach is selected needs to be explained. Pruyt (2013) reports that spreadsheets could be used to model system dynamics models. But spreadsheets implicitly hide many structural assumptions which are rendered explicit with graphical system dynamics packages. There are however graphical add-ins (such as Expose) that improve system dynamics modelling by means of spreadsheets. With Excel numbers are in the forefront and difficult to see the underlying relationships in totality, i.e., relationship can be seen in the formulae bar and requires a good memory of the user. In system dynamics structure is in the forefront and numbers are called upon when needed. In addition, system dynamics tools present a methodology and has a visible structure compared to Excel as a tool. Some system dynamics forums have some comments by the users of software as well a s a comment from Sterman who is at the forefront of system dynamics modelling. Some of the user comments in in Ventana Systems site are:

- User comment: “Properly capturing feedbacks and stock-flow structure in a spreadsheet is tedious, and it is easy for errors to creep in undetected.”
- User comment: “Models built in excel are hard to document, explain, and modify”
- User comment: “Client understanding, and confidence fall, and chances of meaningful implementation plummet, even, in the hands of careful experts”
- User comment: “We have tried out both and I would say that spreadsheets are definitely a good tool to create small models and try things out. When a model becomes more complex (i.e., more than 100 variables) it is quite difficult to handle it with a spreadsheet”
- User comment: “The path defines the numbers more than numbers defining the path. In excel path is not visualized but in system dynamics the path is in the forefront not the numbers. See Figure 1 for the visual representations of both tools.”

In summary the path defines the numbers more than numbers defining the path. In excel path is not visualized but in system dynamics the path is in the forefront not the numbers. See Figure 1 for the visual representations of both tools."

Excel becomes cumbersome to manage the complexity due to the way it has to create a column and a row of numbers for every single parameter being used, causing millions of numbers showing up in the fore front than the causality leading to those numbers.

Figure 1 Comparison of System Dynamics Modelling Environment to Excel based modelling environment (image left : Modified from Geissdoerfer, 2017 and image right: Own)

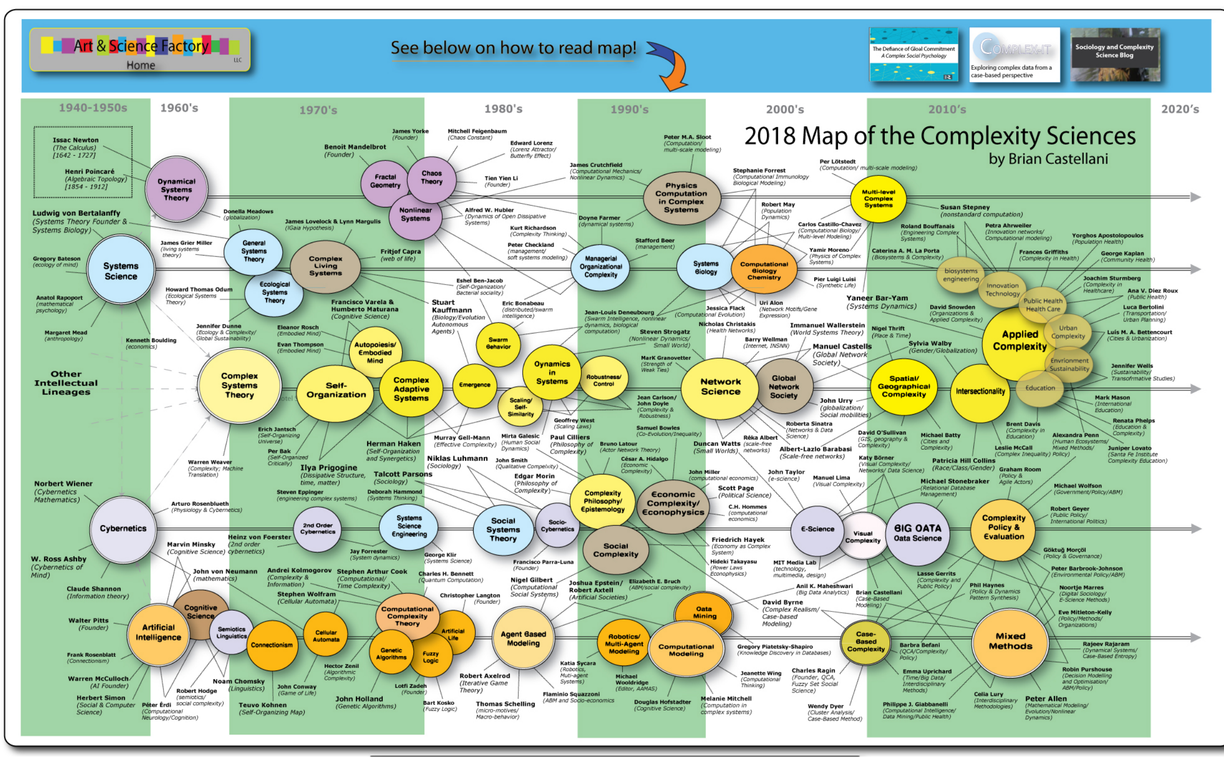


Figure 2 Brian Castellany's complexity map (Castellany, 2018)

The mining complexity is managed with years of established experience of the miners. Recent technological improvements also created vast collections of data therefore complexity arises with all the variables that exist in the mining operation analysis. If complexity science is defined it can be said that it is concerned with complex systems and problems that are dynamic, unpredictable and multi-dimensional, consisting of a collection of interconnected relationships and parts. Unlike traditional “cause and effect” or linear thinking, complexity science is also characterized by non- linearity. A summary of the developments in the field of complexity science can be seen in Figure 2 where systems thinking, and system dynamics also feature (Castellany, 2018)

There are three methods for modelling that are well known, namely, system dynamics, discrete-event simulation and agent-based modelling. Marshall et al. (2015) reviewed these three methods summarised in a table, which can be found in the Figure 2.

The skill set as well as quick construction are reasons why system dynamics seems to be more appropriate. In summary these characteristics are:

- Population size scalability,
- More accessible skill set,
- Aggregate level data and
- Quick construction (depending on the skill level)

with the purpose of

- Deterministic,
- Engaging stakeholders,
- Relevance of patterns and/or aggregate values and strategic level problem solving.

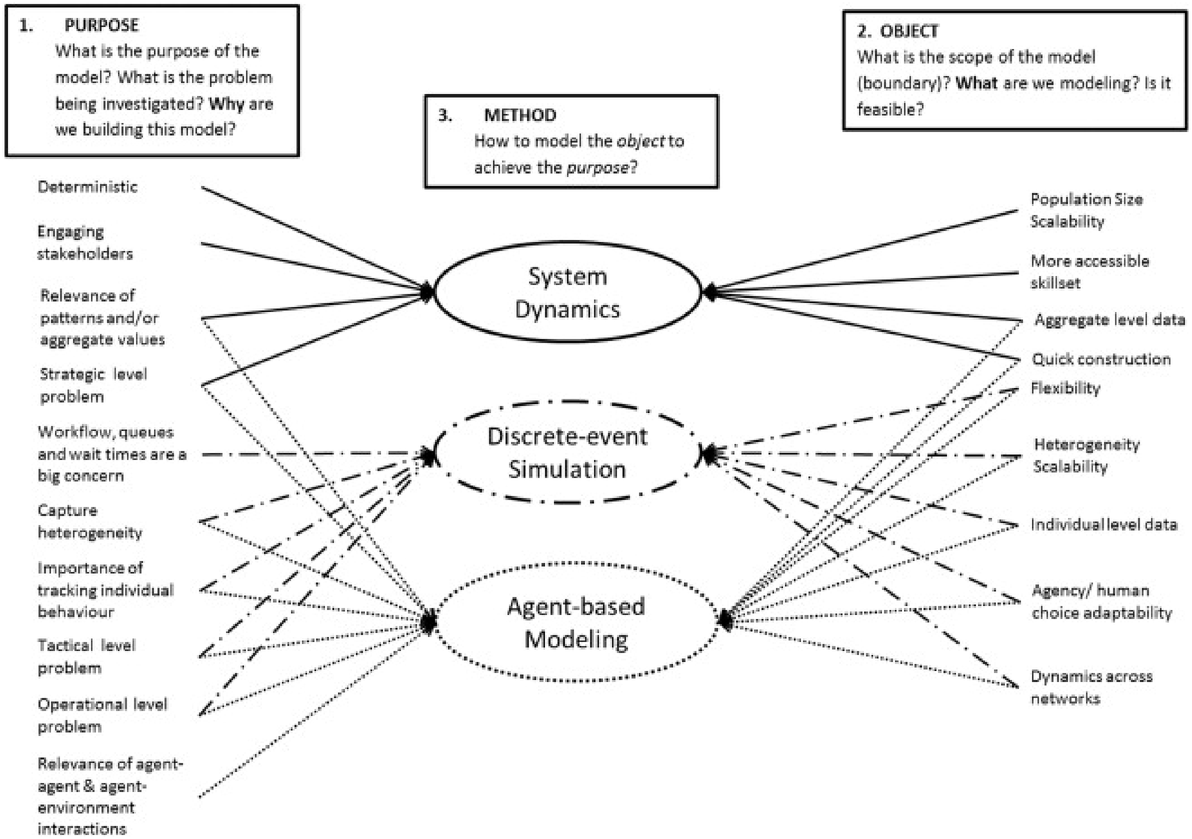


Figure 3 High level summary of criteria for selecting a dynamic modelling method (Marshall et al, 2015)

1.5 Systems Thinking and System dynamics Approach

"As a language, systems thinking has unique qualities that make it a valuable tool for discussing complex systemic issues" (Anderson, 1997):

- It emphasizes looking at wholes rather than parts and stresses the role of interconnections. Most importantly, it recognizes that we are part of the systems in which we function, and that we therefore contribute to how those systems behave (Anderson, 1997).
- It is a circular rather than linear language. In other words, it focuses on "closed interdependencies," where x influences y, y influences z, and z come back around to influence x (Anderson, 1997).
- It has a precise set of rules that reduce the ambiguities and miscommunications that can crop up when we talk about more complex issues (Anderson, 1997).
- It offers visual tools, such as causal loop diagrams and behaviour over time graphs. These diagrams are rich in implication and insights. They also facilitate learning because they are graphic and therefore are often easier to remember than written

words. Finally, they emphasize the dynamics of a problem, not individual blame (Anderson, 1997).

- It opens a window on our mental models, translating our individual perceptions into explicit pictures that can reveal subtle yet meaningful differences in viewpoints (Anderson, 1997).

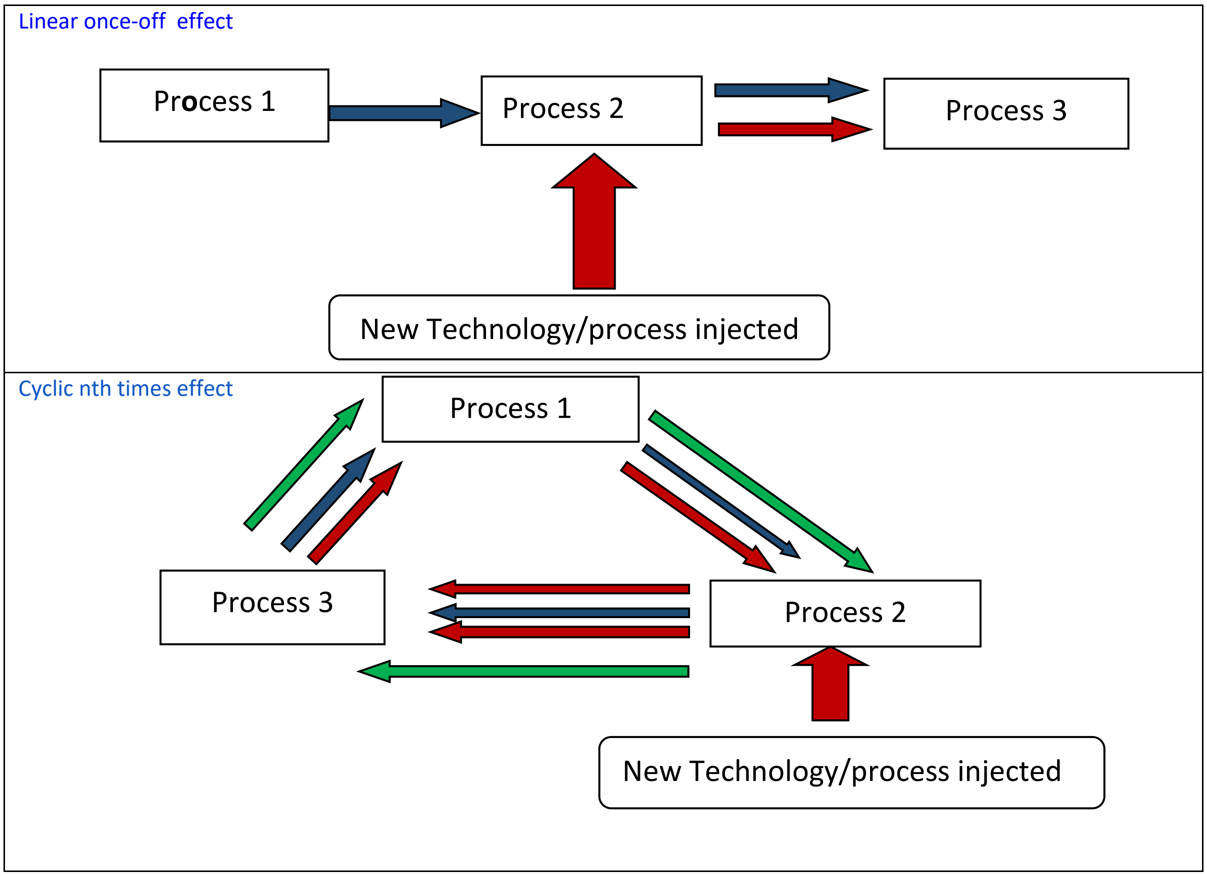


Figure 4 Linear versus cyclic processes creates a multiple of impact nth times in a cyclic process

The systems approach was initially mentioned by Peter Senge and John Sterman (1992); they have emphasized that systems thinking had been advocated in the past for an effective change. Therefore, microcosms of real businesses are proposed to be simulated in bigger organizations. Inferior performance, organizational failure and inability to adapt due to limited cognitive skills and capabilities of individuals result due to the complexity of systems. When a decision is made (such as implementation of automation in the mining environment) it has indirect, delayed, nonlinear and multiple effects. The most important observation is that "as the time delays grow longer and the feedbacks more powerful, performance deteriorates markedly" (Senge, 1992). Conceptually, "feedback concept" is at the core of the system dynamics approach (Sontamino, 2014).

Oosthuizen (2014) also developed a system thinking and system dynamics modelling methodology for complex sociotechnical systems, which requires cognitive work analysis, and system dynamics. He states in his thesis that system dynamics analyse the effect of feedback and delays on an operating system. This methodology is to be mostly adopted in this thesis to determine the effect of "change" in an open pit mining environment and could be used to model qualities and quantities.

The mine behaviour can be unpredictable in nature and is often cyclic. The new process of technology injected into a sub process will impact the next process as shown in Figure 5. The calculation of impact in the case of a series of discrete processes is relatively easy as the process starts at Process 1 and ends at Process 3 to Process N and so forth. If the process is cyclic, then the effect will be more pronounced in all other processes with either a continuous improvement or more of a pronounced failure "nth times. Both cases as shown above are possible in a bigger organization. The above can be best illustrated with a "system dynamics" map as seen in Figure 5.

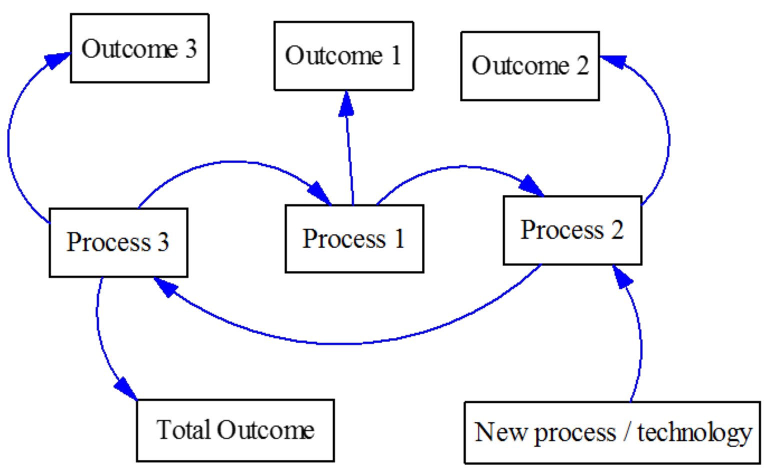


Figure 5 System dynamics map showing the simulated outcome of a new process

System Dynamics (SD) is a methodology used to understand how systems change over time. The way in which the elements or variables composing a system vary over time is referred to as the behaviour of the system (Kunc, 2018).

Mathematically the basic structure of SD can be expressed as nonlinear, first order differential equation or integral equation.

$\frac{d}{dt}X(t) = f(X, P)$ **, where** **Equation 1**

- X is a vector of levels (stocks or state variables)
- P is a set of parameters,
- F is a nonlinear vector-valued function.

This research will investigate how best to model the impact of change by quantitative and qualitative analysis techniques, and this may also answer the question of how beneficial the automation of drilling in the mining context can be, whether it leads to higher productivity per employee or per unit capital cost employed. According to Dessureault (1999) the job numbers will stay the same contrary to common belief that automation will result in job losses. This and other measurables that are identified via a literature review will aid the author to model this impact. Prior research by doing a basic literature review revealed such as Jordaan (2009) Gustavson (2010), Nebot (2007), Corke et al (2014) that there are no practical systems measuring this impact. The research question therefore arises, what would be the impact of the future changes such as automation be to the bottom line?

Dessureault (1999) stated that formal economic justification techniques have defended the trend of short-term gain of quantity over quality. Often mining companies have not adopted automation due to lack of adequate justification to their management. There is also a misconception that technology continually leads to a reduction in labour. Therefore, automation could be seen as a threat by the mine employees.

Dessureault (1999) summarized the reasons behind the failure of modern technology such as automation as follows:

- The justification process is more complex,
- Too many qualitative attributes
- Risks involved in implementing the interpreted technologies.

The mining process has a complex cause and effect relationship. The mine site value chain needs to be understood well in advance before attempting such a measurement. The process model developed in this thesis will indicate the level of impact of each process to the next step. This model can be further used to highlight input and output relationships in terms of costs and volumes.

1.6 Full Database Analysis Systems: Big Data Approach

MineRP was investigated as a potential tool for use for this study. It is a software tool that can be used to integrate mine site data from various processes into one platform. This tool has three components (MineRP web site):

1. Analyzer
2. Spatial Dashboard
3. Publisher

Data in mining is more of a spatial nature and should be analysed in that fashion. Visualization of the data with one of the mining packages may be useful. However, this may require further analysis before attempting to use such a complex tool, if spreadsheet based templates could be planned. Common experience is that the data analysis formulae used is not normally visible to the user if spreadsheet type software is preferred.

Another option in data analysis is IBM analyser, the company which has a division on Mining Data Analysis, but the tool may be complex to use and found expensive for this research.

These tools are based on SQL and uses graphical databases for speedier data analyses. The analyser can be setup with rules written in SQL language. Graphical databases compared to relational databases have an advantage in terms of speed, visualization and building cause and effect relationship in a mining environment. Mine RP can be setup to access already existing mine databases such as Min 2-4 D, MineCad, STP, CAE etc. Since files systems of each process can be understood by Mine RP it can be used as the main tool to analyse the KPI's. This option is perhaps an appropriate choice for the mines in the context of integrated mining software available to the operations, but is beyond the objective of this research, therefore it is discarded as the research tool.

Mine RP, MineCad, Min 2-4d are all relying on the big data which is often very confidential and not accessible by independent researchers. The data types and their translations to carry them to another platform means a substantial change for a mining company and often will not be justified so easily either. Therefore, this research is being carried out.

1.7 Mining Process Analysis for Modelling

The mining process may need to be fully analysed to map the cause-and-effect relationship, which in turn may assist in calculating the effectiveness of an adopted technology.

Surface mining processes can be best explained by the following schematic (Figure 6) developed by the researcher as part of a system dynamics/thinking process that will be presented in more detail later in this research. Elements of this representation is common mining engineering sense and can be found in many basic mining engineering books such as Kennedy (2000).

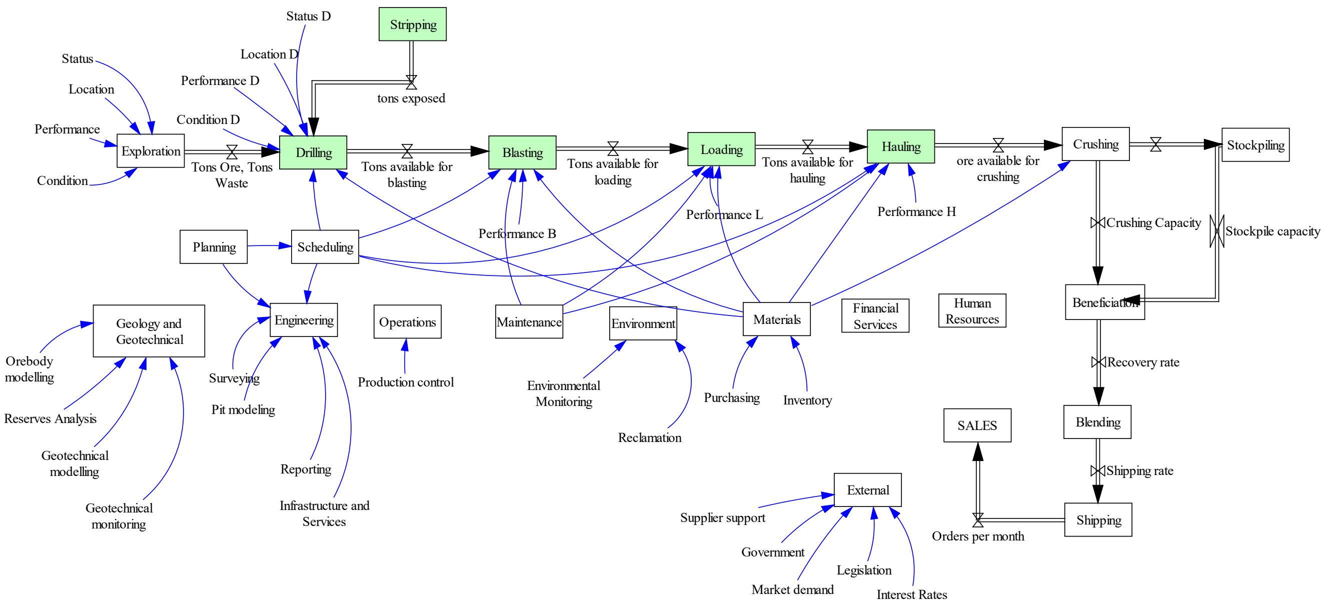


Figure 6 Mining Processes

Table 1 lists typical deficiencies observed by the researcher in drilling process and the impact that process has on the operation in general.

Table 1. The impact of automating the drilling cycle

Item	Automation process	Impact of that process	Impact on productivity
Hole position	Using positioning systems to determine exact location of holes	Holes are positioned exactly according to the blast design	No lost time waiting for survey. Improved control over the blasting process leading to better fragmentation and particle size definition which means that the loading and hauling operations are more efficient and leads to 10% increase in plant throughput
Real time down hole sampling	Using on-board computer to determine the characteristics of the strata and update the geological model on a real time basis	The blast design can be accurately determined, and the explosives accurately positioned to ensure the best fragmentation	Improved control over the blasting process leading to better fragmentation and particle size definition – this means that the loading and hauling operations are more efficient and leads to 10% increase in plant throughput

Drill hole depth control	Using on-board control computer to accurately determine depth of drilling	Less over/ under drilling which reduces dilution. It also results in less damage to the next bench; therefore, less preparation work is required.	Increased plant efficiency due to less dilution. Less work preparing the pad for the next blast on the next level – approximate reduction in dozer usage by 25%
Improved drill positioning	Using on-board computers to accurately position and drill holes for blasting	Leads to improved blasting control which in turn means that benches can be more accurately determined. This allows the geo-technical staff to better predict the slope stabilities	It is anticipated that by improving slope stability by better blast control – a direct result of better drilling that slope angles in open pits could be increased. An increase in slope angle of 1° in a pit of diameter 1km and 200m depth would result in a reduction in handling of approximately 300 000 BCM's of waste. This would increase exponentially as the volume of the pit increases.

Improved drilling will lead to more benefits such as improving loading and hauling of the ROM (run off mine) material, allowing the increased production targets to be realized. The improved mining efficiencies mean that the required *production rates can be achieved with less equipment.*

Well known efficiency measurement tools need to be considered such as operational costs, labour costs, availability, MTTR (Mean Time to Repair), MTBF (Mean Time Before Failure), total tons moved, time, and the most important one is cost per ton.

The next benefit area is production losses due to delays. How much of the delays can be attributed to drilling quality and efficiency? This is difficult to quantify. It is possible to simulate current and future mine processes via typical spreadsheet-based analysis of the actual data however, it is not always easy to isolate the loss/gain due to a certain process such as automated drilling. The automatic capture of events during drilling frees operators from paperwork and removes inaccuracies and inconsistencies of manual systems. Complex issues are broken down into common causes, enabling continuous improvement in reducing costs and streamlining operations. Greg Johnson (2016) focused on quantification of losses due to stoppages and an example is given in Figure 7 below.

The data capturing processes in a mine environment are not often validated in terms of accuracy. Mines have information management technology for mobile equipment, some of them are captured automatically and some are captured manually. Each department interprets the data differently (not necessarily incorrect) often leading to some loss of detail in data.

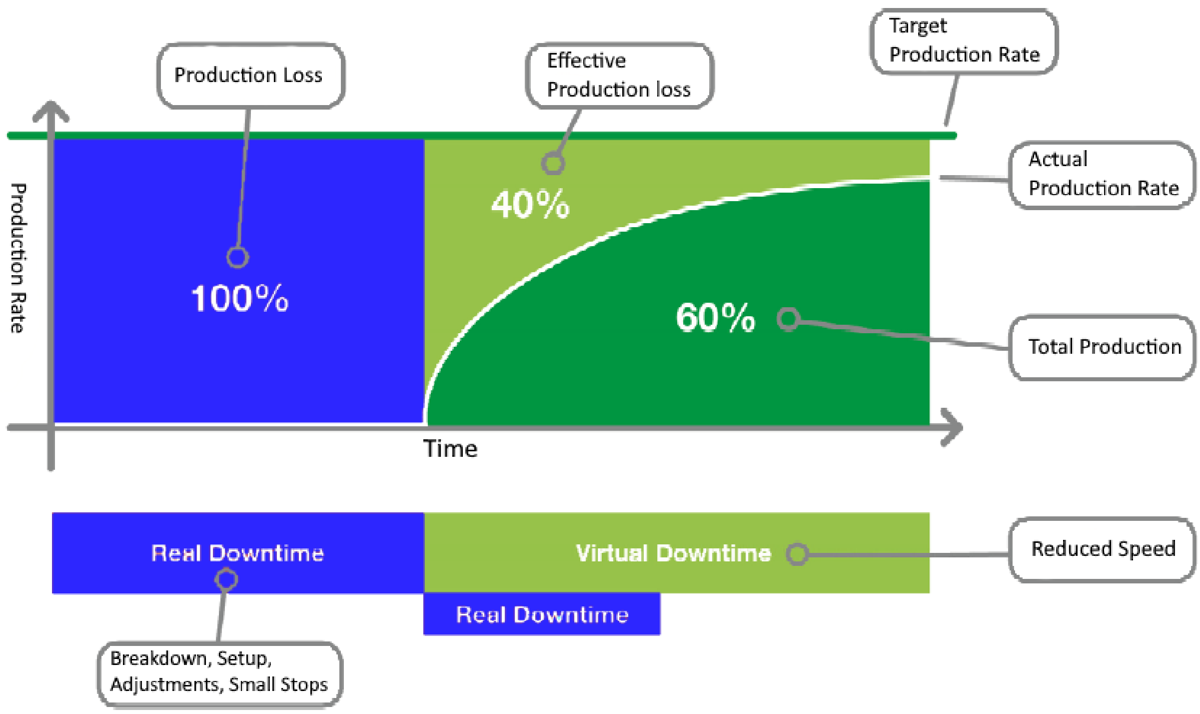


Figure 7 Production losses due to stoppages and slow running (Johnson, 2016)

To design and quantify the physical and economical relationships require valid relationships and scenario analysis. This research intends to quantify losses due to stoppages directly or indirectly related to the drilling process.

The automation also brings automated data capture, which helps manage the mining process better. The following has been further added by Johnson (2016):

- Automatic record creation on occurrence of a downtime event or manual entry
- Editing and of event location, classification and event code. Validation of information and business process workflows if required
- Audit trails and flexible security due to system being only accessible by people that can add/delete/edit.
- Ability to capture real or virtual downtime
- Ability to capture time overruns or delays on production activities,
- Complex event capture conditions (e.g., “conveyor is running but motor is drawing low current)
- Ability to automatically assign causes to alarms and other process data,
- Ability to split events (e.g., the plant is down for scheduled maintenance, but there are delays in re-starting for production reasons).

1.8 Measurement, Metrics, Equipment Effectiveness

The definition and use of overall equipment effectiveness (OEE) over the years have been widely debated (Elevli, 2010). It is a measure of total (complete, inclusive, whole) equipment performance – the degree to which the asset is doing what it is supposed to do, OEE measures total performance by relating the availability of a process to its productivity and output quality (Jonsson and Lesshammar, 2006). It addresses all losses caused by the equipment, including:

- Not being available when needed because of breakdowns or set-up and adjustment losses,
- Not running at the optimum rate because of reduced speed or idling and minor stoppage losses
- Not producing quality output because of defects and rework or start-up losses.

In summary OEE is quantifiable if % availability, % productivity and % quality is known first reported by Nakajima and followed by De Groote as cited in Jonsson and Lesshammar 2006).

	Nakajima (1988)	De Groote (1995)
Availability (A)	$\frac{\text{Loading time} - \text{downtime}}{\text{Loading time}}$	$\frac{\text{Planned production time} - \text{unplanned downtime}}{\text{Planned production time}}$
Performance (P)	$\frac{\text{Ideal cycle time} \times \text{output}}{\text{Operating time}}$	$\frac{\text{Actual amount of production}}{\text{Planned amount of production}}$
Quality (Q)	$\frac{\text{Input} - \text{volume of quality defects}}{\text{Input}}$	$\frac{\text{Actual amount of production} - \text{non-accepted amount}}{\text{Actual amount}}$
OEE	(A) × (P) × (Q)	(A) × (P) × (Q)

1.9 Data Requirements

The cause-and-effect modelling may require detailed insight into the mine systems. Kumba as well as Flanders – supplier of automation systems, have been contacted for support with data and gain an initial understanding of the drill automation setup. Flanders representative, as well as mine team has been met prior to this research.

The mine personnel including automation centre workers, Flanders on site technicians and Kumba automation project manager all agreed at the that system has not been fully evaluated. However, as the research progressed the research was not scoped around evaluating “a” system but on quantification of value of such systems with a generic modelling study which requires a generalist knowledge of typical mine design parameters used, not necessarily mine specific production data. Therefore, mine specific production data is being avoided in this thesis and will not add much value to the modelling efforts at the level of abstraction it is being designed.

1.10 System dynamics Methodology

The strategy of this research is for testing the completeness of systems and integration into the mine operational and management structure by

- Reviewing the current applications and doing an in-depth analysis of how the technology was trialled and implemented at the mine,
- Identifying techniques of measuring the level of success of implemented technology with the available systems and skills in the mine,
- Create a map of the inputs and outputs of the existing mine systems,
- Define critical points in the system map and how measurements will fit strategically into an existing infrastructure,
- Identify main KPI’s that will be used to measure the efficacy of technology and integrate into the existing mine management systems,
- Develop an architecture of the control and feedback mechanism for closed loop control and measurement using a suitable mining simulation tool.

The research approach will be adapted as published by Sontamino (2014) as shown in the process map in Figure 8.

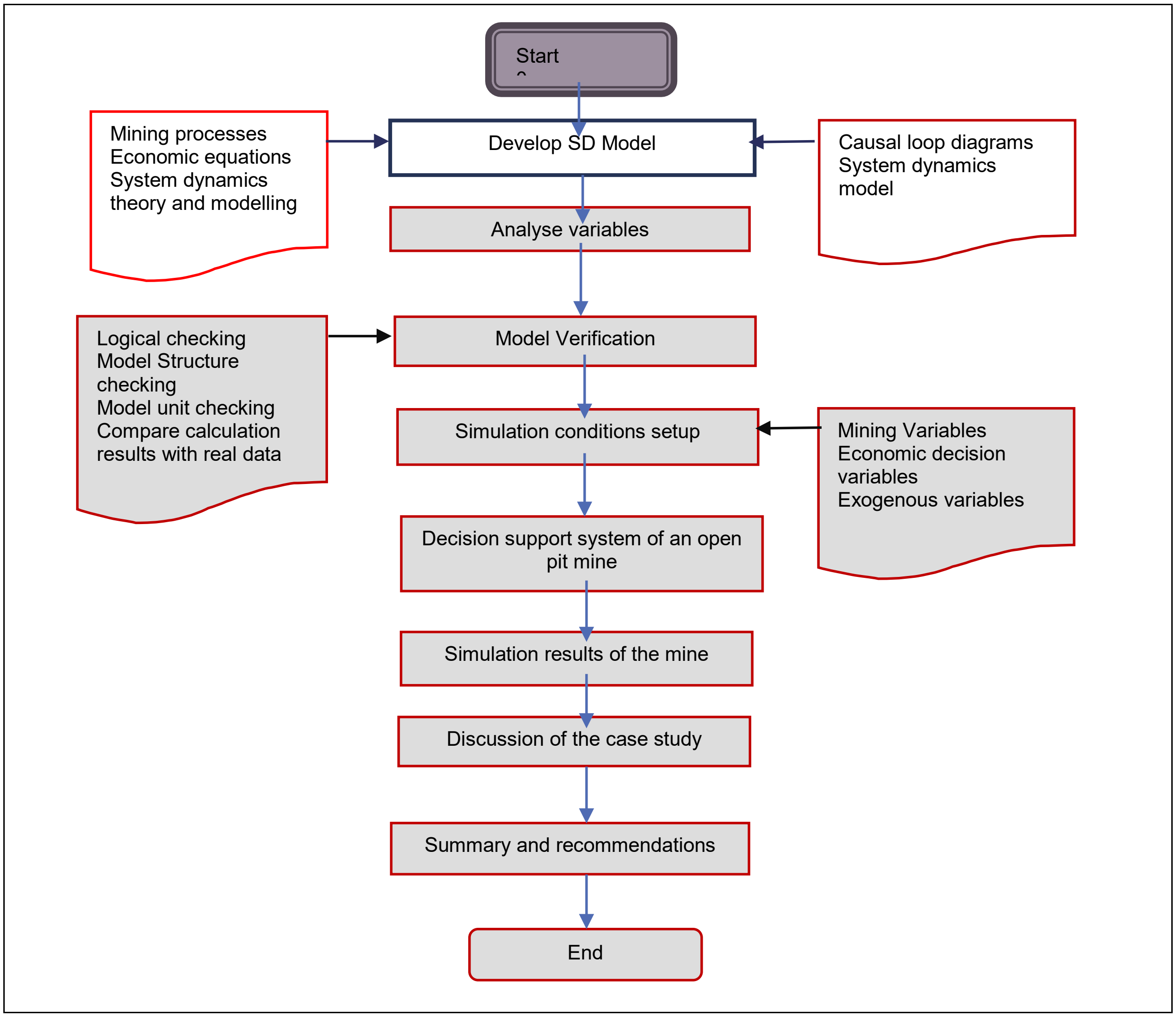


Figure 8 System dynamics methodology

1.11 Research Plan and Thesis Outline

Step1: Expand the literature review to select best modelling techniques suitable for modelling and justify the selection.

Step 2: Determine the input parameters that are exogenous and not dependent on any other parameters that will be used in the model and that their value can be changed freely by the user for sensitivity analysis.

Step 3: Establish relationships that are dependent on exogenous parameters as well as system generated (endogenous) parameters.

Step 4: Create the architecture of causal loop diagrams that the outcome to determine the value qualitatively or quantitatively.

Step 5 Analyse the case study with created model and determine the key inputs and outputs for the case study.

Step 6: Check the correctness of the model based on the model output in real terms.

Step 7 Discuss the results of the case study and the applicability of the model.

Step 8: Summary and recommendations

1.12 Chapter Summaries

Chapter 1: This chapter explains the background to the research, objective, rationale, the research question and research objectives.

Chapter 2: Research methodology is discussed mainly on discovering the extent of system dynamics concepts and applications in various disciplines including medical, engineering, environmental and management studies. This then led the author to decide on the research method to be applied for the thesis study.

Chapter 3: This chapter reviews the literature for various modelling and quantification techniques that can be used for modelling surface mining processes including tools available, logic in the way models is constructed and the mathematical language of the system dynamics is explored. This then helped in constructing some basic models at least at the conceptual level to define the problem boundaries

Chapter 4: This chapter explains how system dynamics is to be used for modelling the surface mine environment. The techniques of modelling applied by other disciplines that can be adapted to mining at least at the conceptual level were reviewed in literature. The mining value chain was reviewed in full, at the level of abstraction for the modelling and also defining the parameters, constants and auxiliaries to be used in the system dynamics model for this thesis. and explores mining processes and their causal relationship with the other processes.

Chapter 5: This chapter presents existing model validation techniques available in literature which are largely discussed by the system dynamics modellers. Parameters constants and formulae applicable to this thesis that were highlighted in chapter 4 are now expanded with more detail and put in a structured causality map for each mining unit process that are drilling, blasting, loading and hauling.

Chapter 6: Further refinements and modifications to answer some of the questions raised in chapter 1 is now put to test in this chapter. Drawing the detailed picture of value chain and selecting the quantifiable endogenous or exogenous variables, inputs, causalities and relationships that exist at a surface mine.

Chapter 7: Discussion of the results and a critical analysis of the unit processes with a view on quality as an outcome of automation was tested in the model. This section has discussions on application of mine specific data using a case study to test if the system dynamics model is giving reasonable results and discussed if the research objectives were met.

Chapter 8: This chapter has concluding sections on the model built and challenges met during the study as well as running the simulation.

1.13 Ethical considerations

Qualitative approaches often rely on the use of surveys, interviews, and informants. These qualitative data gathering methods cause information from contributors, which may or may not be sensitive. Any interaction (none for this study) with contributors is to be executed in a manner, which ensures compliance with the ethical regulations as set out in the Code of Ethics for Scholarly Activities and Policies and Procedures for Responsible Research, by the University of Pretoria.

In this research, by nature, no person, persons, or organizations have been identified or quoted without permission, especially personal communications, or the material has been published and is available for public information. All such material used in the research has been duly acknowledged.

To the best of the author’s knowledge, there has not been any ethical transgressions either in process of the research or recorded in this thesis.

1.14 Chapter Summary

This chapter introduced the research question, objective and the background to the research identified regarding the need to quantify the impact of changes via system dynamics modelling.

"IT IS NOT ABOUT the problem you are going through IT IS ABOUT your approach to the problem to solve it"

Chapter 2

2 RESEARCH METHODOLOGY

System dynamics is a tool to understand behaviour of systems under set conditions and has been around for a while including software available to support the methods of modelling the systems of concern. This chapter reviews the literature for various methods to solve systems problems and select the most appropriate method that suits the objective of this research.

2.1 SSM Methodology

According to Kasser (2019) there are seven stages to problem solving which are based on Soft Systems Methodology (SSM) originally proposed by Walsh (2015). These stages are.

1. Recognizing the existence of a problem or an undesirable situation.
2. Expressing the real-world problematic situation.
3. Formulating root definitions of relevant systems of purposeful activity from different Holistic Thinking Perspectives (HTPs).
4. Building conceptual models of the systems named in the root definitions.
5. Comparing the conceptual models developed in Step 4 with the real-world situation documented in Step 2.
6. Identifying feasible and desirable changes.
7. Actions to improve the problem situation.

Walsh (2015) discussed rational and defendable models called "Conceptual Models" using defensible logic. "Defensible logic" is deduced from statements of purpose and captured in "Root Definition". Conceptual models are than compared to reality. More information can be found on how to use Soft Systems Methodology (SSM) in Brian Wilson's paper (1983) mentioned in Checkland (2000). Checkland states that it is hard to bend thinking from hard to soft even for an experienced open-minded researcher in a review of the history of SSM (2000).

Mingers and Taylor list areas of application for SSM as shown in Table 2. The benefits of using SSM are also listed by the same authors in the resource:

- Managing the intervention
- Provides a structure/framework for study,
- Gives complete/wide ranging/holistic view,
- Appropriate tool for communication
- Improves speed of study,
- In thinking processes,
- Provides clarity of thought/structured thinking,
- Promotes shared thinking/gets people together,
- Frees the person from the current situation,
- Forces explicitness
- Promotes creative and stimulating thinking,
- Concerning the problem content system
- Structures situations which are complex/messy/have much information,
- Generates understanding of other people's perceptions and perspectives,
- Focuses attention on issues and organization culture,
- Does not make assumptions about a situation.

Table 2 Application areas for SSM (Mingers and Taylor, 1992)

Organizational Design	**Performance Evaluation**
Restructuring of roles Design of new organization Create new organization culture	Performance indicators Quality assurance Monitoring an organization
Information Systems	**Education**
Defining information needs Creating IS strategy Knowledge acquisition Initial scoping/players Evaluate impact of computerization	Defining training needs Course design Causes of truancy Analysis of language teaching
General Problem Solving	**Miscellaneous**
Understanding complex situation Initial problem clarification	Project management Business strategy Risk management methodology Case for industrial tribunal Personal life decisions

2.2 Systems versus Discrete and Agent Based Models

There are typically three modelling methodologies used to address problems that are: System Dynamics Modelling (SD), Discrete Event Modelling (DE) and Agent Based Modelling

(AB). SD and DE are originally developed by Jay Forester. They are methods that take a top-down approach. Agent based modelling focuses on individual behaviour therefore, considered a bottom-up approach (Sontamino, 2014).

Three methods of simulation can be compared in terms of abstraction levels (Anylogic White Paper, 2019), see Figure 9 Three approaches to modelling of mining environment (Anylogic) Maluleke (2015) mentions that when the level of detail is limited such as done at macro-economic level, the model does not make any detail split cost components of a mine operation or revenue of a business, since the aim of his simulation was not to achieve the economic accuracy but to establish a pattern of behaviour. Therefore, he selected the System Dynamics (SD) method out of the three methods. The same logic applies to this thesis.

The following is then summarized for each method by Sontamino (2014).

- When a system is individual data, then use an AB (agent based modelling) approach
- When a system is using complex continuous variables, then use a System Dynamics approach
- When a system can be described as a process, then using a Discrete Event Modelling may be appropriate.

The system dynamics in this case is the most suitable for analysing mining processes that that produce large and continuous data. The term dynamics refers to change in time (Sontamino, 2014). Each component that behaves in a certain way can be simulated over time. The system's structure defines the behaviour.

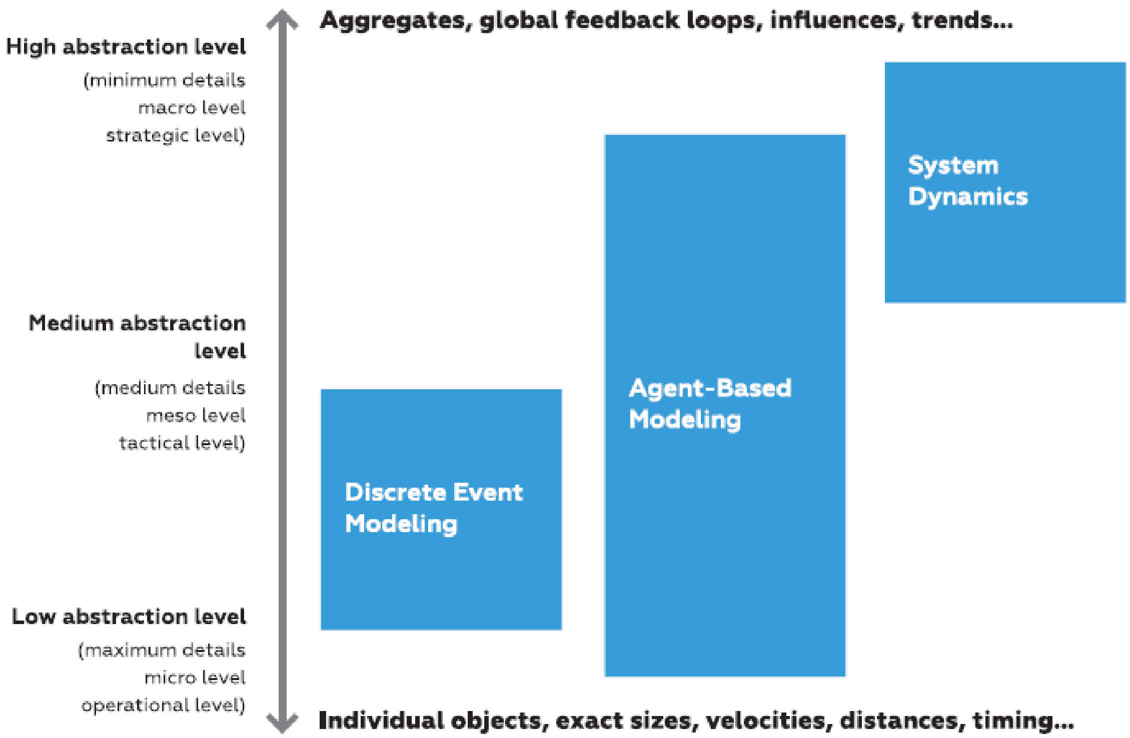


Figure 9 Three approaches to modelling of mining environment (Anylogic)

The analytical approach has remained essentially intact for nearly 400 years, but systems thinking has gone through three distinct generations of change (Garajedaghi, 2011):

- The first generation of systems thinking (operations research) dealt with the challenge of interdependency in the context of mechanical (deterministic) systems.
- The second generation of systems thinking (cybernetics and open systems) dealt with the dual challenge of interdependency and self-organization in the context of living systems.
- The third generation of systems thinking (design) responds to the triple challenge of interdependency, self-organization, and choice in the context of sociocultural systems.

The above is best described by the following image in Figure 10 where there are two directions of the paradigm shift.

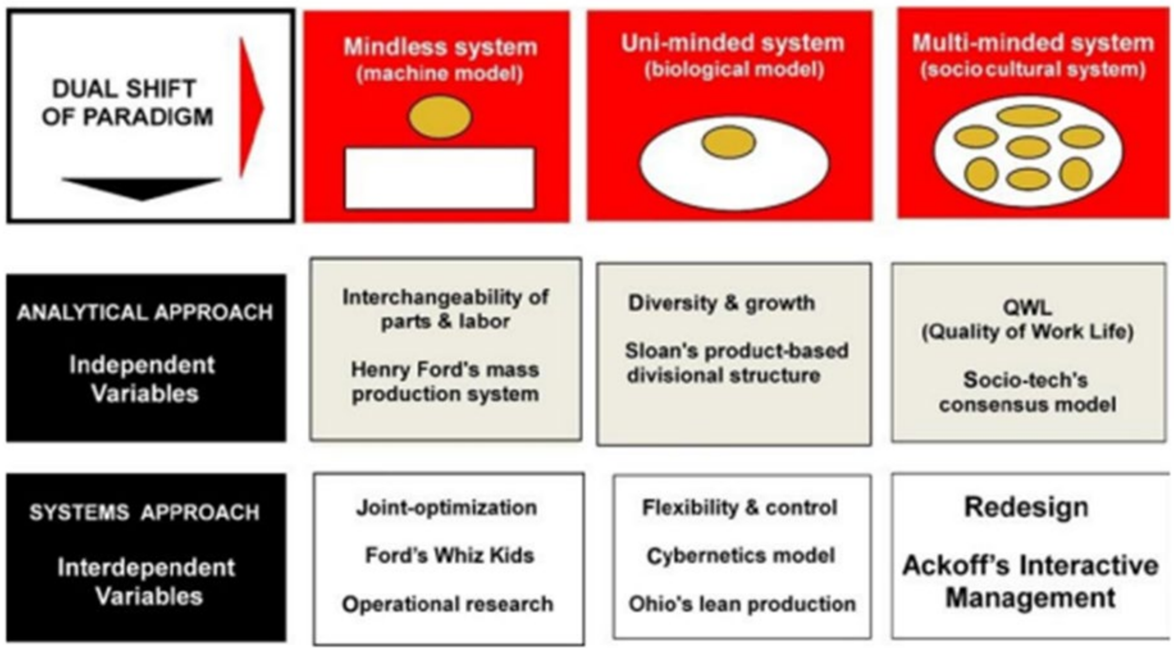


Figure 10 Paradigm shift in systems thinking concept (Gharajedaghi, 2011)

- Determination of activities/processes,
- Determination of policies,
- Determination of starting and end points of activities,
- Determination of flow rates of processes,
- Determine intended changes,
- Determine unintended changes.

As Popper (1972) puts it, “the activity of understanding is essentially the same as that of problem solving”.

2.3 Dynamic Synthesis Methodology

An effective “system dynamics research method” and tool should be able to capture both informal and fuzzy concepts relevant to the theory or models of interest. Simulation modelling is a form of laboratory experimentation with elevated levels of constraints. Visala (1991) contributes to the debate by proposing a conceptual framework to help overcome the gap between positivist and interpretative research approaches. (Williams, 2002). Williams describes research methodology called “Dynamic Synthesis Methodology (DSM)”, which can identify the behaviour of the inherent system by

Analysing:

- focuses on structure and reveals how things work,
- yields knowledge,
- enables description,
- looks into things.

And by synthesis:

- focuses on functions, it reveals why things operate as they do,
- yields understanding,
- enables explanation,
- Looks out of things.

Williams (2002) further states that an effective "system dynamics research methodology" should be able to capture both informal and fuzzy concepts to the models of interest. Combining qualitative and quantitative research methods is said to increase the robustness of results. In addition, a case study will add further value to test correctness of assumptions. Therefore, integration of simulation modelling and case study research methods provides a conceptual framework for a dynamic synthesis research methodology (DSM). The research method by Williams is summarized in Figure 11 Williams (2002) system dynamics modelling approach as well as in Figure 12 Dynamic Synthesis Methodology Research Design (Williams, 2002).

2.4 Top-Down-Bottom-Up Framework

According to Wirsch (2004), while bottom-up examines the data to uncover patterns, a top-down method approaches a problem from the high-level view of a system. His proposed approach of top- down- bottom-up framework, which can be seen in Figure 13 is a combination of both top- down and bottom-up approaches.

In addition, a problem articulation is considered as bounds of a system is an important stage. If there are no bounds then, modelling an all-encompassing system could not properly function. In this thesis, Dynamic Synthesis Methodology (DSM) will be followed where a top-down bottom-up approach is to be adopted as described by Wirsch (2004).

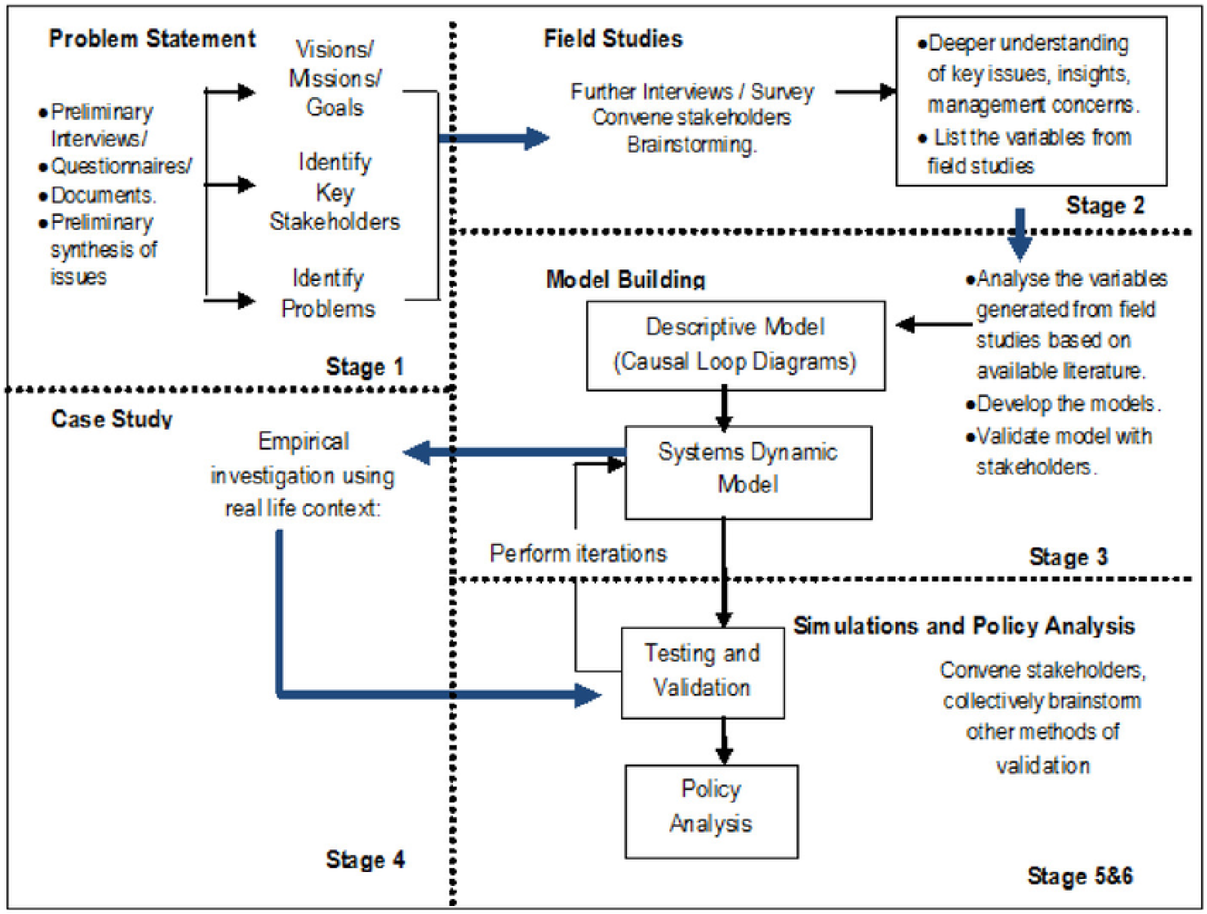


Figure 11 Williams (2002) system dynamics modelling approach

Wirsch (2014) further elaborates on dynamic hypothesis that there are three categories of data or input: endogenous, exogenous and executed. A summary of the Wirsch' s Top-Down- Bottom-Up Framework is mapped in Figure 13.

In a top-down approach, concept definition activities are primarily for understanding a problem by looking at the operational needs/requirements within the problem space and conditions or constrains within the solution space. The activities typically consider functional, behavioural, temporal, and physical aspects.

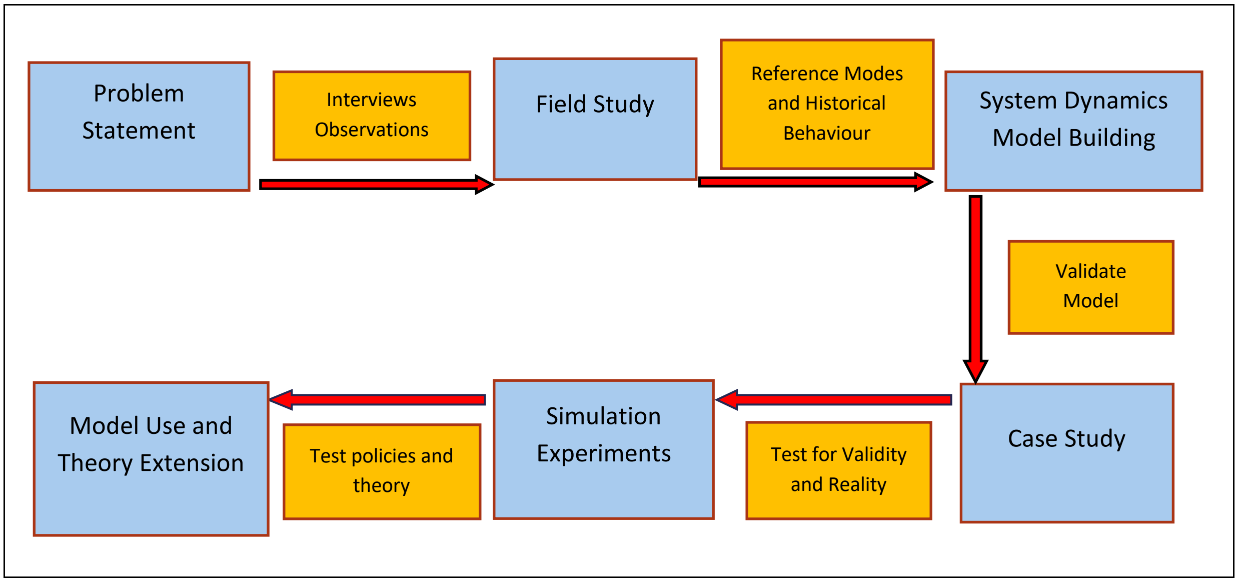


Figure 12 Dynamic Synthesis Methodology Research Design (Williams, 2002)

Although DSM calls for the field study to model historical behaviour this study will not do interviews but will use observations from archives found in literature due to time and security constraints.

2.5 Chapter 2 Conclusion

In system dynamics, a bottom-up approach can be used for determining the need to evolve existing capabilities or add new capabilities to an existing system. If automation is added as a new capability to an existing system, then this would be the ideal approach for the purposes of this research. A bottom-up approach is necessary for analysis purposes, or for re-using existing elements in the design architecture.

In summary a top-down approach provides new designs or architectures while bottom-up approach will analyse various scenarios. Therefore, a combination of both approaches may provide the right mixture. This is called “middle-out” approach (Sebok, 2019).

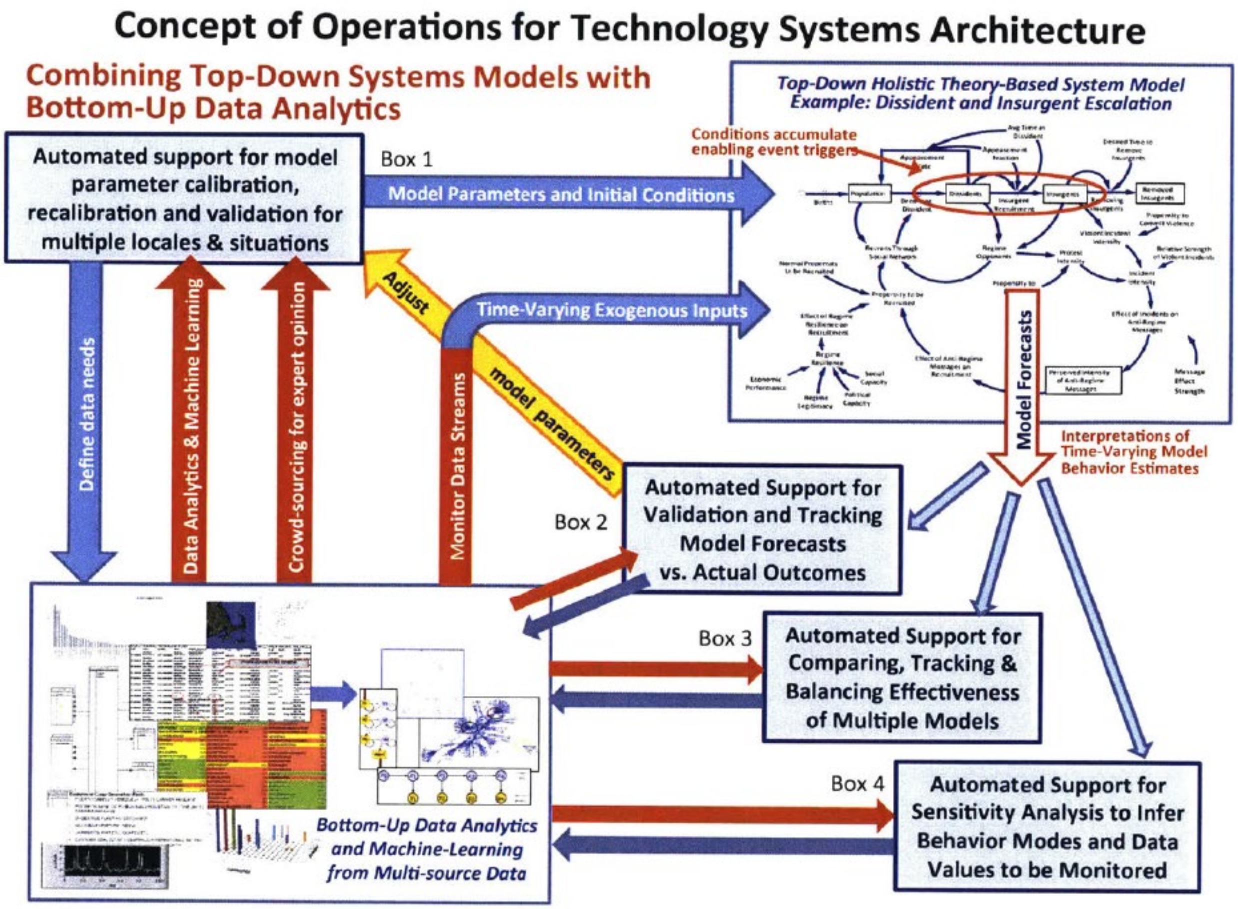


Figure 13 Wirsch' s Top-Down- Bottom-Up Framework (Wirsch, 2004)

In conclusion, system dynamics modelling approach with a top-down-bottom-up design framework is to be adapted in this thesis while using VENSIM as the system dynamics modelling tool. In the next chapter literature review will expand in support of the objective of this research.

If thinking is an art of mining the reality, then the correct way to mine is to think in systems"

Chapter 3

3 LITERATURE REVIEW

Literature review in this thesis is carried out at all levels of the research mainly to understand:

- Historic approaches to quantification of mining systems and the mine value chain,
- Modelling tools such as VENSIM as a system dynamic modelling tool,
- how system dynamics is applied to mining environment.

3.1 Definitions

Below are some definitions to improve the quality of comprehension of this chapter:

A system is a set of elements which exhibit sufficient cohesion, or "togetherness", to form a bounded whole (Hitchins 2007; Boardman and Sauser 2008).

Modelling is the process of producing a simplified representation of complex system of interest (Shishvan, M.S. and Benndorf, J. 2017)

A simulation is the imitation of the systems model operation. It is used to answer what if scenarios or questions. It can be used before an existing system is changed or a new system is built to reduce the chances of failure to meet specifications, to eliminate unforeseen bottlenecks, to prevent under or over utilization of resources and to optimize system performance. (Sebok, 2019)

A simulator can be used as a device to replicate the operational features of some system.

Deterministic model: In deterministic models, the output of the model is fully determined by the parameter values and initial conditions. (Sebok, 2019)

Stochastic models: These models possess some inherent randomness. The same set of parameter values and initial conditions will lead to an ensemble of different outputs. (Sebok, 2019)

Gaming Theory: Game theory is a theoretical framework for conceiving social situations among competing players. In some respects, game theory is the science of strategy, or at least the optimal decision-making of independent and competing actors in a strategic setting. (Investopedia.com)

System Dynamics: Sterman (2010) states that side effects are perceived as if they are reality, but they are just effects. When an action is taken the effects are considered beneficial if intended in the beginning. The results that are not anticipated are considered side effects since it was not thought in advance. Then the decisions taken lead to new decisions taken as the un-anticipated results are observed. These actions are a direct result of ineffective policies.

3.2 What is System Dynamics

Sterman (2000) describes complex systems as dynamic that is what appears to be unchanging over time is in actual fact varying. It is complex because, everything else is connected to one another (tightly coupled), governed by feedback, nonlinear, history dependent, self-organizing, adaptive, counterintuitive (cause and effect are distant in time and space), seeking solution in symptoms rather than underlying cause, policy resistant (many seemingly obvious solutions worsen the situation), characterized by trade-offs (long run response of a system to an intervention is often different from its short-run response).

Our brains' cognitive maps and ability to use them tend to process it in a linear fashion rather than nonlinear. In addition to this tendency our ability to analyse and simulate data is limited to several data points and variability of the data is not felt in the judgement which is often biased and full of errors.

Forester (1961) defined system dynamics as "the investigation of the information-feedback characteristics of managed systems and the use of models for the design of improved organizational form and guiding policy".

Wolstenholme (1990) defined it as "a rigorous method for qualitative description, exploration and analysis of complex systems in terms of their processes, information, organizational boundaries and strategies, which facilitates quantitative simulation modelling and analysis for the design of system structure and behaviour".

Coyle (1996) states that none of the above definitions is completely satisfactory. He states that Forrester does not say what type of models are involved, and he states that neither Forrester nor Wolstenholme's definitions refer to time, which is somewhat difficult to except as truth, based on Sterman (2000)

Coyle (1996) gives the definitions as follows: "System dynamics deals with the time-dependent behaviour of managed systems with the aim of describing the system and understanding, through qualitative and quantitative models, how information feedback governs its behaviour, and designing robust information feedback structures and control policies through simulation and optimization."

A structured approach to system dynamics analysis is described by Coyle (1996). A summary is presented by Coyle in four stages:

1) First stage: Recognize the problem and find out who cares about this problem.
2) Second stage: An influence diagram (also called a causal loop diagram) is to be constructed as a diagram of the forces at work in the system, which appear to be connected to the phenomena underlying people's concerns about it.
3) Third stage: This is a qualitative analysis stage. It is normally carried out by closely looking at the influence diagram in the hope of understanding the problem better. It has been stated that the most important stage is this, an incorrect influence diagram that did not capture the most critical influencer will result in inaccurate results. The problem sometimes is solved at this stage and no need to move to the fourth stage.
4) Fourth stage: The construction of a simulation model. The influence diagram or causal loop diagram is drawn at various levels of aggregation, and it is necessary to show every single detail.

3.3 Mathematics Used in System Dynamics

System dynamics is stemming from control theory and the modern theory of nonlinear dynamics. Mathematics behind System Dynamics is elegant and is also based on industrial engineering concepts such as strategic thinking, operations management, etc. Many managers do not think in mathematical terms such as nonlinear differential equations or even calculus or have forgotten about it. Also, diversity of technical managers and their approach to problem solving is high (Choopojcharoen and Magzari, 2012)

Fear not, says Sterman (2000) since system dynamics have tools developed that use the high mathematics in the background but in the foreground, all seems logical, and causality is traceable with visual tools used to define the environment.

3.4 System Thinking Tools

There are various tools available that can be used during simulation modelling of reality and a comprehensive list has been found at the following site:

https://www.burgehugheswalsh.co.uk/systems-thinking/tools.aspx

Kasser mentions in his book (2019) that there are five layers to a system engineering as listed in Table 3 which was first mentioned in Hitchins in 2000 (Kasser, 2010, pg. 172.). Kasser also listed most of the tools available to system engineers. They are summarized in this section.

Table 3 System or product layer complexity (Kasser, 2019)

Layer 1	Product Level	Many product subsystems or products make a system that are tangible artefact level
Layer 2	Project/system Level	Many projects make a business
Layer 3	Business system Engineering	Many businesses make an industry. At this level system engineering seeks to optimize performance somewhat independent of other businesses.
Layer 4	Industrial Systems Engineering	Many industries make a socio-economic system. A global wealth creation philosophy
Layer 5	Socioeconomic	The layer of regulation and governmental control

A summary of these tools will be described in the following sub sections.

18 WORD STATEMENT

18-word statement is a simple tool that defines the purpose and context of a system.

In this thesis the following statement can be made in 18 words: Quantifying the downstream effects of technological system changes at a surface mine by using system dynamics simulation techniques.

AFFINITY DIAGRAM

Affinity tool is about generating ideas about a situation or problem. We need to be more holistic and consider the whole situation or problem. Achieving this logically comprises two activities

1. Generating information
2. Organizing information

These two activities require different mental skills, and they are referred to as divergent and convergent thinking.

Divergent thinking includes brainstorming, similarities and differences, spray diagram and affinity diagram. Convergent thinking is concerned with organizing and making sense of ideas or information, affinity diagram, tree diagram and multiple cause diagram.

The affinity stage is throwing in all ideas with sticky notes (word cloud) and then the convergent stage is organizing them into classes.

CONCEPTUAL MODEL

A conceptual model is a diagrammatic representation of what logical activities need to be done to achieve the aim/purpose. It should be describing future states but deduced logically to achieve the purpose. In order to successfully reach a solution all constraints, preferences, alternatives and information need to be considered.

DECISION MATRIX

It is done only after all possibilities are identified before reducing to a final solution. All the tools discussed earlier around divergent thinking (problem) and convergent thinking (solution) are used.

FUNCTIONAL FAILURE MODE AND EFFECTS ANALYSIS

Functional Failure Mode and Effects Analysis (FFMEA) is a tool that allows a team to systematically identify, document and prioritize functional failure modes, their effects and causes. At this stage getting bogged down in too much detail or get side-tracked is possible.

The way to do it requires the following steps:

Step 1: Identify and list the system functions,

Step 2: For each function identify potential failure modes,

Step 3: For each failure mode identify effects experienced by the user,

Step 4: For each failure mode identify causes,

Step 5: For each failure mode identify current detection methods employed, Step 6: Rate probability of occurrence, severity, and probability of detection,

Step 7: Determine RPN (Risk Priority Number),

Step 8: Consider high RPN for new functionality or design ideas.

FUNCTION MEANS ANALYSIS

"Function Means Analysis tool is a highly structured approach to generating, selecting and documenting system design concepts. The basic idea is to consider the functions that the system performs and identify all the means of achieving that function".

This tool is used during the design process. Function "means" table can get complicated and levels are not always determined easily hence requires full understanding of the system. It is not very practical for the problem at hand.

FUNCTIONAL MODEL

Functional Modelling is a tool that allows to produce a behavioural/operational model of an existing or planned system. The model shows system functionality with logical interconnections. By constructing the model, it is possible to

- Deduce the necessary system functionality.
- Test the basic operational concept.
- Determine potential system interfaces.

It is a network representation of the system. It helps one understand how the system works rather than modelling technique used.

Functional Modelling uses three sub models which are Functional Flow Diagram (FFD), Flow Dictionary (FD), and Function Specification (FS). FFD is a network representation of the system in terms of component functions and the logical interdependencies with no modelling technique. FD sets definitions of component elements which become real interfaces later in the model. FS specifies the component functions by converting inputs to outputs which later are represented with formulae that define them.

Constructing a set of Functional Flow Diagrams is often not easy due to the complexities. It often requires to be modified. Constraints need to be captured at this level. Functional specification needs to be included. This tool will be extensively used in this thesis (see sections 5, 6 and 7).

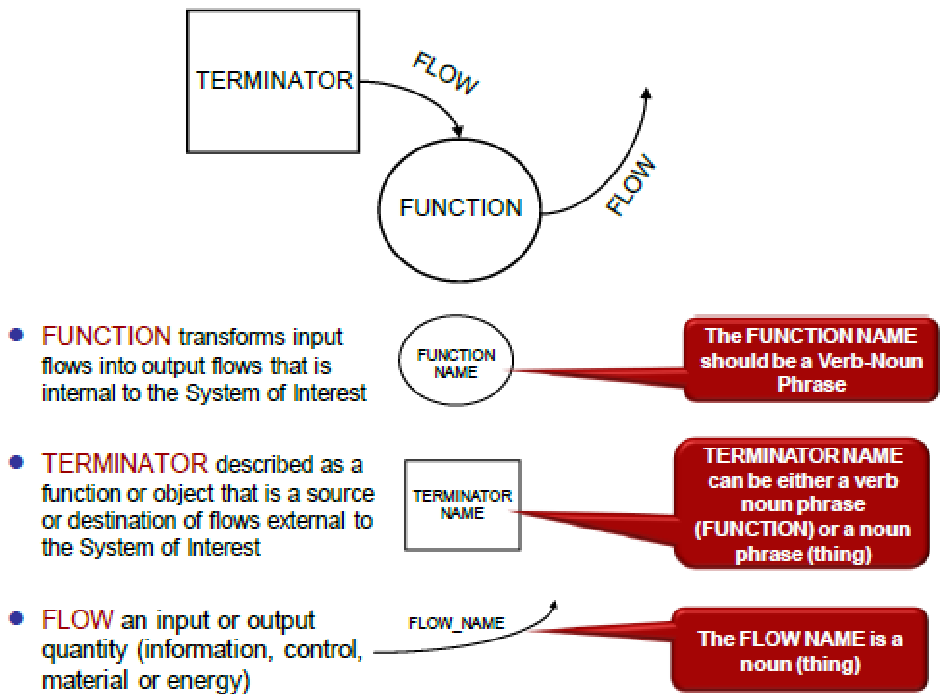


Figure 14 Function flow diagram modelling conventions (Burgess, 2011)

GRAPHICAL ANALYSIS

Graphical analysis is about determining the nature of variation in a system. This could be done via frequency plot or time series plot.

A key aspect of understanding a system is looking for behaviour patterns. Sometime the complexity of the system masks a pattern, which is not recognized by simply looking at one pattern which could be hiding issues or opportunities. Figure 15 shows levels of variation scenarios.

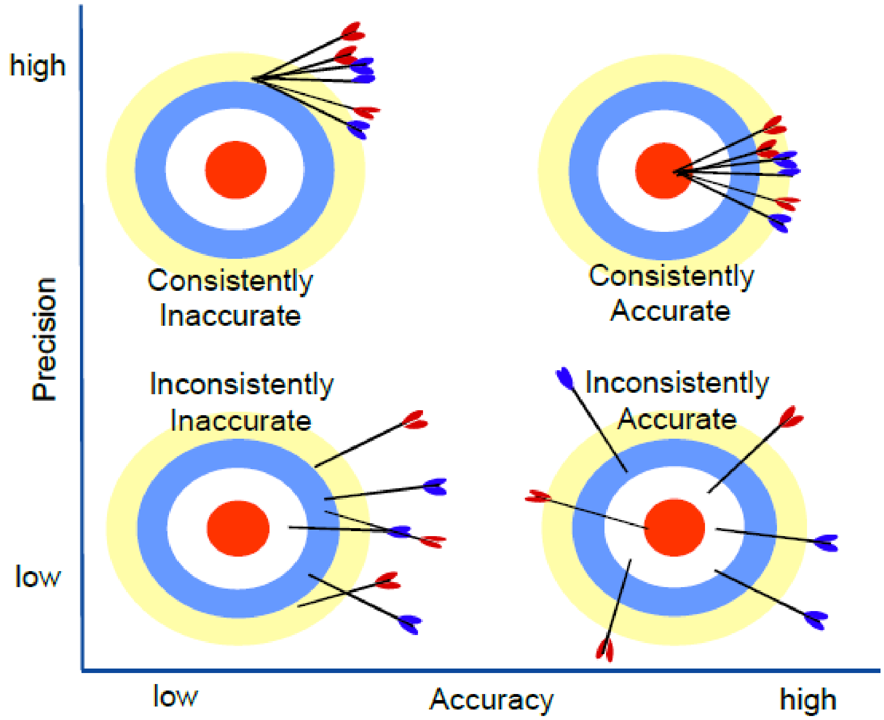


Figure 15 Variation scenarios (Kasser, 2010)

Frequency plots can take statistical distributions looking at variations within a system. The wider the variability the higher the inconsistency. Six sigma tools are useful in this analysis tool. Variations in the system "shifts the burden".

Variation can detect a change in one of the system inputs or a change in the internal workings of the system. In all cases it was assumed that the data captured is accurate.

There are two types of data Discrete or Attribute. Discrete hides the data as it is often represented with percent values of ratios or counts. Attribute data tells the truth.

In this research where large amount of data may exist, then it is necessary to rely on discrete data rather than attribute data. Some critical design parameters can be represented in actual form. Output can be converted to discrete to be used as input in the next process.

INFLUENCE DIAGRAM/CAUSAL LOOP DIAGRAMS

An influence diagram is a tool for identifying the important relationships or influences that exist between the elements of the system. It is possible to do an influence diagram for a complete organization, it is doable but not recommended. The rest of the model can be drawn at a detailed level. Each element on the high-level system can only be detailed where influence is high impact. An example of an influence diagram from Craig et al (2018) is shown in the Figure 16 below.

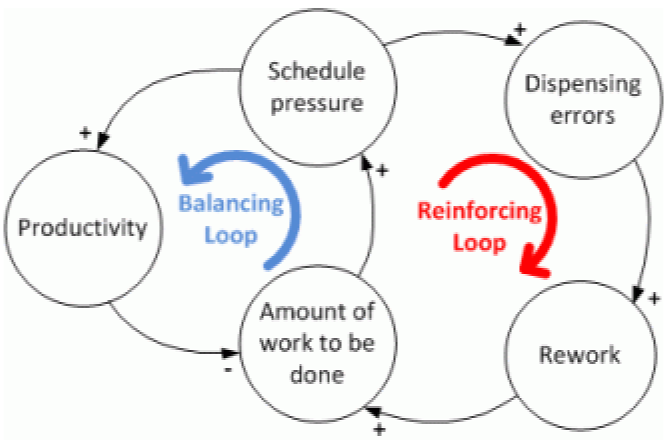


Figure 16 Causal loop diagram (CLD) (Shire, 2019)

An Input-Output diagram is a simple high-level representation of a system that shows major inputs and their suppliers, the major outputs to customers and components that are necessary to achieve the system's purpose.

At this stage what happens between inputs and outputs are not shown and becomes necessary only later. Outputs can be physical production outputs, documents, information or waste. Waste can be in many forms, i.e., production waste, time waste, money waste.

Typically, input can be people, materials, equipment, methods, environmental information.

MATRIX DIAGRAM

A Matrix Diagram is a tool that allows to identify the presence and strength of relationships between two or more list of items. The context of the lists that are being matrixed are data, information, functions, concepts, actions, people, material, equipment, etc. There are L-type, T-type, Y-type, X-type, C-type and QFD-type matrixes.

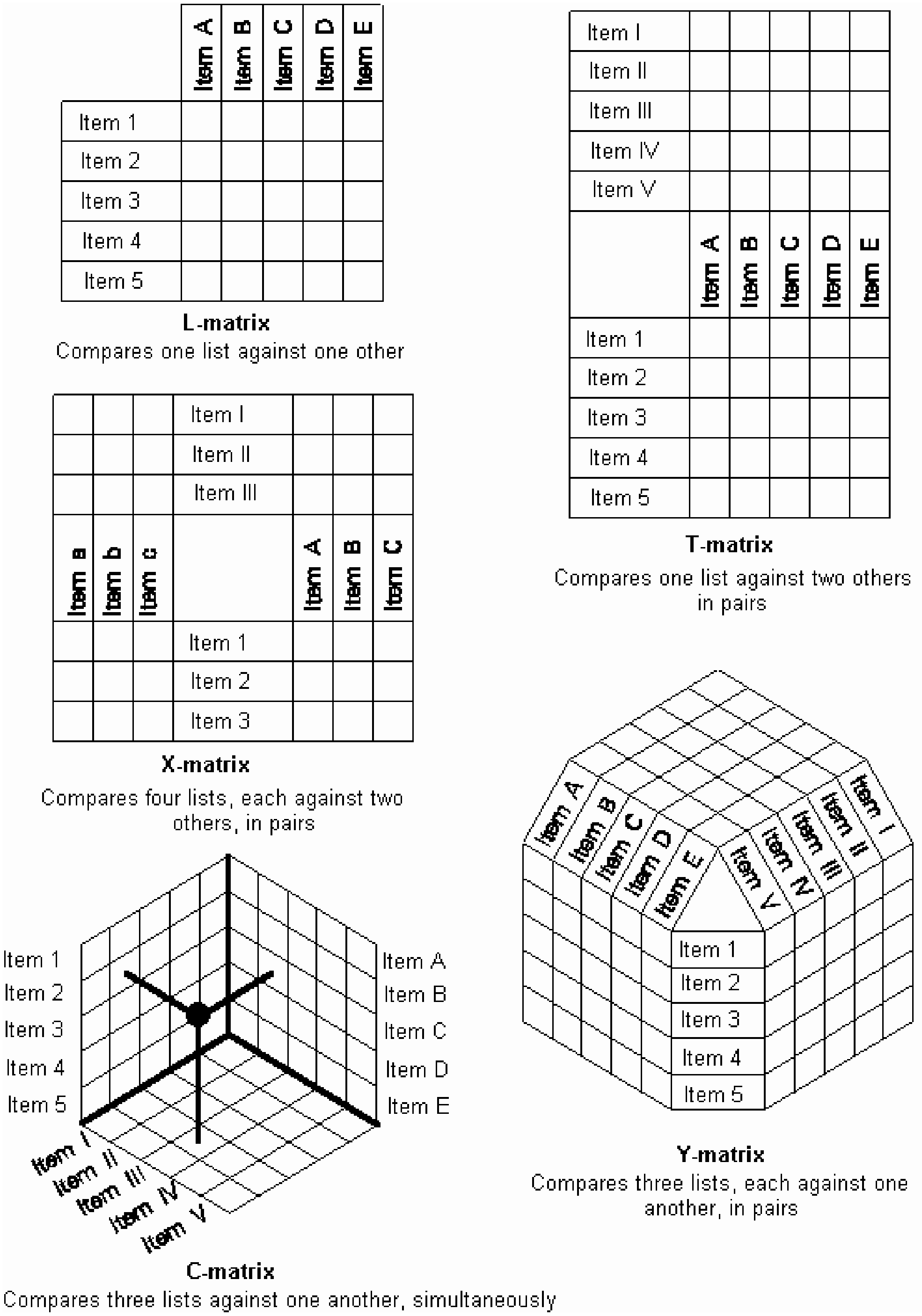
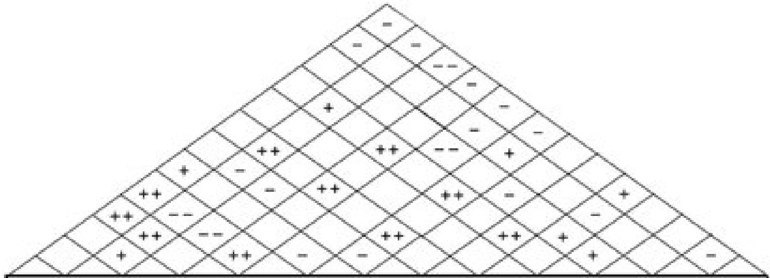


Figure 17 L, T, X, Y and C types of matrix (Kasser, 2019)



	Customer Requirements / Needs	Relative Importance	%-Area of Recycled Materials	Area of Opening	Exterior Volume	Collector Plate Area	Price of Unit	Reflector Present	Set up Time	Temperature Gain Rate (30 min)	Removable pieces	Time to Reach Pasteurization Temperature	WAPI	Weather Resistant	Weight
			Engineering Metrics												
End User	Low Cost	9	**9**		**3**		**9**	**3**		**3**		**1**	**3**	**1**	**1**
	Reaches Cooking Temperature Quickly	9				**9**		**3**		**9**					
	Able to Pausterize Water and Measure	9										**9**	**9**		
	Easy to Use	3		**1**					**9**						
	Durable	3												**9**	
	Capable of cooking large meals	3		**1**		**9**									
	Easy to clean	1		**3**		**1**					**9**				
	Light-Weight	1			**3**										**9**
	Easy to Store / Portable	1			**9**										**3**
	Made of eco-friendly materials	3	**9**												
Measurement Units			%~in^2	in^2	in^3	in^2	USD	y/n	sec	$^{\circ}C/_{min}$	y/n	min	y/n	y/n	lb
Object Target Values			20	200	4500	250	35.00	y	26.2	1.00	n	80	n	y	20.0
Raw Score			108	9	39	109	81	54	27	108	9	90	108	36	21
Relative Weight (%)			13.52	1.13	4.88	13.64	10.14	6.76	3.38	13.52	1.13	11.26	13.52	4.51	2.63
Actual Values for Generation III			35.6	270.25	4557	279.06	32.68	456.75	24	1.43	n	95.0	y	y	38.5
Meet Criteria (y/n)			y	y	y	y	y	y	y	n	y	n	y	y	y
Engineering Metrics Benchmark	Unit A		0	324	5989.5	255	220.00	934.97	25	n/a	n	n/a	n	y	26
	Unit B		0	433.5	3786.8	209	121.94	0	21.67	1.83	n	78.2	n	y	11
	Unit C		0	210.25	4201.8	159.5	210.85	1350	30.67	n/a	n	n/a	n	y	21

Figure 18 QFD Matrix (Thorn and Carrano, 2006)

Quality Function Matrix Diagram (QFD) is basically an L type matrix with ancillary list that has a many to one relationship with one of the main lists. It is used to investigate the relationships between sets of requirements that are developed during new system introduction.

MORPHOLOGICAL BOX

It is a creative thinking tool for generating whole solutions to complex problems where possible solutions can be explored. It is mostly useful for re-design or inventions.

MULTIPLE CAUSE DIAGRAM

Multiple Cause Diagrams are similar in nature to Cause and Effect Diagrams (sometimes called 'Fishbone Diagrams' or 'Ishikawa Diagrams') but allow more freedom to represent the various levels of contribution to the overall matter of interest. It is like fish-bone diagrams. Arrows between elements indicate causal paths, showing the direction of causality.

N2 ANALYSIS

N^2 analysis uses an NxN matrix to record the interconnections between elements. It helps understand the system design. In addition, helps determine critical elements, which may cause complete failure, i.e., candidate of redundancy. See the example in Figure 19.

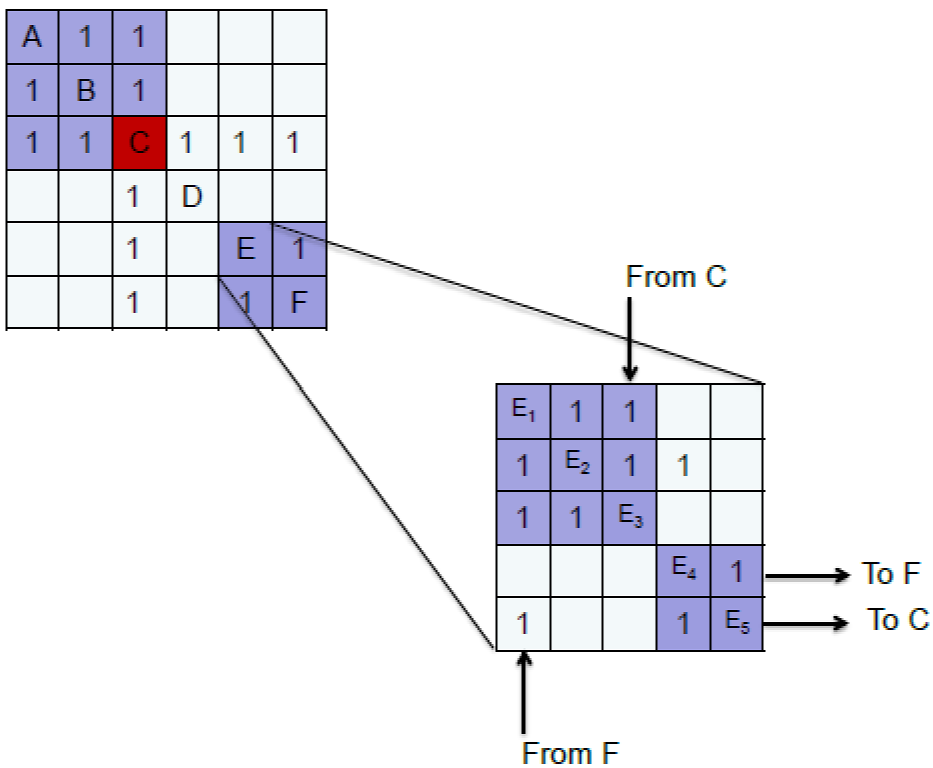


Figure 19 N2 analysis with top down application (Kasser, 2019)

N^2 analysis can be a very powerful tool to analyse subsystems within systems where horizontal squares are outputs and vertical squares are inputs. The interaction matrix can be used as a thinking tool to construct a system using parameters and interactions between those parameters.

The key rules of constructing N^2 matrix require identification of the system functions or elements, which are all required to be on the diagonal. One drawback of this tool is that a software is not available to use this method, therefore, clustering of elements is better done manually.

QUAD OF AIMS

This is a simple tool that can help clarify aims and objectives therefore determine smarter aims. The SMART acronym stands for Specific and Succinct, Measurable, Achievable, Relevant and Time bound. This tool is being used to clarify of this research's aim and objective.

RICH PICTURE

Rich picture as the name suggests is about visualizing with simple sketches of the elements of the system and the interaction between them.

ROOT DEFINITION

The root definition can be used when investigating a situation to obtain a common and clear understanding of the system being studied. It involves customer, actors, transformation of input to output, world view, owner and environmental constraints (outside the system boundary).

SEQUENCE DIAGRAM

A sequence diagram is a schematic of the system being studied and very high level. It helps clarify engineering requirements of a new system or analysing the existing system. It can be used to determine boundaries; it can also be abstract focus of input-output transformation. Therefore, it is seen as part of a divergent thinking tool.

SPRAY DIAGRAM

Spray diagrams are useful when it comes to exploring possibilities and prevents the modeller from jumping to conclusions or quick answers.

An example of the system diagram is shown below which also explains why spray diagrams are used (Figure 20).

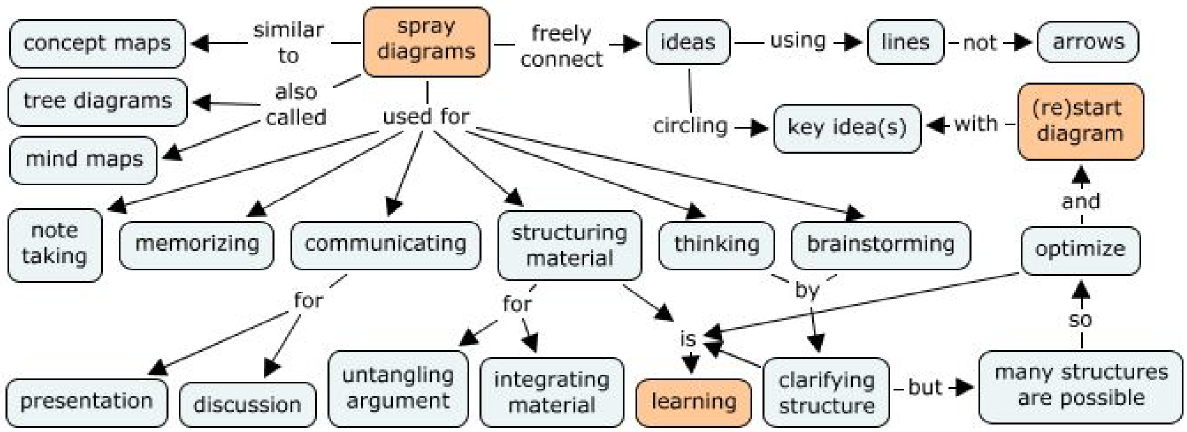


Figure 20 Spray diagram explained in the form of spray diagram (Kasser, 2019)

SYSTEMS MAP

Systems map is a conceptual map of all elements interacting with boundaries defined, indicating major components, indicating major items in the environment, and identifying major subsystems between components. A simple system thinking map of a mine has been used here as an example (Figure 21).

TREE DIAGRAM

A tree diagram is mainly used to analyse components of a system, such as parts of a bicycle or hierarchical analysis of a system such as human resources. The way to construct a tree diagram is by starting at highest level with a goal or objective, task or activity, thing or a product, need or requirement or problem/issue. This level is at a "What" level. The next step is to answer how. The sequence of "whats" and "hows" alternate as the tree grows.

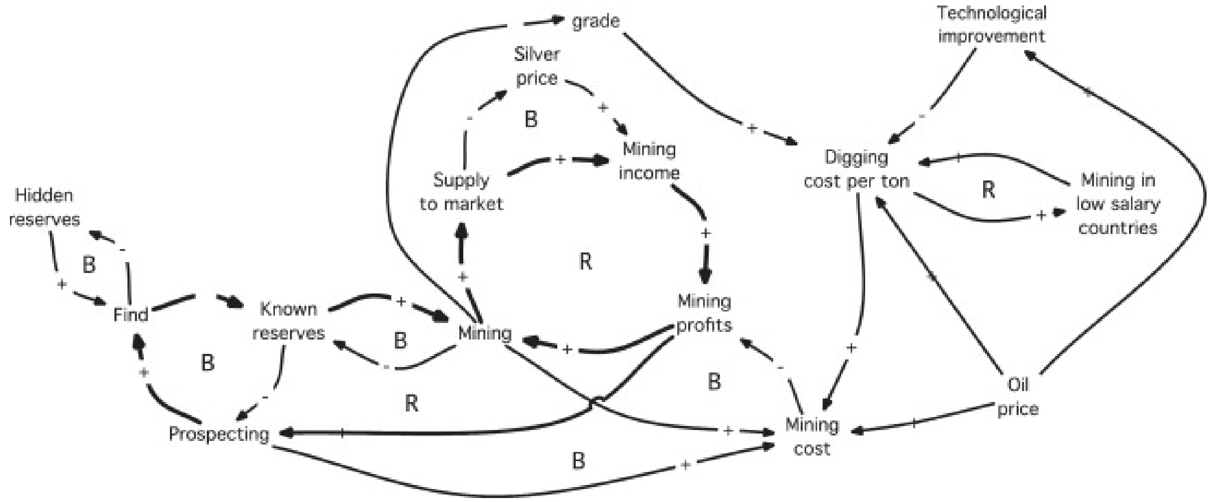


Figure 21 Systems map of a mine (Harald et al, 2014)

In summary and in addition to the above a holistic thinking perspective (HTP) may be required. HTP's are nine in total, that are, "big picture", "operational"," functional", "structural", "generic", "continuum", "temporal", "quantitative" and "scientific" (Kasser, 2019). The definitions of HTPs are given in Table 4.

Table 4 Nine holistic thinking perspectives: HTPs (Kasser, 2019)

Big Picture:	includes the context for the system, the environment and assumptions.
Operational:	what the system does as described in scenarios; a black box perspective.
Functional:	what the system does and how it does it; a whitebox perspective.
Structural:	how the system is constructed, and its elements are organized
Generic:	perceptions of the system as an instance of a class of similar systems; perceptions of similarity.
Continuum:	perceptions of the system as but one of many alternatives; perceptions of differences. For example, when hearing the phrase "she's not just a pretty face",* the thought may pop up from the Continuum HTP changing the phrase to "she's not even a pretty face",† which means the reverse.
Temporal:	perceptions of the past, present and future of the system.
Quantitative:	perceptions of the numeric and other quantitative information associated with the other descriptive HTPs.
Scientific:	insights and inferences from the perceptions from the descriptive HTPs leading to the hypothesis or guess about the issue after using Critical Thinking.

3.5 Why a System dynamics Approach

"As a language, systems thinking has unique qualities that make it a valuable tool for discussing complex systemic issues" (Anderson, 1997). Some of these qualities as reported by Anderson are:

- It emphasizes looking at wholes rather than parts and it stresses the role of interconnections.
- It is a circular rather than linear language. In other words, it focuses on "closed interdependencies," where x influences y, y influences z, and z come back around to influence x.
- It has a set of rules that reduce the ambiguities and miscommunications that can crop up when explaining about complex issues to the others
- Visual tools, such as causal loop diagrams and behaviour over time graphs give better insights. They also facilitate learning because they are graphic and therefore are often easier to remember than written words. Finally, they emphasize on the dynamics of a problem.
- It allows translating our individual perceptions into explicit pictures that can reveal differences in viewpoints.

"Emergent" properties of the parts sometimes do not explain the result. The definition of "emergent" can be best explained by the question of "Do all facts determined by the basic facts as to be explainable (at least in principle) in terms of those basic facts?" (Mittal et al, 2018). It marks a single problem that can be stated very simply with this example of an all-star team may not mean in totality they will be superior in comparison to other teams. How they interact in a team has more meaning than individual stardom. In mining, the essence of the system is to manage the interactions rather than focusing on individual stardom of processes. The best example is where cost efficiency of blasting may have been aimed without considering a negative impact on other processes, which is a well-known cause and effect relationship in mining as explained by McKee (2013) and Thornton (2001), and mainly by Kanchibotla (2014, 1999). High productivity of drilling does not necessarily mean the optimal condition for the mine as it also depends on the quality. Reducing drilling and blasting costs may mean decreased drilling density therefore decreased cost of blasting but higher cost of loading and hauling as well as high processing costs per ton mined. Therefore, a mine should be examined in totality rather than analysing individual processes.

3.6 System Dynamics Modelling Process

There are six important steps in building a system dynamics model. It starts with the problem identification and definition, followed by system conceptualization, model formulation, model testing and evaluation, model use, implementation and dissemination and design of learning strategy/infrastructure (Bala et al, 2017). They are all brought together in the following schematic (Figure 22) by Martinez-Moyano and Richardson (2013).

The summary of system dynamics steps by Forrester (1994) may be useful in this research and is indicated as follows:

In step 1 the system is described. This stage is the most critical since understanding the problem always a challenge for modellers. Understanding comes first then the goal must be the improvement.

In step 2 formulation starts. This requires defining rates, equations, etc. that define the system behaviour.

It is said that writing equations reveals gaps and inconsistencies that must be remedied in the description of the process. There are two major areas of disagreement on how to formulate models. Starting small and continuously simulate and preferably, always have a running model is one way to define the behaviour. There is a disagreement between groups of experts where one group formulate piece by piece, always trying to have a running model at hand; and another group formulate in big chunks and is not concerned about continuously having running prototypes. This difference can be understood as a procedural difference in the modelling effort.

Another difference in approach is that some modellers may think the extreme condition tests are crucial while others do not" (Martinez & Richardson, 2002).

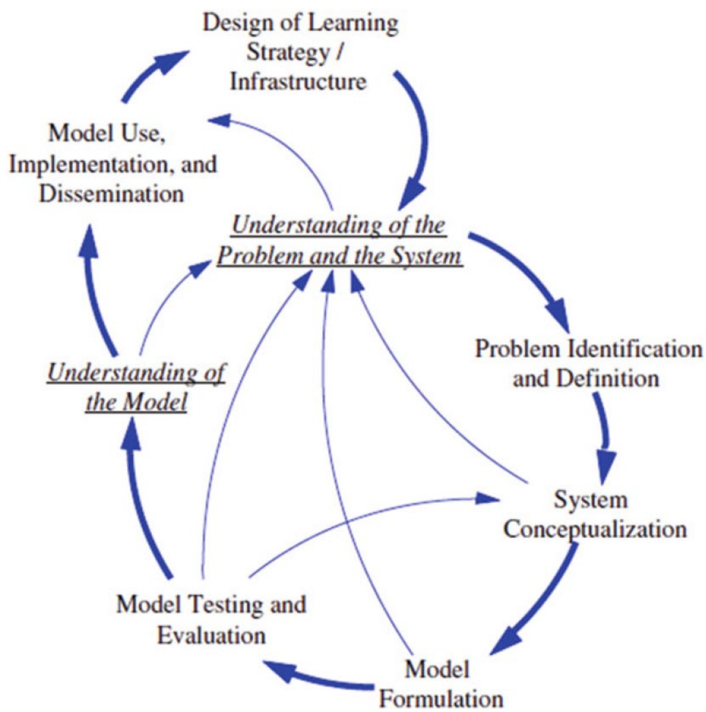


Figure 22 Overview of system dynamics modelling approach (Source: Martinez-Moyano and Richardson, 2013 as cited in Bala, 2017))

In Step 3 "simulation of the model can start after the equations pass a logical criterion of an operable model". During this stage logical checks may reveal unrealistic behaviour. This is followed by further checks and improvements on equations. This stage is where present is replicated up to now. If some improvement is required, then this model is further used to define improvement scenarios.

Step 4 identifies policy alternatives for testing identifying which policies have the biggest promise. I.e., automatic testing of parameter changes. This is the stage where some skill is required for imagining the most creative and powerful alternatives. It was suggested that the best alternative behaviour will often come from changing the system structure rather than parameters.

Step 5 is consensus towards implementation

Step 6 is implementation (Figure 23)

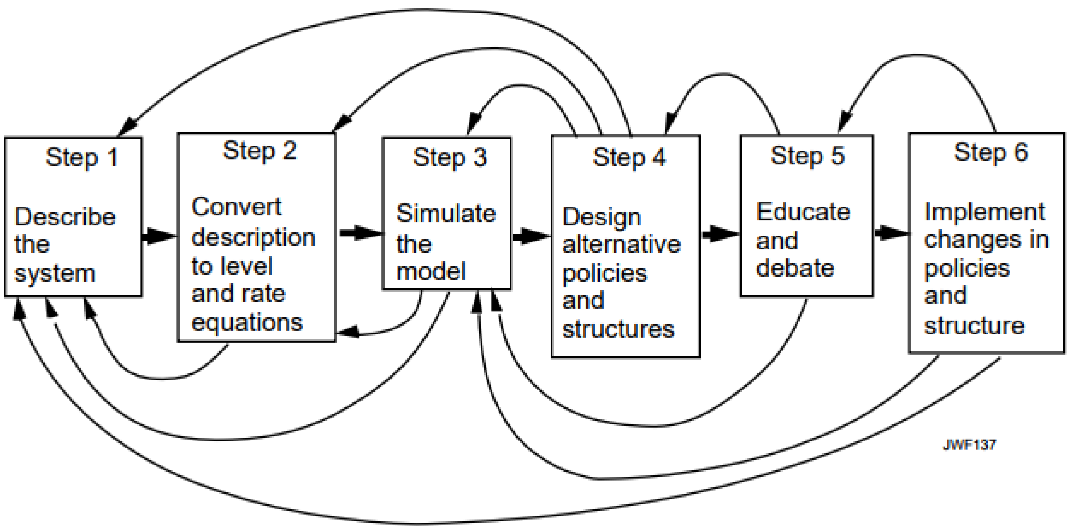


Figure 23 System dynamics steps from problem symptoms to improvement (Jay W Forrester 1994)

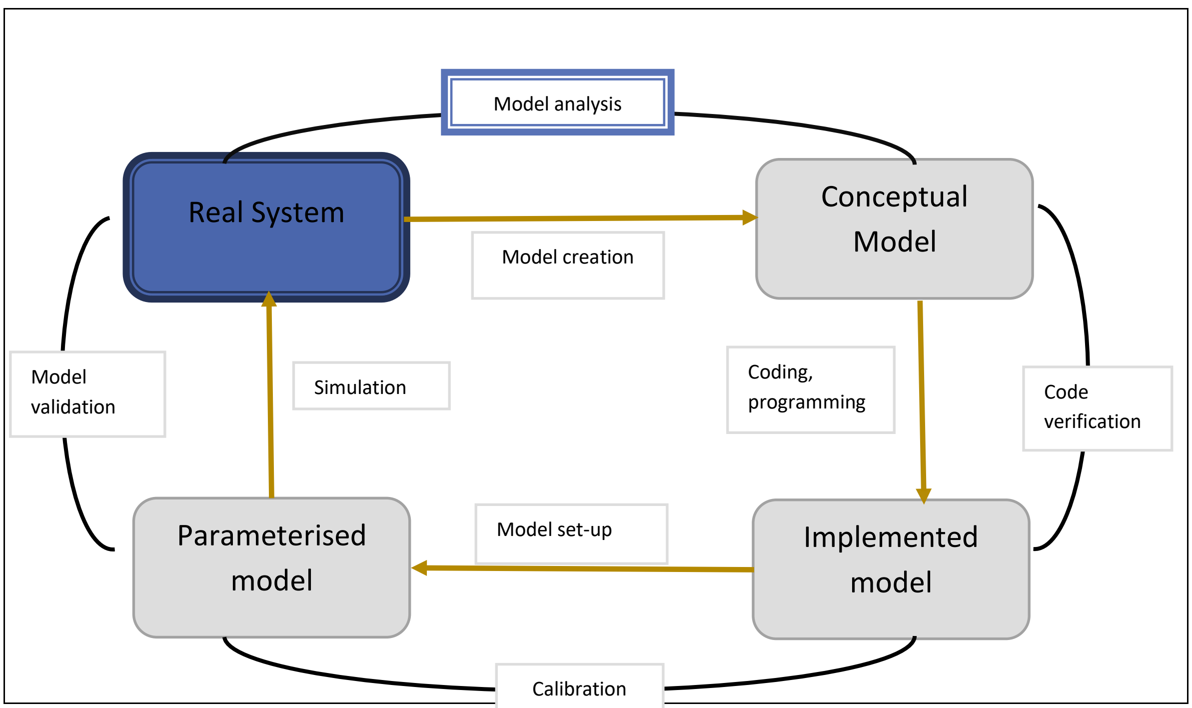
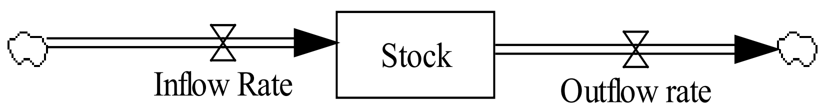


Figure 24. Modelling and simulation processes (Refgaard & Hendriksen, 2004)

3.7 Stock and Flow Diagrams (Influence Diagrams)

There are various modelling and simulation processes best described by Refgaard and Hendriksen (2004) as an iterative process (See Figure 24).

Systems thinking as a concept is explained with stock and flow diagrams and causal loop diagrams. As seen in the following diagram, a stock-flow diagram has parts such as the inflows (resources), rates in the shape of a valve (rate of production) and outflows (product or sales or can be input for the next process). The net rate of change in the stock is the difference of the outflow and the inflow, which is defined in mathematics with a differential equation with upper and lower limits. In the example below the differential is time based (dt) from t_0 to t (2). The rate depends on single or multiple factors and formulae for any of the components can be simplified to complex. (Sterman, 2010)



$$\frac{d(Stock)}{dt} = Inflow\ (t) - Outflow\ (t)$$ **Equation 2**

$$d(Stock) = \int_{t_0}^{t} (Inflow(t) - Outflow(t)dt + Stock(t_0))$$ **Equation 3**

Each variable is built with a formula.

Diagrammatic conventions that are used throughout the thesis are listed below.

- Solid Lines: Physical flows that are results of processes
- Broken lines are influences that are not physical flows
- A box denotes an external driving force over which the system has no control of and to which it must respond. For example, external temperature is a driving force that changes over time. Desired temperature on a thermostat is also driving force which is set by a human action. The central heating in a typical system is then going to respond to driving force to keep the temperature at the selected setting.
- A + sign means that when the variable at the tail of the arrow changes, the variable at the head always changes in the same direction. Thus, for example the greater the quantity of heat in the room the higher the temperature will be. On the other hand, if the quantity falls, so does the temperature.
- A - sign has the opposite effect: if the tail variable changes then the head variable changes in the opposite direction. Thus, the greater the rate of losing heat the less heat will remain in the room and, the smaller the rate of losing heat, the more heat will be left.
- It is not always obvious whether a parameter has a positive or negative effect on the variable it influences and, for that reason, signs on links from parameters are optional in an attempt to describe behaviour.

Here is a simplified problem also relevant to the current mining research: The mine will convert drill machines to automatic drilling to fill any shortfall between the number of drilled meters made available and the number needed to cope with the demand from the blasting. When all machines are converted to automated drill rigs there needs to be a marked improvement of quality and quantity of drilling. The method to model is described by Coyle in steps (1996)

- "The first step is to identify all separate entities or actors in the problem
- For each entity all the possible states in which members of that entity can be
- For each state identify flows which can cause the state to increase or decrease
- Check the connections between flows. Does the outflow from one state feed another?
- Ensure any delays in flows are represented, especially when relating the outflow from one state to the inflow to another
- Identify the controlling flow rates which drive the system. In general, there will have arrows coming out of them, showing that they influence something, but no arrows going to show what influences them
- Identify and represent the information and action influences on the controlling flow rates. This is usually done from the parsing of the narrative account."

In order to solve this simplified problem in system dynamics the choice of correct method of modelling is important.

3.8 Types of Models in Systems Engineering

There are three models identified in system dynamics engineering:

- Physical model
- Quantitative model
- Qualitative Model as explained in Pruyt (2013)

Mainstream system dynamists often start with qualitative SD, and then turn to quantitative SD. The results of which are interpreted in a qualitative sense and communicated using qualitative SD and that in a single intervention. There are nevertheless system dynamists who prefer qualitative SD modelling –mainly using CLDs and IDs, and sometimes ADs– over quantitative SD modelling. Supporters of purely qualitative SD modelling argue that if a close representation cannot be reached, the analysis should be limited to the qualitative level (Coyle 2000). Qualitative SD modelling is satisfactory if the 'insights from the diagram are convincing or the uncertainties surrounding the numerical data are so great that a quantified model may contain such uncertainties and inaccuracies that it is not worth the effort of building (Coyle and Alexander 1997, p206). Other arguments pro qualitative SD are that it is useful

(i) for describing a problem situation and its possible causes and solutions, potential risks (Wolstenholme 1999, p424) and uncertainties3, hypotheses and constraints,
(ii) to 'capture intricacies of circular causality in ways that aid understanding' (Richardson 1999, p441) (Wolstenholme 1999, p424),
(iii) as a medium by which people can externalize and share their mental models and assumptions (Wolstenholme1999, p424),
(iv) for the 'inference of modes of behaviour by assisting mental simulation of maps' (Wolstenholme 1999, p424),
(v) to show people the dynamic system they are part of, the strategic ramifications, and
(vi) to propose solutions (Coyle 2000).

It could create a common language and understanding of the structure and the feedback loops, reveal the big picture, hidden and different world views, hypotheses, constraints, structural problems (boundaries), uncertainties, threats, risks, opportunities, possible leverage points, policy variables and policy structures, and could therefore be a problem structuring and discovery tool. However, there are also good arguments against purely qualitative modelling and in favour of qualitative-quantitative-qualitative modelling, namely that as reported in Pruyt (2013):

(i) Maps are misleading (Richardson 1999, p441) and unreliable tools for behavioural inference (Rahmandad et al 2015)
(ii) They do not enable estimation of the scale or speed of change of key items (Rahmandad et al, 2015),
(iii) 'Feedback-based insights, especially those based on multiple loops of uncertain strength, can often be difficult things for people to understand and believe in' (Rahmandad et al, 2015),
(iv) They are 'less likely to lead to commitment, consensus or system changes than quantitative models' (Rouwette et al. 2002, p32), and
(v) They are but 'tools for hypotheses generation without the systematic approach to falsify the hypothesis' (Oliva 2003). (As reported in Pruyt)

Many system dynamists would argue that simulation nearly always adds value (Homer and Oliva 2001, p347) but demands a deeper and more rigorous analysis than qualitative mapping (Wolstenholme 1999, p424), and hence, also much more time and resources. The question to be asked in each and any case is thus whether the additional effort of quantitative modelling is justified given the time and resources available."

The following types of models are possible in SD environment:

- Functional *Model*
- Structural Model
- Process Model
- Object Oriented Model
- Mental Model:

3.9 Behaviour Modes

Feedback loops define the system behaviour and can be used for decisions that controls action which results in flow, the stock or level of the system.

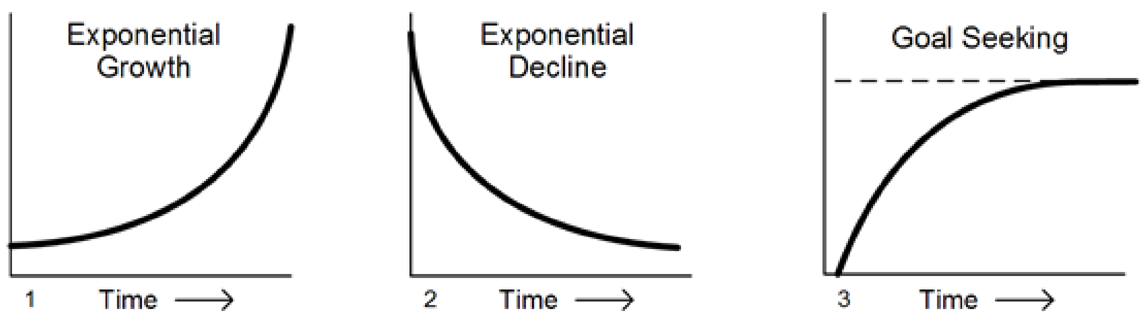


Figure 25 Simple feedback loop behaviours

In Figure 25 simple feedback loops are demonstrated that defines the output of a system, namely, positive feedback exponential growth, exponential decline and positive goal seeking.

Positive feedback loops show the behaviour of exponential growth or exponential decline. It must be noted that growth or decline are positive loops not because the outcome is positive but because it keeps increasing or decreasing either linearly or exponentially. Goal seeking types of behaviour describes a state where a capacity is reached, and no further increase or decrease is expected. However, simple unbounded growth is not the only behaviour a single loop is able to exhibit. Other behaviours are exponential decay, damped oscillation, sustained oscillation, and expanding oscillation. Each of these additional behaviour modes is unstable, and each will turn to exponential growth (Ashward, 2001). Further types of 2nd, 3rd and 4th order feedback loops can be found in the literature cited.

Wolstenholme (2003) also defined some more growth behaviours namely, s-shaped growth oscillation growth with overshoot and overshoot collapses as seen in Figure 26.

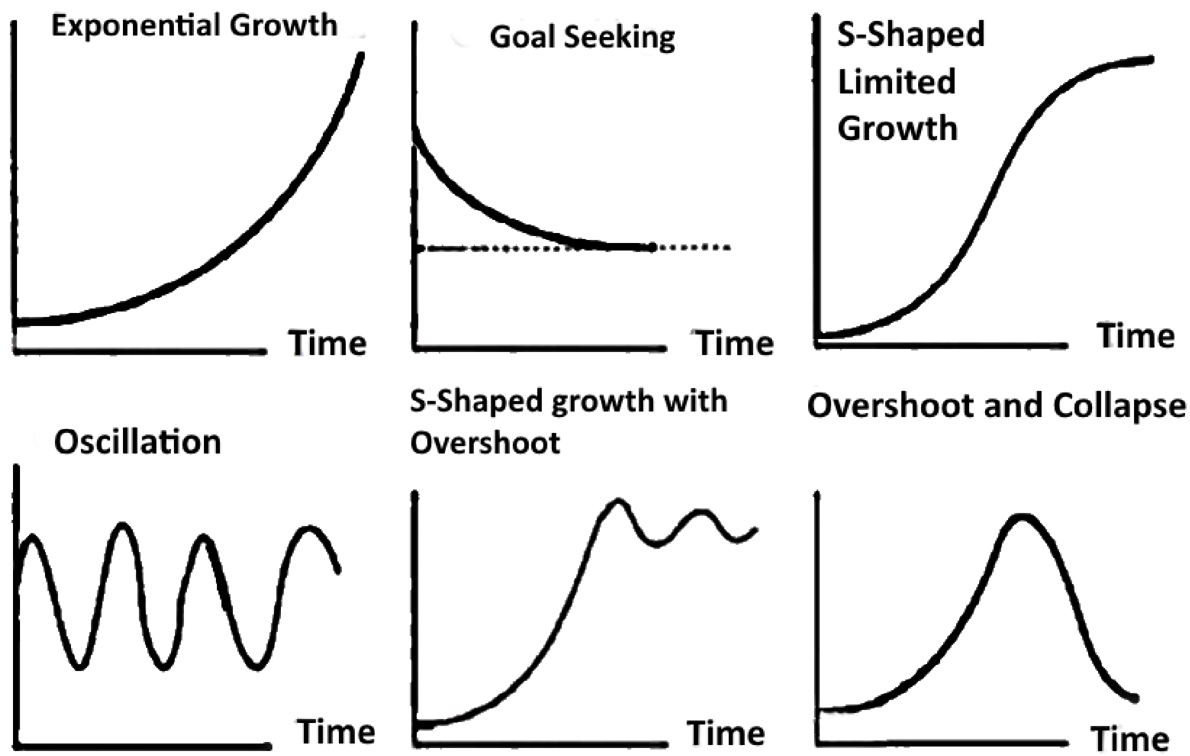


Figure 26 Modes of dynamic behaviour (Wolstenholme, 2003)

3.10 Side Effects of Corrective Measures

Corrective measures are typical actions in any industry and mining is no exception. For example, as demonstrated in Figure 27, the effects of inferior work such as drill quality and measures taken to correct the "side effects" is possible to be quantified provided the rate at which the "rework" is generated is known. The illustration or conceptual model presented by Sterman (2000) also highlights the consequences of inferior work in the form of "out-of-sequence work", "worksite congestion", "coordination problems" and "moral problems" due to schedule acceleration and "fatigue burnout" due to overtime.

Similar to the model in Figure 27, other models by Sterman are also useful as seen in Figure 28, Figure 29 and Figure 30. Sterman also provides the formulae applicable to these models in his book which can be adapted to define these side effects for this thesis.

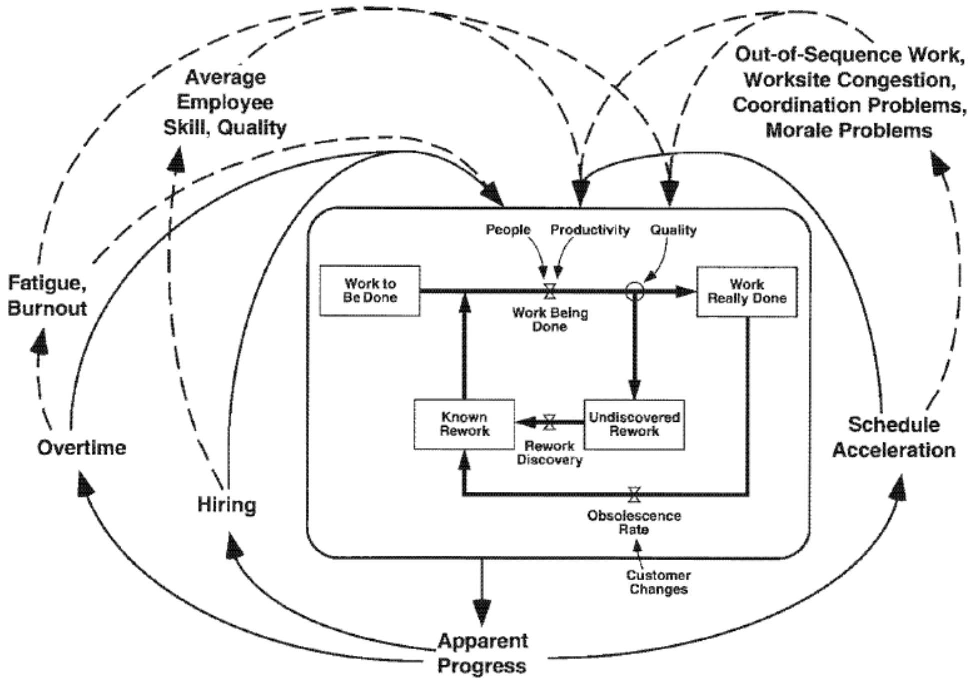


Figure 27 Side effects of corrective measures lead to vicious cycles (Source: Originally published by Pugh-Roberts Associates, Cambridge, MA, cited in Sterman 2000)

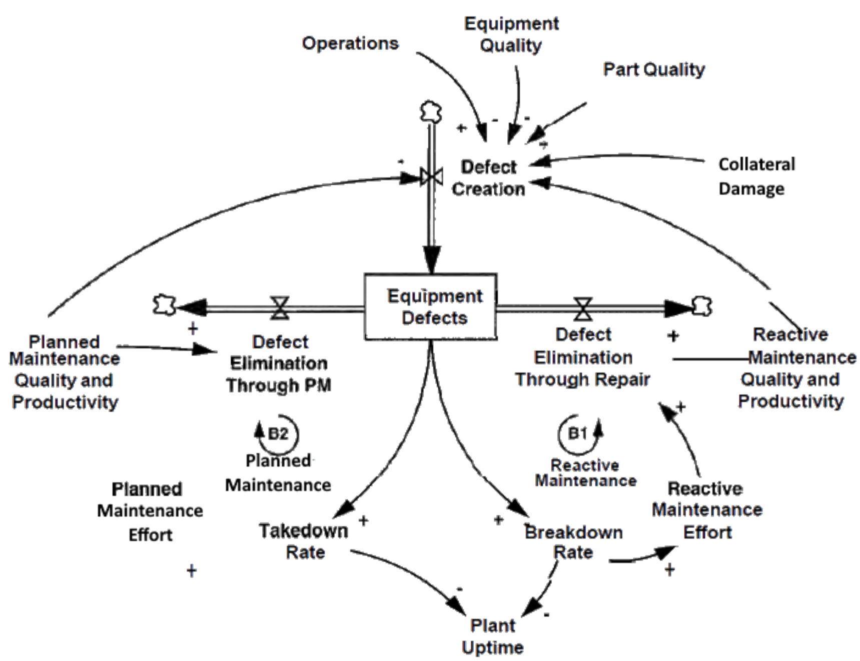


Figure 28 Planned versus reactive maintenance model. (Sterman, 2000)

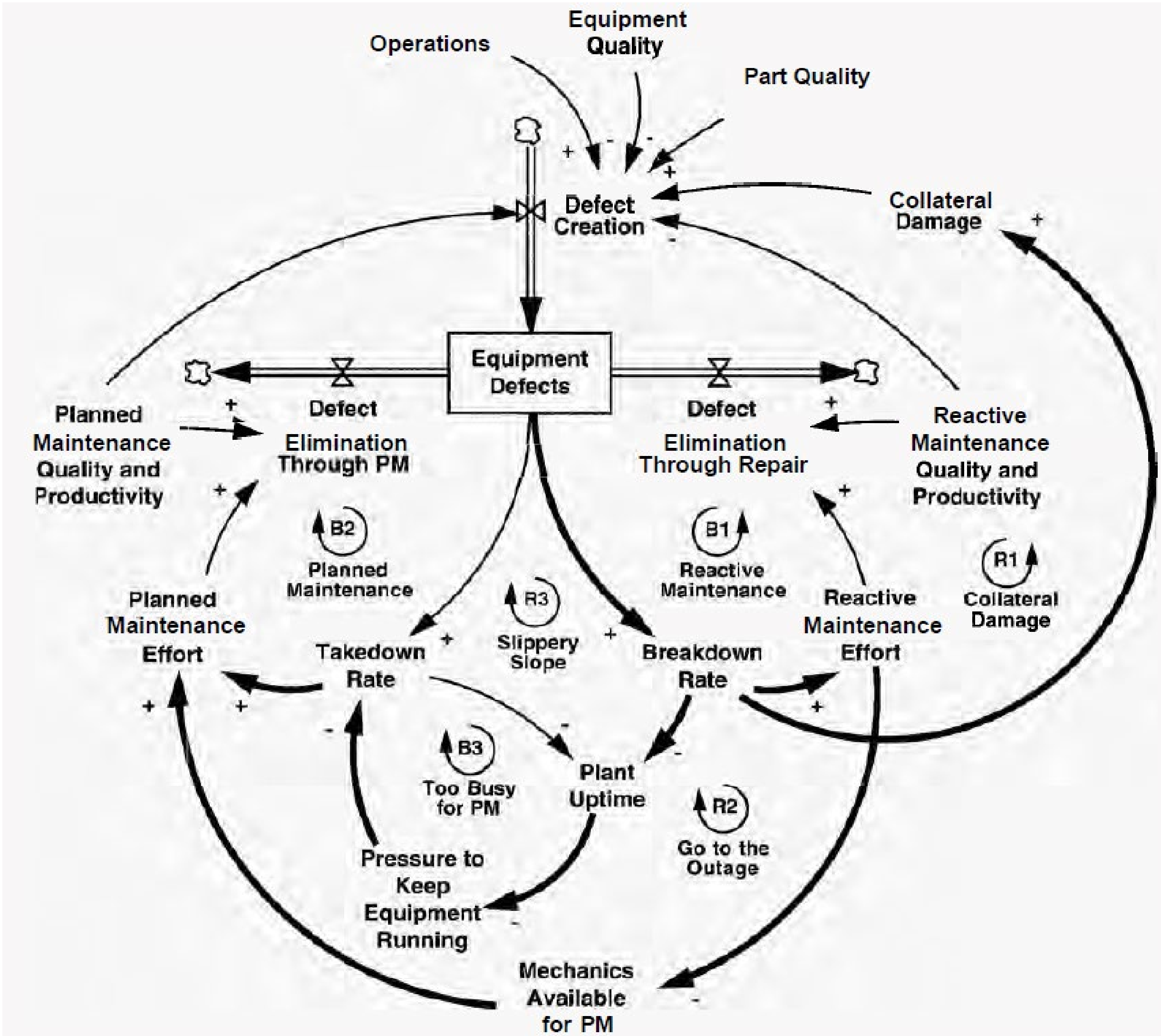


Figure 29 Positive feedbacks undercutting planned maintenance (Sterman, 2000 pg. 94)

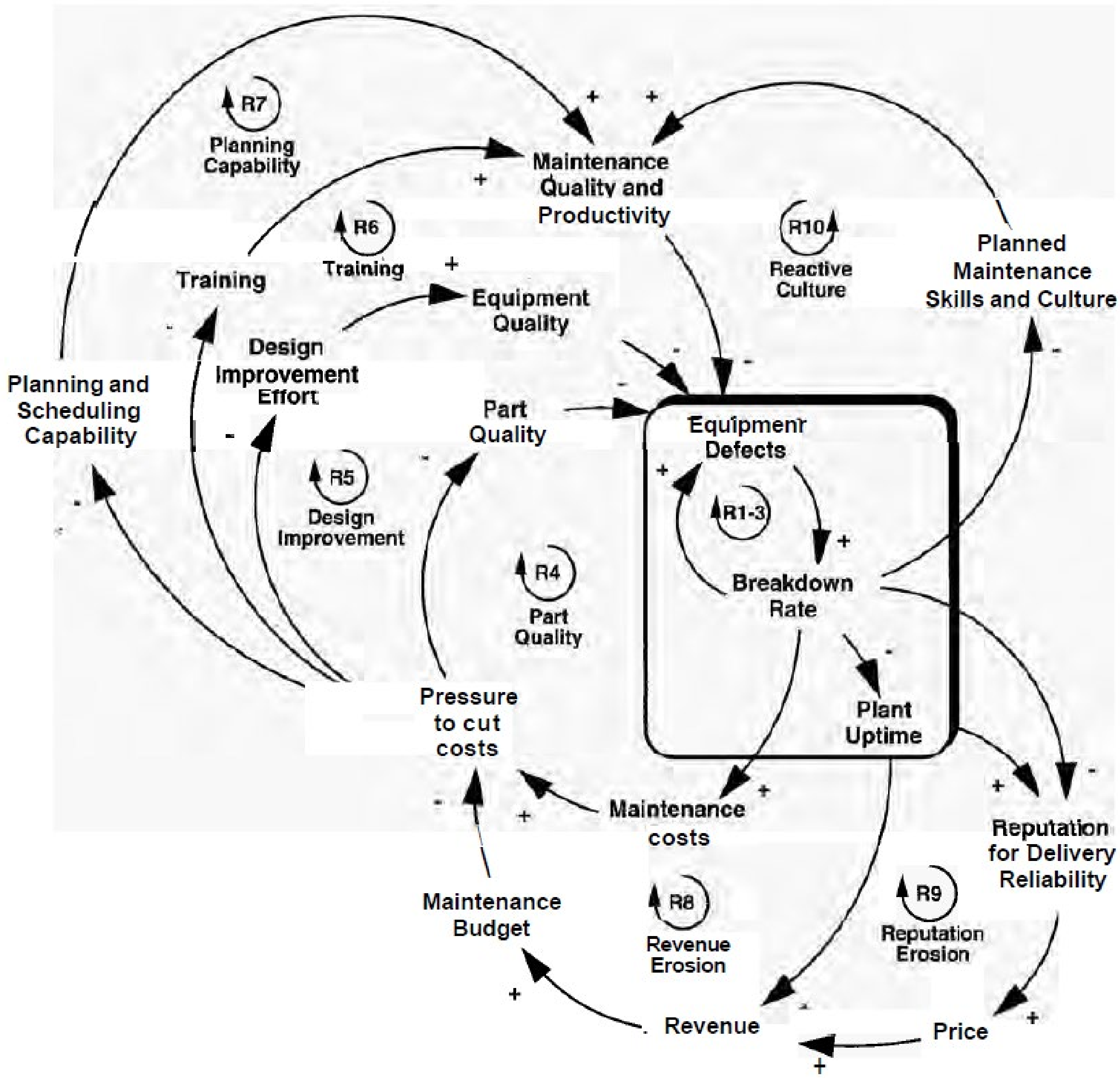


Figure 30 Additional positive feedbacks leading to a reactive maintenance culture (Sterman 2000)

3.11 Testing of a System Dynamics Model and Model Validation

Barlas (1996) states that although model validation takes place in every stage of the modelling methodology, a significant portion of formal validation activities take place right after the initial model formulation has been completed and before the policy analysis/design step. Barlas also mentions that models that are built primarily for forecasting purposes belong to the "output" type model. In addition, causal-descriptive (white box) models are statements as to how real systems operate. Accurate output behaviour is not sufficient reason for model validity but the validity of the internal structure of the model is. In this thesis the approach will be similar to these types of "output" type of mode by first replicating the real system internal structure. Suggestions will be followed for "output" type real systems. In summary, testing starts as soon as the first equation is written.

Barlas further discusses philosophical aspects of model validity. He argues that a model may produce what the system generates but that does not necessarily mean the arguments behind the concepts are correctly built. The model can be either correct or incorrect. The empirical facts, he states, would be automatically revealing the model's correctness.

In summary the following will be used as a guide for this thesis for model validity:

1) Validity of a system dynamics model primarily means validity of its internal structure
2) The recent relativist/holistic philosophy argues that validation of the internal structure cannot be made entirely objective, formal and quantitative since even the scientific theory confirmation has informal and subjective aspects.

Oosthuizen (2014) stated that "V" model enables planning and coordinating the verification and validation of the system during its phases originally as published by Forsberg & Mooz (1994).

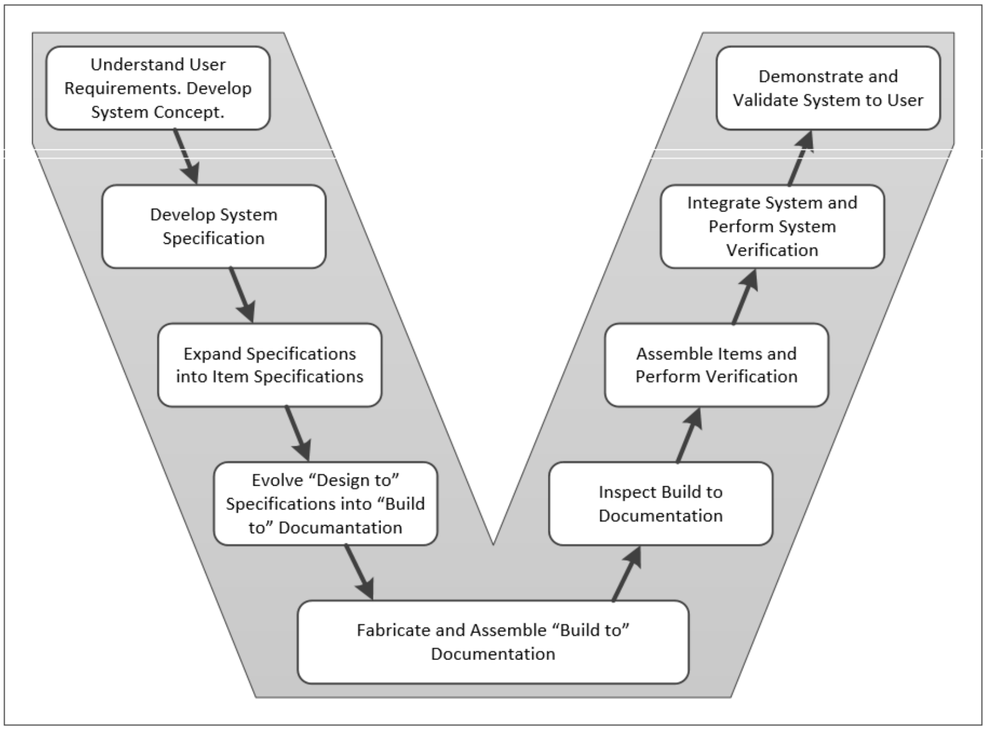


Figure 31 "V" model (Forsberg & Mooz 1994)

A more recent V model has been described by Mittal et al (2018) as seen in

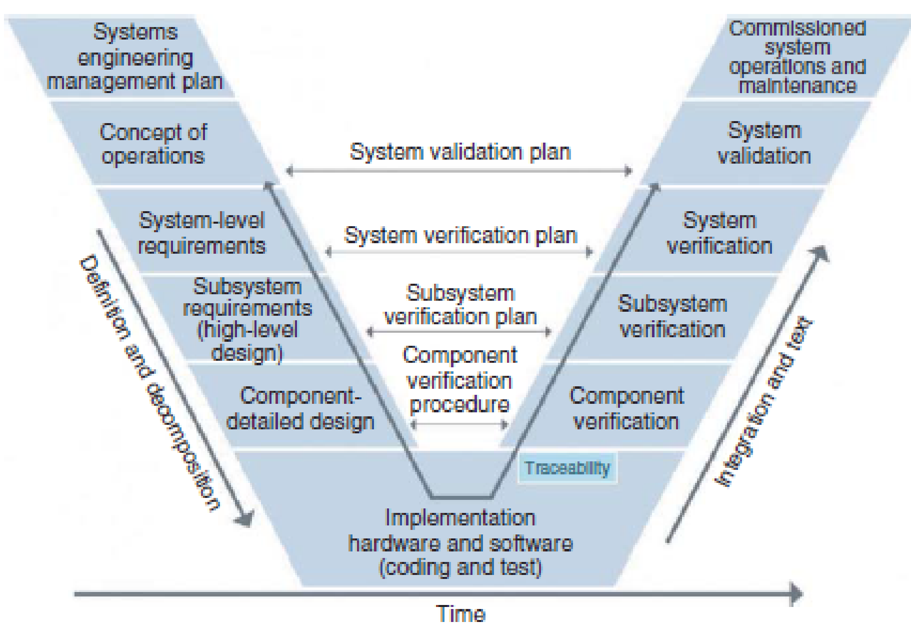


Figure 32 The "Vee" model of systems engineering (Mittal, 2018)

Formal aspects of model validation are summarized in the schematic in Figure 33.

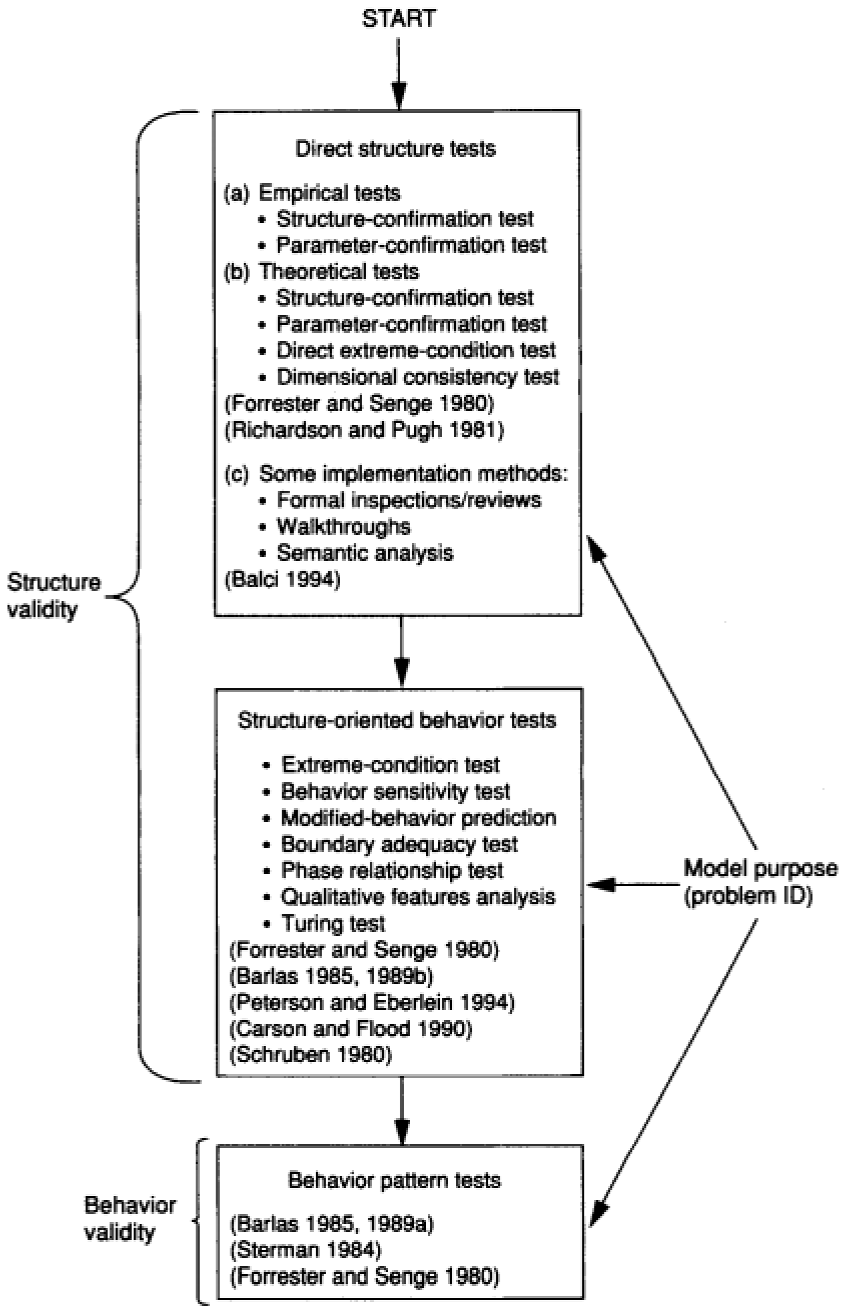


Figure 33 Overall nature and selected tests of formal model validation (Barlas 1996)

It should be further mentioned that Vensim simulation software being used for this research has a reality check feature. Such as, if input A is forced on the system, then behaviour B should result is the argument built for the reality check, then the software can be used to check that the anticipated behaviour is observed.

Behaviour patterns need to be in a logical order as discussed by Barlas (1996). He recommends that the best approach is to compare graphical/visual measures of the most typical behaviour-pattern characteristics, such as amplitude, peak, time between peaks, minimum value, slope, number of inflection points, time to settle, etc.

The ultimate objective of model validation therefore is to develop confidence in predictions (Bala et al, 2017).

The tests for building confidence in system dynamics are typically classified in to three groups by Bala (2017), they are

- Tests for structure
- Test for behaviour
- Tests for policy implications

Also, note that not all the tests are required for validation. The most important tests are structure and behaviour related (Bala, 2017). Bala listed some important tests that need to be performed.

Structure tests are:

1. Structure verification test
2. Parameter verification test
3. Extreme condition test
4. Boundary adequacy test
5. Dimensional consistency test

Checking the formulae forms the basis of the **structure testing** and is said to be relatively easy compared to the other tests.

Parameter verification test is the test that compares real systems both conceptually and numerically. Sometimes statistics sometimes judgement is used. Appropriate initial values such as constants or table functions belong to the parameters.

Extreme condition test is the test that checks the robustness of the system with smallest or largest inputs into parameters. If the model behaves as it should outside historically inputs or parameters, then it is considered robust. For example, zero production or zero initial cost of an item can be considered as extremes.

Boundary Adequacy test considers structural relationships to satisfy a model's purpose. Feedback loops are part of it and all relationships need to be tested and the system should check that additional feedbacks have any significant impact.

The last test in structure test class is the **dimensional consistency test**. Dimensional consistency is applicable to the equation's units. If the formulae result in same unit for both sides of the equation, it is considered dimensionally consistent. Apparently, this is the test most models fail at.

3.12 Chapter Summary

In this chapter, system dynamics as a tool for mining simulation is explored and is explained in terms of the type of language it is, types of tools available , background mathematics used in system dynamics and the types of models that are possible in this type of modelling environment. System dynamics is a well-established problem solving technique via modelling. It has well established testing techniques as well to test for the correctness of the setup simulation model.

Next chapter is exploring the mining value chain system requirements towards building a system dynamics model.

"The coal miner: -I go into darkness to bring light to your world"

Chapter 4

4 MINE VALUE CHAIN ANALYSIS WITH THE VIEW OF DRILL AUTOMATION

4.1 Introduction

The aim of this chapter is to establish the framework for the creation of the simulation model to meet the objectives discussed in chapter one based on existing approaches as described in chapter two and three. The framework requires a good round of mining systems defined that are directly or indirectly impacted with the move to automation or any other changes that may be built in an already established mine.

The mining cycle comprises a combination of many cycles interlinked with more complex ones, often all carried out by one machine or piece of equipment. Drilling is an example to one of these cycles. The actual physical drilling of the blasthole is relatively a straightforward process and this part of the drilling process is often automated at the machine level; the operator sets the boom and then begins the drilling process by pushing a button, at this point the rig takes over and on-board controllers drill the hole and retract the drill steel (Gokhale, 2010).

Automation is defined by Webster's new world dictionary as "In manufacturing, a system or method in which many or all of the process of production, movement and inspection of parts and materials are automatically performed or controlled by self-operating machinery, electronic devices etc.".

In essence this means the removal of a person from a process and controlling that process by means of a computer, machine or robot. Any task that involves routine, repetitive work should, in theory, be automatable. Automation becomes more complex when the task involves many variables and evolving environments such as those that are encountered on a production face in the mining context.

The complexity in the drilling process arises because of the drilling cycle as part of the overall cyclic mining process; for a drill rig to be able to drill a blast hole, many sub-processes often referred to as unit processes by their very nature are complex, therefore challenging. Firstly, a drill pattern must be completed, this must be delivered to the drill operator, and the operator must then tram the machine to the production bench or face all the time interacting with ever-changing scenarios such as other equipment and pedestrians.

Once at the production bench over the blasting block the operator must position the rig, set-up and then position the drill to begin the drilling round. See illustration of a mining bench (Figure 34). The drilled hole is then charged with explosives. The subdrill area is below the designed bench level to properly break the confined toe region. The accuracy of the drill depth is therefore critical. Once all the drilling planned for the bench is complete, the drill rig needs to be moved from the bench and navigated to the next block of ground. Due to GPS positioning systems, this process is also part of the automation, and it is a choice that needs to be made by the company, based on own experience the process is not error free. For example, the pattern drilled by the automated navigation system is so perfect that the pattern drilled by the automatic GPS navigation systems does not align to the manually spotted holes on the same bench leaving a large triangular gap between the two types of drilled holes, the two (manual and automated navigation) should not be mixed in the same blasting block.

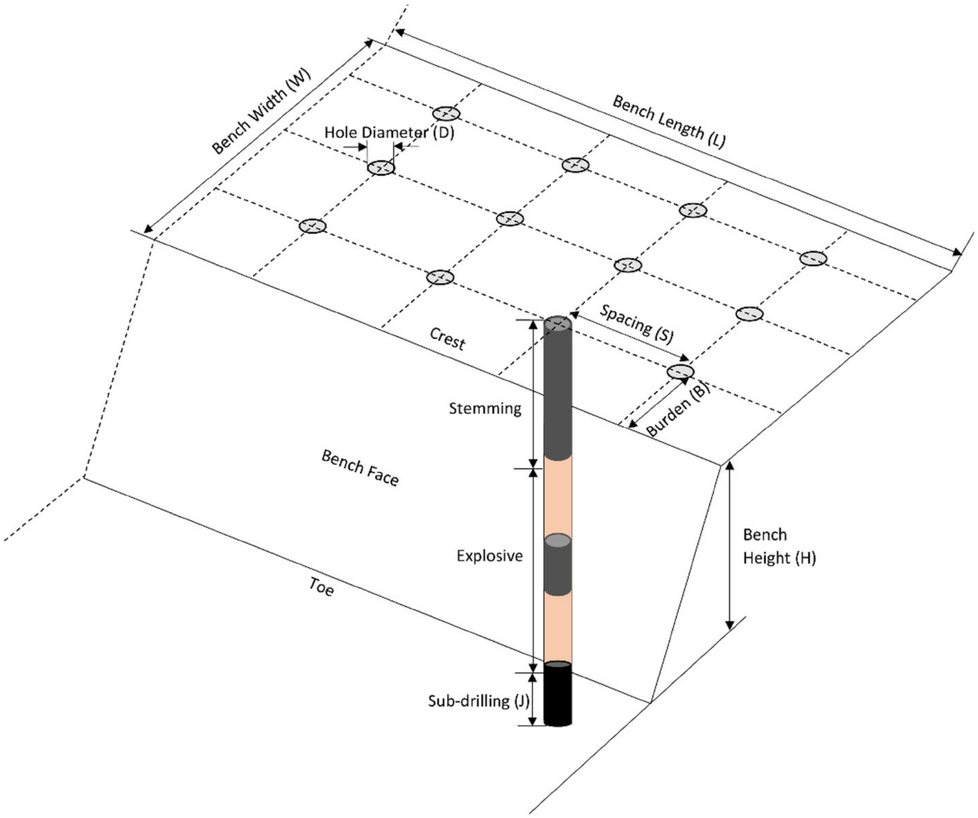


Figure 34 Illustraion of a mining bench (Ozdemir and Kumral, 2018)

In addition to the complexity surrounding the automation of the cycle, many other issues must be overcome before automation can be adopted into the mining cycle. Primarily, automation implies that management is no longer confined to people; to effectively control an automated operation, the processes will have to be effectively managed as well. An automated operation will be controlled by extracting and analysing data to provide meaningful information that can be used to manage and may improve processes.

The management of information is a key aspect in implementing automation into any production cycle. If the operation is not equipped to manage the data produced by the equipment, and react to it accordingly, then any attempts at automation will not be successful. It must be stressed at this point that this management is not limited to managers and supervisors only; equipment operators will have to be highly skilled in managing the information flowing from the machine to make effective, real time decisions. (Somarin, 2014). This level of skill implies that operators will have to be increasingly well educated to effectively manage their equipment. This could raise concerns regarding the sourcing of these individuals; particularly given levels of education found amongst the labour force in the traditional territories in which the company operate.

The management of data, and the subsequent information delivered, is a key component in effectively introducing an autonomous system into the mining cycle so it is anticipated that the introduction of automation is likely to be a growth process rather than a step change. it is necessary to take a view on these stages and apply definitions to the operations that are currently in production, for effectively managing and implementing the automation process (Anonymous, 2012)

it is necessary to categorize what needs to be achieved for applying these definitions to the mining cycle for the purpose of this research. The introduction of a piece of automated, operator-less equipment into the mining environment is the culmination of a process that includes the mechanization of the mining cycle, implementing effective remote control of the equipment, the optimization of that equipment by managing data flows from that equipment, the effective introduction of tele-remote equipment and finally the full automation of individual pieces of equipment. The author has seen some very large databases of automated drill rigs that are generated on a per second basis per drill rig. This translates into millions of data lines with 100’s of attributes. The novelty of the modelling attempt of such data will add value to the researchers and mines who do not necessarily immediately grasp the information behind this live data. The simulation technique that is used for this research is being designed in such a way that the data is not in the forefront, but the characteristic of the data is, that is behaviour. System dynamics is very strong in terms of integrating behaviour (as explained in 3.9) into simulation models rather than hard data. Therefore, this study will reveal the influence of behaviour in the output being studied.

The steps in the automation process can be categorized into the levels of automation as described in Table 5 as most mining machinery manufacturers and automation service providers such as RockMa™, Flanders™, Sandvik™ and Atlas Copco™ would describe.

Table 5 Levels and Definitions of Automation

0	Conventional Mining with handheld drilling and scraper or loader cleaning
1	Mechanized mining – operator ‘ride on board’ operation on CM’s/ drills/loaders, Mechanized loading and haulage using LHD’s/Shuttle cars/trucks/conveyors
2	Mechanized mining – line of sight remote control; operator needs to be next to machine, on board control of machine effected by on-board computer
3a	Automated data recovery and vehicle tracking. Information effectively utilized to increase productivity of the machine by providing information on maintenance requirements, machine health and machine utilization. Information and reports automatically downloaded and produced within minimal operator input
3b	Tele-remote operations: machines operated remotely from control room; automated tramming; load/drill/cutting function controlled by operator. Individual pieces of tele-remote machinery operating in pre-defined independent safety zones – no people or other machines, anyone enters systems stops.

4	All machinery functions automated; Machinery monitored remotely from control room. Individual pieces of autonomous machinery operating in pre-defined independent safety zones – no people or other machines interacting, anyone enters systems stops
5	Fully automated mine all aspects of the cycle fully operator less, the only human interaction is required for maintenance and monitoring of equipment, this can take place on-site or at remote locations

Automation potential varies across sectors and specific work activities. Based on US Bureau of Labour statistics mining seems to have a 51% ability to automate (Chui, 2017). The statistics indicate that there is an automation potential of about 20% for management, 30% on expertise, 30% on interphase, 40 % on unpredictable physical environment, 60% on collecting data, 60% of processing data and 70% on predictable physical data. These figures are evidence that mining can benefit from automation in all levels. Automation of management is the least likely amongst all. However, management can be assisted with a tool that can predict the future conditions based on predicted physical conditions. This is the gap mining is facing currently and by simulation of these processes it is also possible to observe behavioural changes in the productivity that can be hypothetically called automated management via system dynamics.

Mining efficiencies can be improved by overcoming operational problems by automation. Drilling specifically has problems such as alignment, positioning, setting up, drilling with correct pressure, torque and trust etc. Global Navigation Satellite System (GNSS) is a technique that handles alignment and positioning related problems (Vrublova at al., 2016). When drill automation is mentioned, the first that comes to mind is navigation of the drilling rig to the correct position. The next one is during the drilling that controls rotation, torque and trust. Most drills now come with electronic drilling systems that handles correct pressure torque and thrust related problems but not all will be automatically navigating or self-drilling.

Automatic positioning systems can be explained best with a diagram as described by Pirenan (2014).

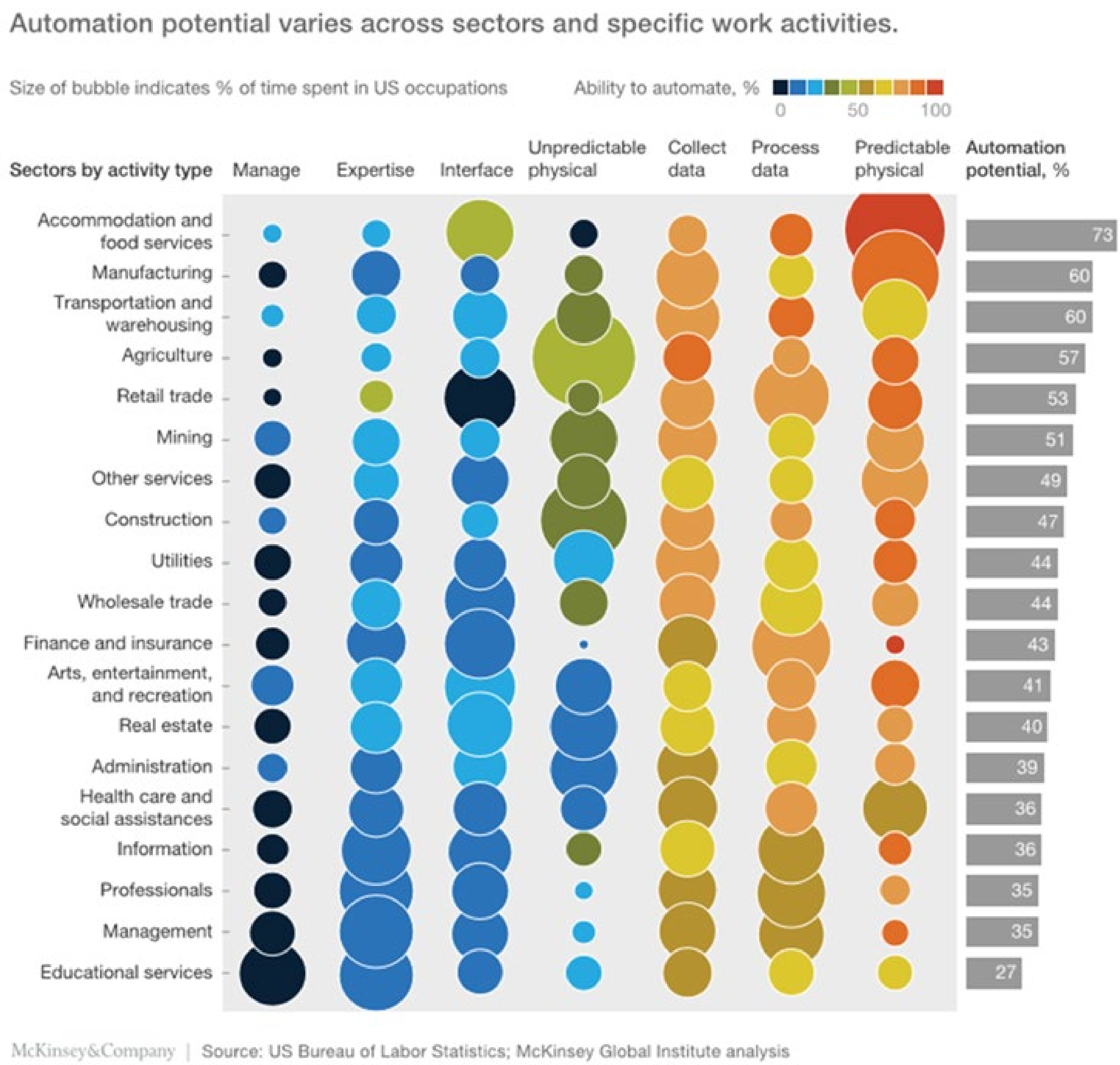


Figure 35 Automation potential across various industries (Chui, 2017)

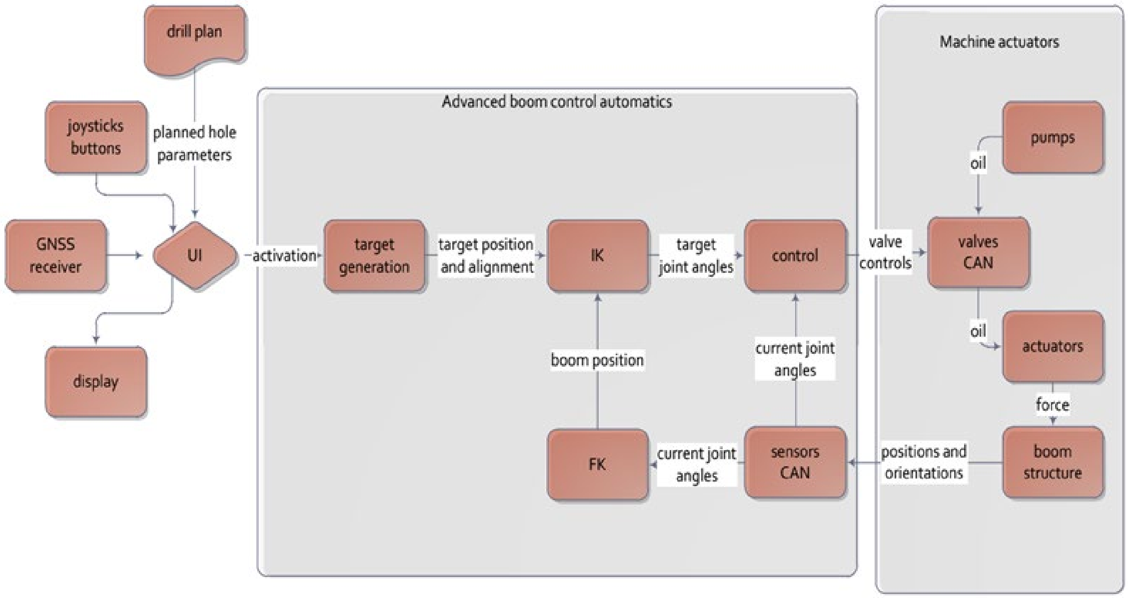


Figure 36 Automatic positioning systems as described by Pirenan (2014)

“Automated and remote machines increase productivity as mining and logistics equipment becomes more reliable, moves faster and covers longer distances, removes shift change requirements and requires fewer operator” according to Rio Tinto web site that has been involved in developing and trialling several automated technologies over the past decade. In many cases these trials are stated as complete and automated equipment is reportedly being permanently utilised across several of RTIO’s Pilbara mines plans to expand autonomous equipment utilisation in the future as quoted by Anna L. Matysek and Brian S. Fisher (2016). They further add to their discussion on benefits of autonomous drills and smart explosive trucks with the following benefits:

- Improved safety outcomes associated with removing drill operators from hazardous areas, as well as a reduction in health risks associated with dust, noise and vibration. Since commencing operation, ADS has had zero injuries.
- Increased use of availability of automated drills over manned drills of approximately 15 per cent (see Figure 4.2).
- More efficient recovery of the orebody by reducing the amount of waste created and improved fragmentation of the blasted rock.
- More consistent and predictable outcomes from precision drilling and blasting, reduced requirement for re-drilling, and a reduction in consumables such as drill bits and explosives.
- A smaller, upskilled, more productive workforce operating multiple drills remotely, and new roles created in system engineering, communications and data analysis.

In support of future automation related areas, a mind map has been constructed in this research on all types of automation related areas in a mining environment. This can be seen in Figure 37.

The following statement by Anna L. Matysek and Brian S. Fisher (2016) makes good sense and also supports the reason for the model development and prediction in this thesis: “Fully automating mining and logistics processes remains a difficult task. The requirements to develop and operate these technologies are correspondingly complex and *rely on high-level interdisciplinary skills*. Additionally, automation technologies are difficult to retrofit to existing equipment, and significant practical problems remain in making all the pieces of equipment and software fit together and work with each other.”

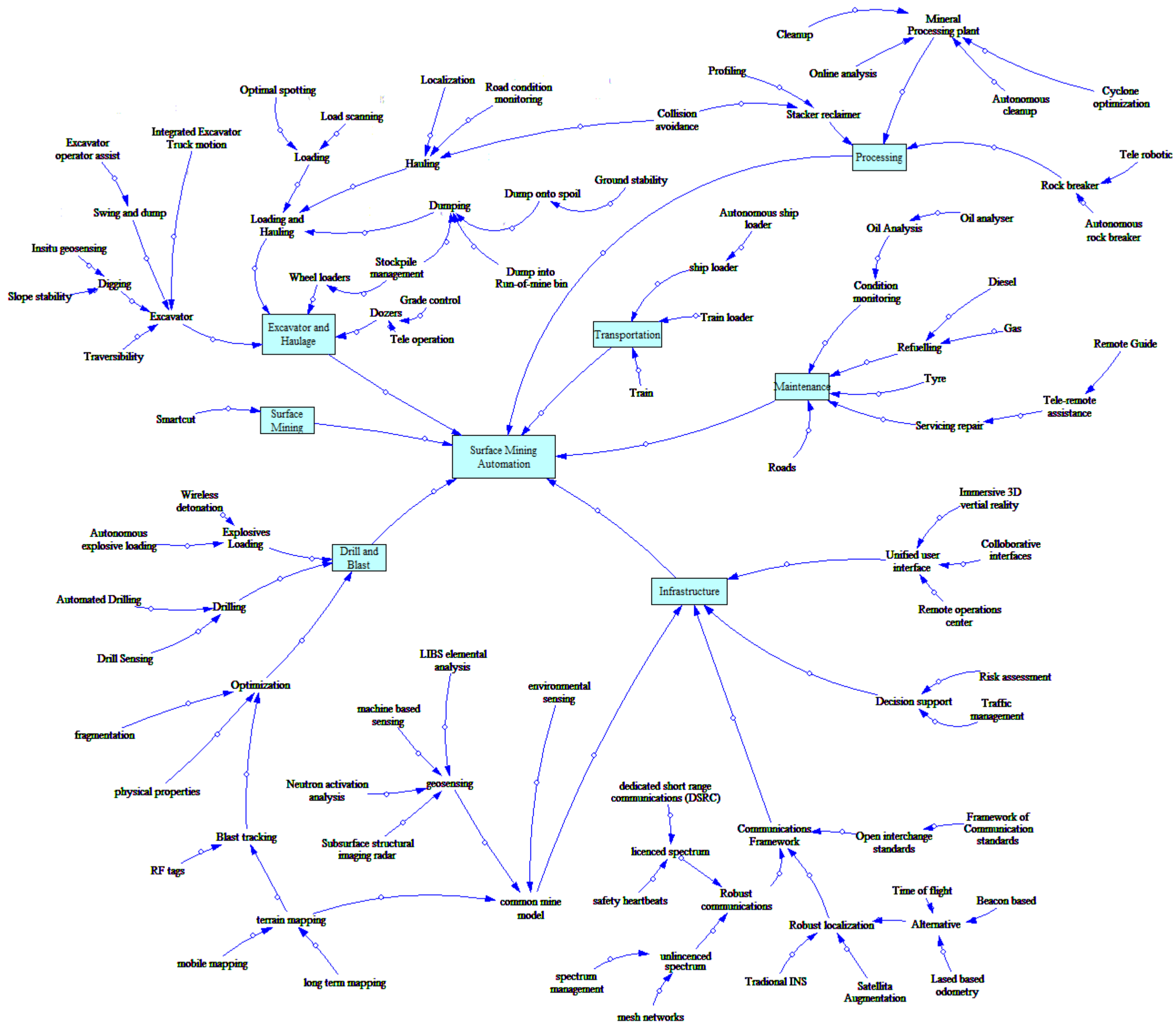


Figure 37 Mind map showing all automatable areas in a mining environment

Flanders Electric has an automation system named ARDVARC^{tm} which is short for Advanced Rotary Drill Vector Automated Radio Control. ARDVARC drill automation that offers four types of drill automation defines the levels of automation. In level 1 drill automation converts manually operated machine into a *one touch* drilling machine. The machine operator needs only to put the machine over the desired target and press a button to start the drilling process. The drill automation system will then level the machine, collar the hole, drill to the desired depth then retract the bit and reset the jacks in preparation of the next propel cycle. In level 2 automation, the data is collected in a Microsoft SQL database and is stored machine ensuring that no data is lost. Level 3 Automation: Level 3 drill automation control incorporates the high precision GPS system to locate holes using target drill hole locations that are uploaded to the drill rig. Level 4 Automation: Provides the drill automation by loading the map and then telling the machine what holes to drill and in what order to drill the target holes, this level is named as *one touch drill mode* (https://www.flandersinc.com).

Automation systems available for drilling are valuable tools to provide mining engineers with the real-time measurements during the drilling by capturing information such as drill hole location as intended on plan and drill hole location achieved. The precision of the measurements is obtained from sensors and satellite information. It is the author's experience that the information or feedback from this large database created by the automated system does not necessarily tell a manager the information about how effective it is or what it does to the rest of the mining sub processes. This is another motivation in this thesis towards for the system modelling of what is the real consequence, and how much is it in monetary terms.

Research carried out by Oparin et al. (2007) highlights the fact that mining today requires more futuristic technologies to deal with the emerging challenges. Therefore, their research aims to look at mine of the future technologies. They further state that future mining development requires a jump in production per worker by introducing advanced mining methods with automation and robotization of the basic and secondary processes. The research highlights those industrial processes "markedly improve mining safety", raises mining "equipment readiness and application indices" and "productivity, increases extraction output per block", panel or district in a mine. But these statements are more of a qualitative nature and no direct quantification has been suggested or reported in this resource. Once again, the motivation for this research as evidenced by these researchers who always talk qualitatively rather than in quantified research for the measurement of real impact.

Nebot (2007) mentions some problems facing mining automation and adds that for automation to be successful a structured environment, well defined automated task requirements and site's willingness to adopt the modern technology should be addressed. In addition, the technology adopted should be simple and robust with no interaction with manned machines. Fully automated mines are rare; therefore, it is expected that systems should accommodate a mixed environment, i.e., manned loading, unmanned drilling. Any benefits that can be predicted for automation may be biased due to interaction with manned machinery and other environmental factors that hinders the success. Nebot (2007) also states that equipment interactions need to be eliminated or very proactively managed to the point where limitations outweigh the benefits of automation. Another issue mentioned is autonomy integrity. Nebot adds that the design of autonomous system is to have enough *sensory redundancy* to detect possible faults.

Research carried out by a Mining Industry Skills Centre Inc in Australia article (Anon, 2010) mentions some goals towards successful automation. The first one mentions transparent communication where strategy sharing between all stakeholders is a priority since careful planning of workforce and skills in the next 5 to 10 years is critical to understand by all parties. The second goal mentioned is investing in apprentices. The third goal discusses establishment of a change management framework. This is especially important to ensure that companies employ strong leadership in human resources, increase awareness regarding the reasons for automation and the impact it will have on them between senior management, overcoming resistance to change and preparing internally for major changes that the automation will bring.

4.2 Review of automation in the mine value chain

The objective of this section is to explore ways of quantification of the mine value chain and investigate possible modelling approaches towards quantification of the impact of automated drill rigs on the quality of drilling. Chapter 1 mentioned that justification of benefit tends to be qualitative rather than quantitative. The research will focus on modelling the qualitative as well as quantitative for a complex surface mining system. Therefore, the focus of the literature review is towards identifying the methods for assessing the mining value chain and determine the cause-and-effect relationships and interactions between the unit processes. This required reviewing systems thinking approaches towards quantification methods, mining efficiencies and behaviours of various processes and mining process causalities typically observed at a surface mine. The literature review further goes into exploratory nature looking at key relationships and variables in the form of formulae that define the relationships, constants and inputs for inclusion in the dynamic model to be developed in this thesis.

4.2.1 Value Thinking

Mining is complex not because of the number of processes involved but the combinations of interactions between processes are complex. Sterman calls this as combinatorial complexity (2000). In addition, optimal solutions to individual problems cannot be added to find an optimal solution as stated by Rosenhad (1989).

Stevenson & Wolstenholme (1999) states that "semi-systemic" developments in management thinking broadened needs and opportunities for system dynamics, i.e., value-based management, human capital management and balanced scorecards. Companies favoured or adopted VBM (Value based management) but most importantly "the need for analytical tools" that can accommodate operational realities by describing the development and deployment of the major operational resources of the business, and interactions between them." (Stevenson & Wolstenholme, 1999).

The next emerging trend is knowledge management. Knowledge management is concerned with collecting sharing or developing both implied and clear knowledge. It is concerned with learning from the past and shaping the future based on the knowledge. (Stevenson & Wolstenholme, 1999). Knowledge management is often too complex for big organizations, such as corporate mining companies where several mines exist within the same management. The importance of having an "easy to implement" type of solution to generate knowledge based on the system behaviour is realized from the above argument and this is a good motivation for this research. Further emphasis on the importance of the value management in knowledge-intensive industries is made by various authors including Urzua (2012) and Olvera (2021)

Further emphasis has been made by Stevenson and Wolstenholme (1999) on uncertainty about how to measure and manage the quantity and quality of human resources relative to business strategies over time. Therefore, added that the system dynamics is a powerful tool to relate human assets and attributes to other resource-based perspectives of strategy and value.

In summary, value thinking means developing a broad understanding of how future cash flows are impacted by operational and management strategies. Therefore, value thinking means thinking systemically and dynamically. (Stevenson & Wolstenholme, 1999). This statement perhaps underestimates the term *Value* in a mining environment. Although, the bottom line is about cost per ton but there are other gains that are not easily quantified.

Neiger et. al. (2009) summarized the SD concept as a good approach for quantification of complex systems by using simulation techniques available within the SD frameworks, the impact of changes in the lower-level objectives on the higher-level objective can be evaluated. He further points out that this approach to business modelling allows for representation of time delays and non-linearity inherent in the dynamic nature of a business. The rigorous mathematical foundation for system dynamics provides the ability to develop a link from a business model to a simulation model for quantitative evaluation of *what if* scenarios.

4.2.2 Value Chain Dynamics

System dynamics (SD) provides the means to test alternative futures in terms of potential value, through a process of knowledge capture, knowledge integration and application. Value chain dynamics is a combination of system dynamics and value thinking. (Stevenson & Wolstenholme, 1999). Elements of Value Chain Dynamics listed by Stevenson and Wolstenholme (1999) are:

- Tool set
- Knowledge integration process
- Templates or accumulated knowledge
- Balanced measurement of performance and value outcomes

Competencies Value Chain Dynamics as a project methodology has been described in a schematic reproduced from Stevenson and Wolstenholme (1999) as seen in Figure 38.

4.2.3 Mining Chain Value Analysis

The bottom line in any operation is cost over revenue. This depends on many parameters in the mining context including geological setting, overall operational efficiency (OEE), labour productivity, proper planning among many others (Kennedy, 2000). These will be systematically explained here.

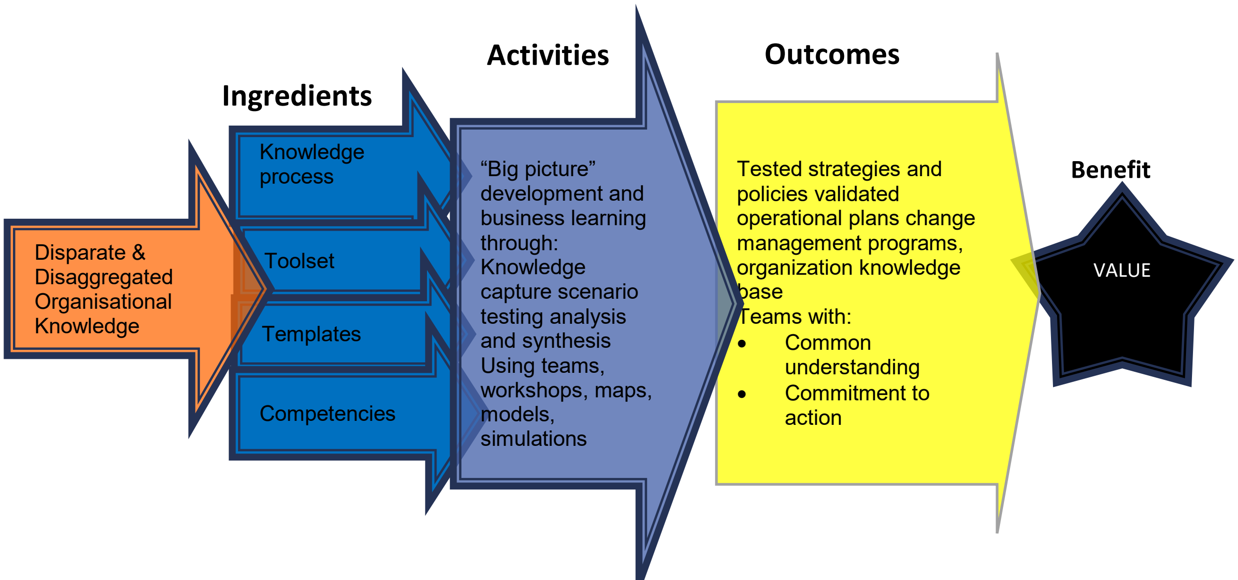


Figure 38 Value Chain Dynamics as a project methodology Stevenson and Wolstenholme (1999)

Availability, reliability, maintainability, and capability are components of the effectiveness (Barringer, 2019) which are helpful for deciding which components are causing the diversion from the expected performance measures.

For helping managers focus on desired business impact measures, a distinction is made between hard and soft data; where hard data are the primary measurements of improvement that is undisputed and desirable. According to Phillips (2002, pg. 142) the criteria for measuring the effectiveness of management rest on hard data items, such as productivity, profitability, cost control and quality control where hard data is listed as:

- Easy to measure and quantify
- Relatively easy to convert to monetary values
- Objectively based
- A common measure of organization performance
- Credible with management

And soft data is:

- Sometimes difficult to measure or quantify directly
- Difficult to convert to monetary values
- Subjectively based, in many cases
- Less credible as a performance measurement
- Usually behaviourally oriented

Examples of hard data are listed by Phillips et al (2002) are listed in Table 6.

Table 6 Examples of hard data (Phillips at all, 2002)

Output	Time	Costs	Quality
Units produced	Cycle time	Budget variances	Amount of scrap
Tons manufactured	Response time to complaints	Unit costs	Amount of waste
Items assembled	Equipment downtime	Costs by account	Rejects
Items sold	Overtime	Variable costs	Error rates
Sales	Average delay time	Fixed costs	Rework
Forms processed	Time to project completion	Overhead costs	Shortages
Loans approved	Processing time	Operating costs	Product defects
Inventory turnover		Delay costs	Deviation from standard

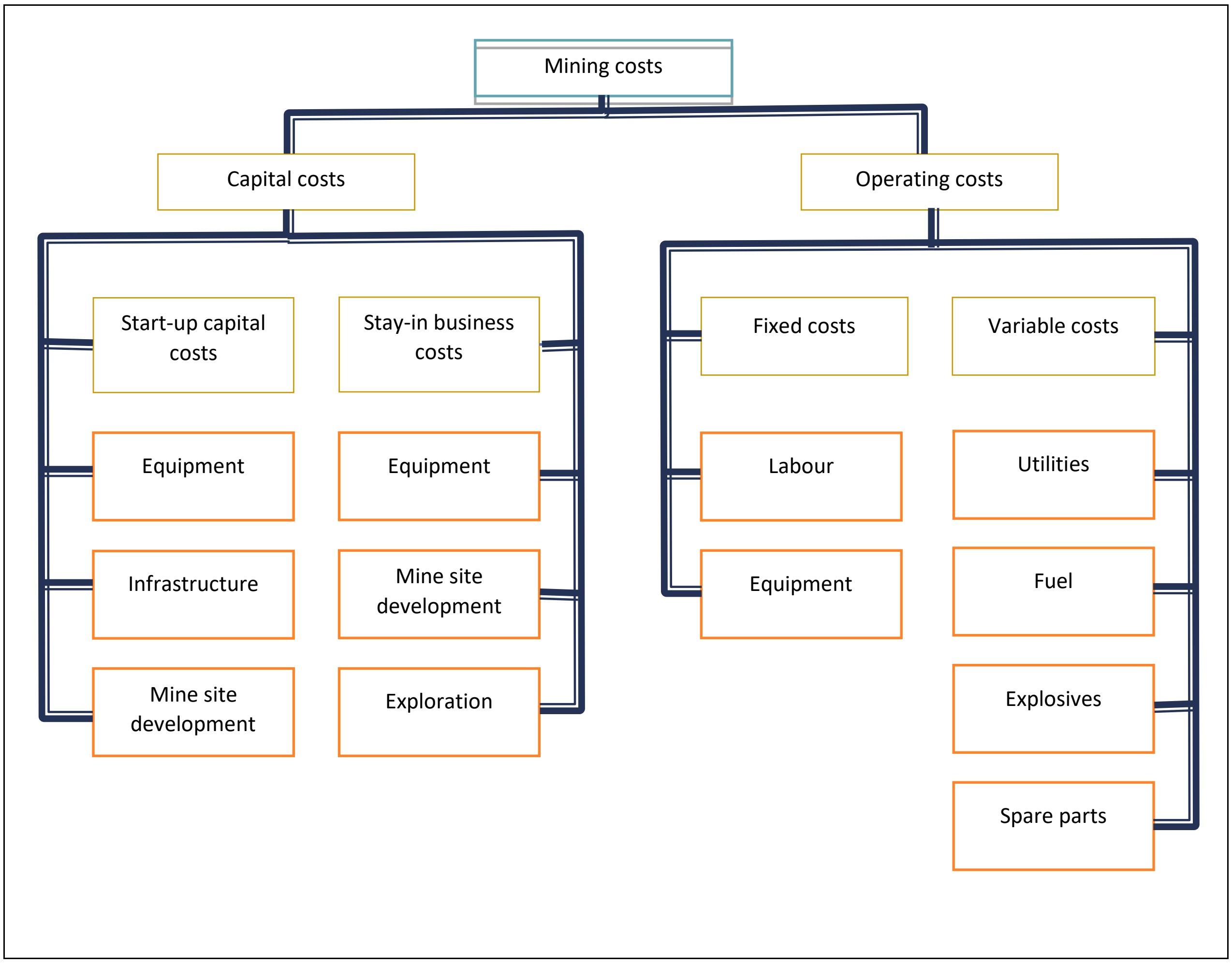


Figure 39 Generic mining cost structure (Mohutsiva and Musingwini, 2015)

Further, a generic mining cost structure is published by Mohitsiwa and Musingwini (2015) presented in Figure 39. They have described cost estimation processes with bottom-up as well as top-down approaches. They have shown using regression analyses is as good as actual cost data obtained from the mine. This statement gives confidence to this author that statistical data can be relied on when establishing the system dynamics model without relying on spreadsheet data, therefore, actual data based on statistical descriptions has been decided to be used in the model presented in this thesis. Mining costs need to be an integral part of the envisaged value model to be constructed. Typical mining costs by activity can be seen in Figure 40.

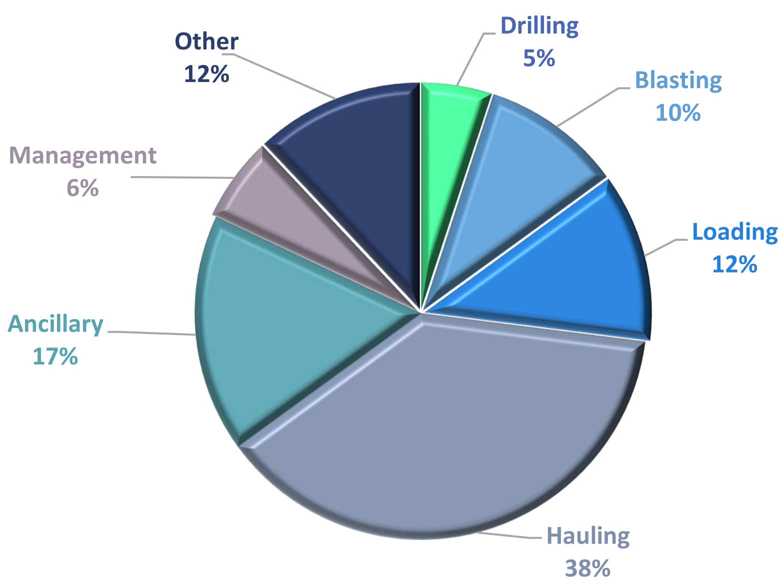


Figure 40 Mining processes costs by activity (Gregory, 2002).

It should be noted that depth factor is a major contributor to unit costs and as the mine deepens the hauling costs will increase significantly, as much as up to 75% according to Hardy (2007). This information is significant but difficult to incorporate in a valid relationship for systems modelling due to 3D and spatial nature of the problem. The hauling costs need to be based on a hauling distances function per mine. The relationship therefore is empirical and changes from mine to mine. Hardy further states that mining engineering adopts a basic methodology of considering relevant individual attributes of systems in isolation. Any comprehensive understanding of a system depends on degree of uncertainty even though degree of uncertainty is a normal consequence, variability can be described with stochastic methods.

Further insights can be gained by analysing the behaviour of the individual processes due to the level and quality of rock fragmentation. Since performance of the mining processes are directly impacted with rock breaking characteristics. Conclusions drawn from individual attributes should be done with caution, especially when future value is being predicted in a combined and complex system.

4.2.4 Measuring Performance

The efficiency of the mining equipment is usually measured using availability and utilisation. The availability and the utilisation of drill rigs have substantial influence on the production output (Elevli and Elevli, 2010). Design and operating variables that are linked to components of the drilling system are controllables and they are the thrust force, rod diameter, bit types and inserts used on the drill bits, flushing and circulating fluids. They are not going to be part of the discussion in this research even though they have considerable influence on the drilling performance. For this research they do not serve a purpose as they are deemed to be not as variable as the other parameters, essentially, they are assumed to be constants. Availability, utilisation and production parameters are important measures for measuring the performance. These measures as well as effectiveness will be discussed further below as they play an essential role in the SD modelling efforts. Drilling system performance is being discussed by Kansake (2015) and some of the well-known formulae are listed in the article, which are listed here further to be used in modelling efforts.

$$\boldsymbol{Availability} = [\boldsymbol{Scheduled\ Shift\ Hours} - (\boldsymbol{Maintenance\ Hours} + \boldsymbol{Breakdown\ hours})] \div \boldsymbol{Scheduled\ Shift\ Hours} \quad \textbf{Equation 4}$$

$$\boldsymbol{Utilisation} = [\boldsymbol{Scheduled\ Shift\ Hours} - (\boldsymbol{Maintenance\ Hours} + \boldsymbol{Breakdown\ Hours} + \boldsymbol{Idle\ Hours})] \div \boldsymbol{Scheduled\ Shift\ hours} \quad \textbf{Equation 5}$$

Typical operating times for drill rigs are summarised as in Figure 41

Effectiveness is a measurement of value where lowest long-term cost of ownership using life cycle costs for the value received (Barrington, 2019), therefore

$$\boldsymbol{System\ effectiveness} = \boldsymbol{Effectiveness/LCC} \quad \textbf{Equation 6}$$

Where LCC stands for life cycle costs.

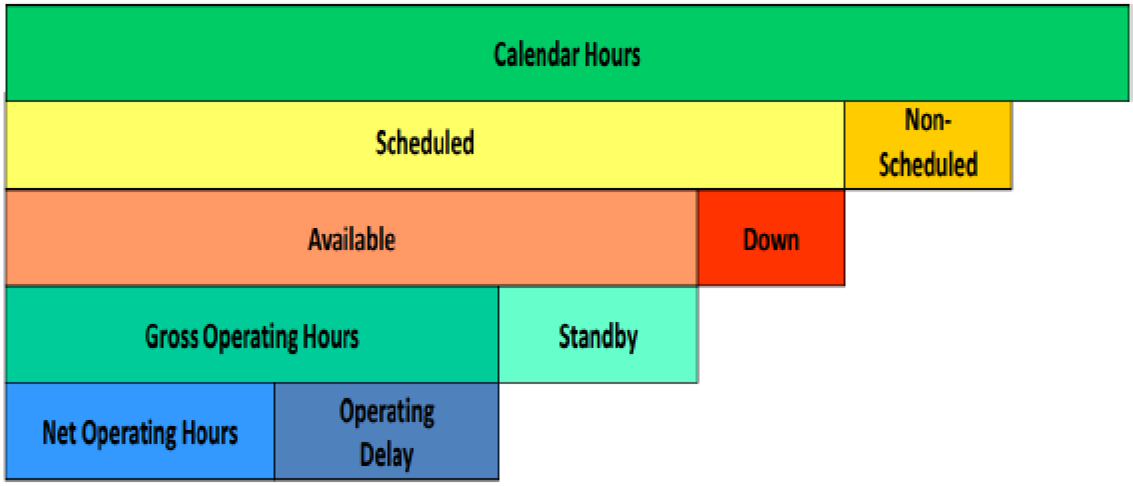


Figure 41 Breakdown of operating times (CIM, 2012)

Barrington states that cost is a measure of resource usage. He states that lower cost is generally better than higher costs and hoping that it includes most principal elements. Effectiveness is a measure from 0 to 1. Another definition is given by Berger (1993) as cited in Barrington:

$$\boldsymbol{Effectiveness = availability * reliability * maintainability * capability} \qquad \textbf{Equation 7}$$

Time is significant in mining due to cyclic processes that are highly dependent on cycle times of the heavy mining equipment. Hence dynamic modelling is appropriate. The effectiveness measures are generally valid for a standard production environment with predictable machinery behaviours. Since the most important variable is the time component, it needs to be analysed based on task priorities and dependencies and they are sequenced and scheduled. Therefore, measuring the benefit of the time is significant in valuation.

There are two kinds of time capture for production equipment in an active mine namely controllable and uncontrollable time. Controllable time includes uptime and downtime. Uptime is made of direct operation time and lost time. Equipment downtime can be due to operational stops, scheduled maintenance and unscheduled maintenance events. The uncontrollable time events and non-scheduled time events are the remaining types of time variables that are used in calculations of system efficiency and effectiveness.

Consider two systems A and B respectively. Availability for systems in series are calculated as:

$$\%\,\boldsymbol{System\ Availability} = \%\,\boldsymbol{Availability\ of\ A\ x}\,\%\,\boldsymbol{Availability\ of\ B} \qquad \textbf{Equation 8}$$

While availability of systems in parallel can be calculated as:

$$\%\,\boldsymbol{System\ Availability} = \mathbf{1} - [(\mathbf{1} - \%\,\boldsymbol{Availability\ of\ A})\boldsymbol{x}\,(\mathbf{1} - \%\,\boldsymbol{Availability\ B})] \qquad \textbf{Equation 9}$$

Similarly, the equipment performance metrics are necessary in the modelling environment and need to be defined. Therefore, Anglo American equipment metric standard is being used for that purpose which is summarized in Table 7.

Table 7 Equipment Performance Metrics (Equipment Metrics Definitions)

Variable	Acronym	Description
Best Demonstrated Rate	BDR	Best Demonstrated Rate is defined as the best demonstrated performance defined by calculating the average of the 5 best monthly production rates.
Controllable Time Loading	TL	Controllable Time Loading: The percentage of total calendar time that is actually used by the operation. TL = T100/T000 x 100
Downtime	D000	Downtime attributable to Maintenance and Operational that renders the equipment inoperable
Unscheduled Maintenance Downtime	D100	Downtime as a result of maintenance work not included in the confirmed weekly maintenance plan
Scheduled Maintenance Downtime	D200	Downtime as a result of maintenance work included in the confirmed weekly maintenance plan
Operational Stops	D300	Necessary downtime attributable to Production that renders the equipment inoperable e.g. replacement of consumables
Engineering Availability	ENA	A performance indicator of how much time the Engineering function requires to maintain equipment (Scheduled and Unscheduled). Percentage of Engineering Available Time over Controllable Time. ENA = (T100-D100-D200)/T100 x 100
Equipment Availability	EQA	a performance indicator of how much time equipment can be productive. Percentage Uptime over Controllable time EQA = T200 / T100 x 100
Equipment Utilization	EQU	Equipment Utilization: a performance indicator of how much time the equipment is used when it can be productive. Percentage Direct Operating Time over Uptime EQU = T300 / T200 x 100
Lost Time	L000	Time during Uptime (T200) when the machine was available for production, but was not utilized
Consequential Lost Time	L100	Impact on equipment, section or modules as a result of upstream or downstream stoppages causing lost time.
Standby	L200	Standby: Time allocated to spare equipment available for production. This is identified as a requirement for the operation, activity or section in their operational strategy and long to short term plans. Standby may not be used if the equipment is not available.
Delays	L300	All production delays that occurred when the equipment was available but not utilized
Non Controllable Time	N000	Total Time equipment is not scheduled to be productive
Non Scheduled Time	N100	Equipment time allocated as not being required in the production plan. (Only applicable to non-full calendar operations)
Uncontrollable Time	N200	Time attributable to external factors beyond the control of the operation that affects the whole operation e.g. outside utilities such as power, water, rail systems, environmental disasters
Mean Time Between Failures	MTBF	Mean Time Between Failures: T200 / Count(D000events) Where Count (D000events) is the number of equipment downtime events contributing to downtime D000

Mean Time to Repair	MTTR	(D000)/ Count(D000events) Where Count (D000events) is the number of events contributing to downtime D000
Overall Equipment Effectiveness	OEE	Overall equipment effectiveness (OEE) is a hierarchy of metrics which focus on how effectively equipment is utilized. The metrics are described below: • Overall Equipment Utilization - OU • Performance: The portion of the OEE Metric represents the production rate at which the operation runs as a percentage of its best demonstrated rate. • Quality: The portion of the OEE Metric represents the Quality achieved at an operation as a percentage of its targeted Quality. OEE = OEQU x (Actual Production Rate)/(Best Production Rate) x (Actual Quality)/(Target Quality)
Overall Equipment Utilisation	OEQU	The ultimate performance indicator of how total calendar time is utilized (Sweating the asset). OEQU = T300/T000 x 100
Secondary/Non-Production Time	P100	Time that equipment is operational but performing non-production activities e.g. training Production equipment used for construction will be classified as non-production time
Primary Production Time	P200	Time equipment is utilized for production
Operational Availability	OPA	A performance indicator of necessary downtime attributable to production that renders the equipment inoperable e.g. replacement of consumables.
Production Buffer	-	A pile or storage location for bulk materials, forming part of the bulk material handling process. In this document a buffer will be defined as a production time buffer between production systems of at least 12 hours
System	-	A set of interacting or interdependent entities forming an integrated whole in between Production buffers. In this document, a system is defined as equipment (or activities) coupled in such a manner that when one is affected the complete system is affected.
Total Time	T000	Total Time: The total possible hours available. Annualized = 24 x 365 = 8760 (non-leap year)
Controllable Time	T100	Controllable Time: Available equipment time attributable to any internal factors under the control of the operation that impacts production. Total Scheduled Time – Unscheduled Time - Uncontrollable Time T100 = T100 – N100 – N200
Uptime	T200	Uptime: Equipment time available for production activities. Available Time – Operational Downtime T200 = T100 – D000
Direct Operating Time	T300	Direct Operating Time: Time during which the equipment is operating. T300 = T200 – L000

Parameters in Table 7 are now constructed in Vensim as seen in Figure 42 where measured values and calculated values are indicated in assorted colours. In this example loading equipment performance is indicated. The formulae can be applied to all main machinery for performance modelling.

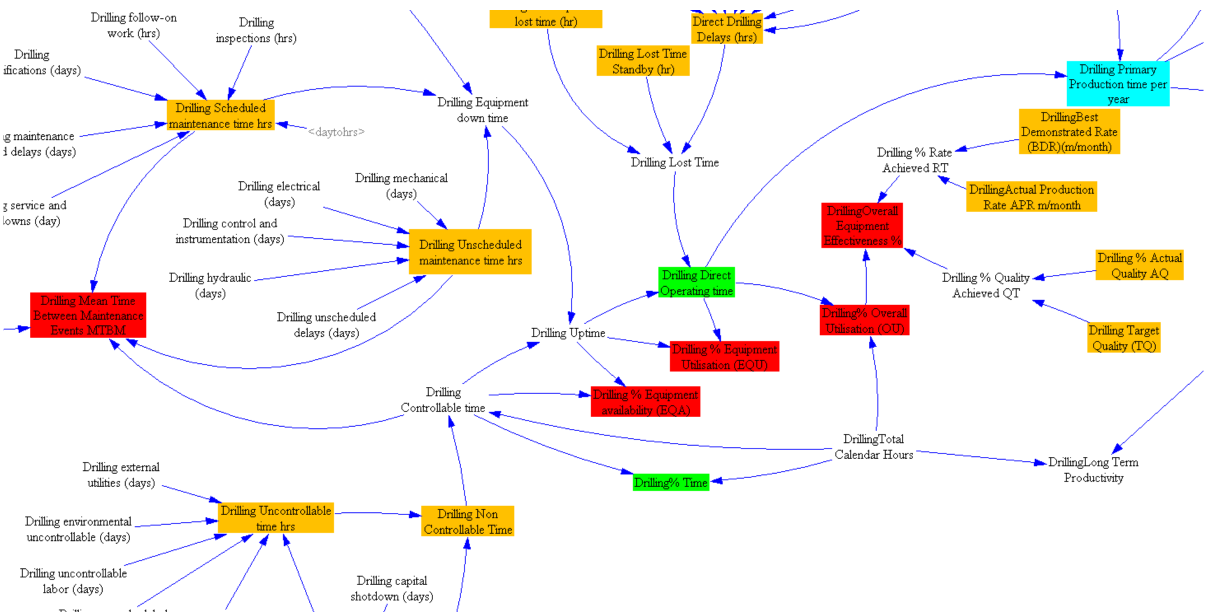


Figure 42 Measurement of performance

Even though the thesis mainly focuses on the drilling process, all mining processes and dependencies need to be built into the model in enough detail to capture downstream effects. The drilling process is typically evaluated using drill hole accuracy (drill hole position and depth), on time drilling, drilling rate, drillability, drill rig parameters and ground engaging tools (GET) consumption rate. Drilling in turn affects the blasting quality and it is one of the most important connectors in the overall system efficiency analysis. The shared factor in this case is the volume of drilled blocks of rock compared to volume of blasted rock. Blasting parameters are powder factor, the charge length, the amount of explosive used per charge, effectiveness of initiation systems used, and the outcome is sufficiently fragmented rock that can be loaded out effectively with loading machines. Typical loading and hauling parameters are digability, muckpile density, loading factor, teeth life, engineering and operational costs, bucket capacities, swell factor, etc. All of this eventually leads to cycle times and effort spent; hence the fuel consumption, which is significant due to loading and hauling are the main fuel heavy processes largely effecting the production costs.

Digging conditions are defined by the geometry of the muckpile matching the loader and the muckpile characteristics; therefore, the effect of fragmentation on loading and hauling performance is discussed in the next section.

Drilling always impacts the next round of drilling process due to incorrect X and Y coordinates as well as Z (depth of drilling), which may lead to bad fragmentation and floor damage in the bench below if not managed well. Similarly, the stemming area of the blasting is affected due to incorrect lengths of drilling, if the blast hole is not drilled at the correct level blasthole charging may overflow, too long means the blasting charge is way below the charging elevation as indicated in red colour in Figure 43.

Too long a stemming means the blasthole is going to create boulder due to ineffective interaction of the detonating columns.

The incorrect positioning of holes in a blast has a similar effect on hole quality as does over-drilled holes from the previous bench. For safety purposes, holes on succeeding benches are normally laid out so they can be positioned between the sockets of the preceding holes. The aim is to avoid drilling into potential misfires (Rorke, 2019).

Rorke (2019) also discussed the consequences of holes drilled too close together. When this happens rock fragments tend to be too fine affecting sales of most commodities. Explosives may also become damaged and detonate improperly or detonate sympathetically, either case leading to poor fragmentation.

The Figure 43 is an analysis of a blasting block in terms of stemming lengths variation due to incorrect elevation of the blastholes which is often linked to previous bench blasting activities. This is an example of the author's previous experience regarding some field work.

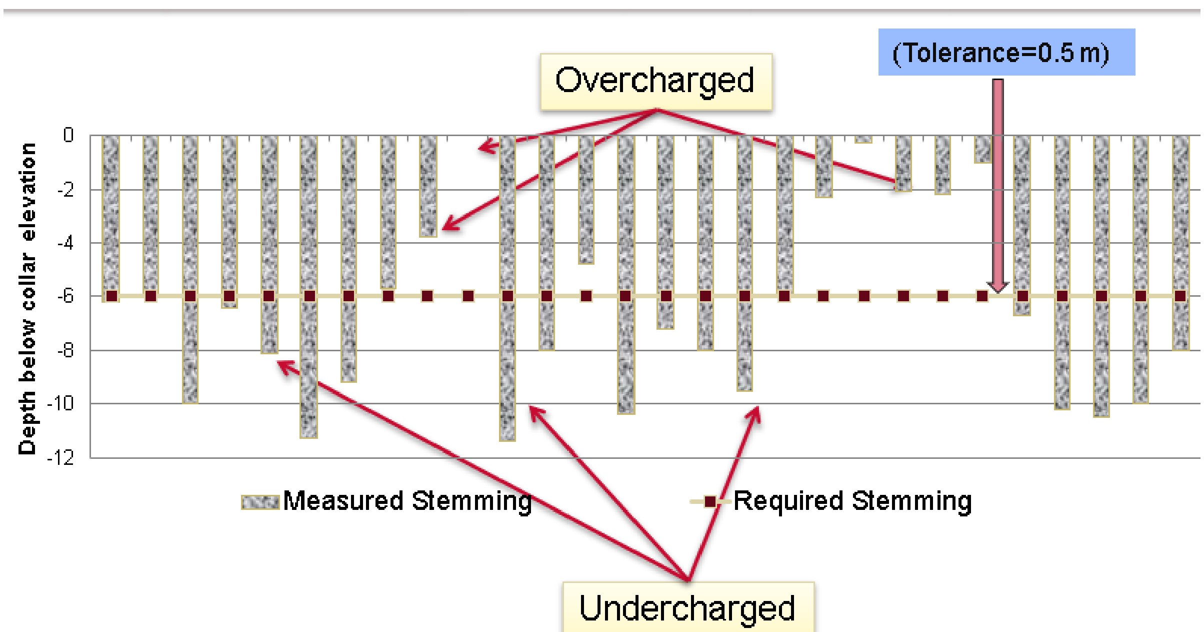


Figure 43 Incorrect stemming lengths defined graphically

Incorrect stemming lengths are a result of a combination of the following:

1) Quality of the bench preparation in terms of levelling efforts
2) Drill depth not managed well due to manual drillers incompetency
3) Blasthole wall too weak leading to collapses of the drill hole
4) Drill chippings falling into the blasthole due to high traffic in the drill block, drill rig walking over the previously drilled holes during redrilling efforts or bad planning of drill sequence

Most of the problems listed above can be avoided if drill sequence as well as drilling itself is automated to avoid operator influence. The GPS driven drill rigs are precise to a few centimetres due to satellite positioning systems (Bester, 2018). In manual drilling mode spotters are used to position the drill rig in the correct position and their abilities often not enough to get it at the correct X-Y position.

Drilling and blasting are the first steps in rock removal process where rock is drilled first to be fragmented; then explosives are loaded into the blastholes and then blasted. Hauled material then ends up at beneficiation cycle where crushing grinding and other metallurgical processes are involved to concentrate or refine the mineral being mined. The following three processes are the main ones that need to be evaluated in a complete mining cycle; best depicted in the value chain schematic by Wentzel in Figure 44 (2012).

- Drill quality
- Blasting quality
- Loading quality
- Hauling quality

4.2.5 Fragmentation

Fragmentation is the joint factor, and it should be optimal for all processes to minimize costs and maximize revenue per ton of rock blasted. Fragmentation is a result of complex interactions between rockmass, blasting geometry, explosive, and timing sequence of blast hole (Beyoglu, 2016). He further adds that target fragmentation is necessary for efficient loading and crushing while emphasizing the pivotal role of fragmentation in an open pit mine. Beyoglu further states that it is difficult to outline a definitive answer to the question of efficiency in open pit mines. The results of this study were found useful to incorporate into the model developed in this research. The questions raised in his research are also relevant and helps the purpose of this research towards quantification of the parametric changes in the mining environment.

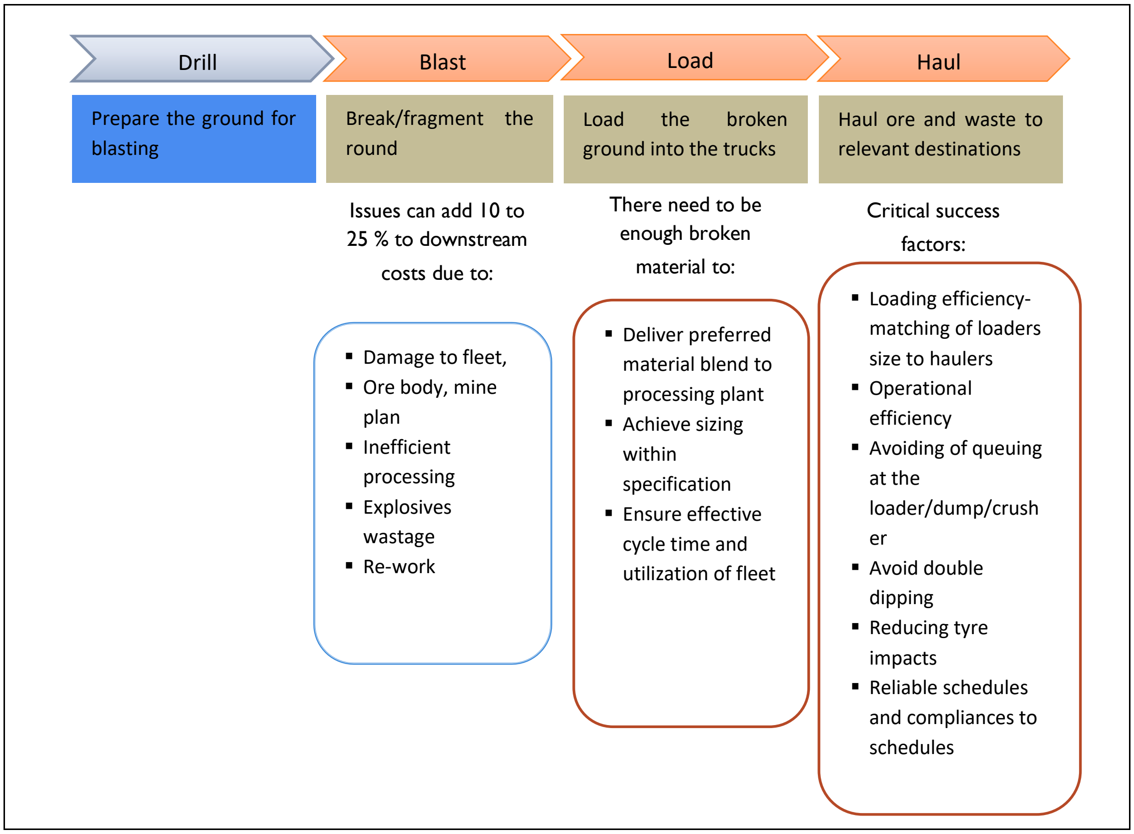


Figure 44 Mine site value chain (Wentzel, 2012) How can rock mass characteristics be implemented in open pit operations to improve fragmentation and efficiency

The main question asked was what the effects of fragmentation on the efficiency of downstream tasks are and what fragmentation is most favourable. In his study, Beyoglu identified a research by Hendriks (1990) and Onederra et al (2004), where dig cycle times are not only related to digging effort or fragmentation, but digging trajectory has a major influence on shovel performance. Digging trajectory itself is dependent on how well an operator can engage the shovel, Muckpile stiffness and digging trajectory effort are interrelated. Muckpile stiffness in turn is a direct outcome of fragmentation quality the root cause of which is drilling accuracy and drill pattern design in general.

A typical fragmentation distribution curve is shown in Figure 45 where the effect of bench height on fragmentation has been constructed by the author of this thesis. The energy demand from the crusher as well as loading and hauling effort increases as the fragmentation size distribution curve moves from left to right. In addition, the steeper the curve means the more uniform the fragmentation is. This is not desired due to difficulties experienced during digging of the blasted muckpile is. The graph has significant regions where actual versus desired fragmentation differs in the coarser fragmentation zone and fines end of the curve.

The objective of the blasting is to shift the curve towards the desired curve (in red) by pulling the coarser end to the left and finer end to the 0 level for iron ore mines. This way the crusher costs will be minimized while reducing the oversize which requires secondary blasting.

1) The inaccuracy of drilling will create variation in the fragmentation expected by design. This has also been discussed further with a model in the modelling section 6.1 in this thesis.

It is known that costs vary from mine to mine and dependent mostly on the current economic outlook including commodity prices, cost of fuel and energy, as well as exchange rates. In South Africa, mines are affected greatly due to daily fluctuations of above.

Figure 46 illustrates cause and effect relationship constructed in VENSIM by this author that shows the value map of a mining process from the resource to the closure adopted from Sontamino's (2014) approach.

Blasting economics has a relationship with the other cost items in such a way that as the blasting cost increases the rest of the mining processes become much cheaper. This has been conceptually defined with the well-known relationship in Figure 47. The relationship basically explains that spending on the quality of drilling and blasting will reduce the costs which will then bring down the overall mining cost. There is no common fragmentation index for each unit mining process (i.e., loading, hauling, crushing). There are numerous studies looking at fragmentation and its effect on the mining processes. P80; 80% level of fragmentation has been used as the level of fragmentation measured for crushing (Asbjörnsson, 2013) as most metallurgical processes are taking P80 as the . The same value is going to be used for loading efficiency based on P80 (Brunton, 2003) level of fragmentation.

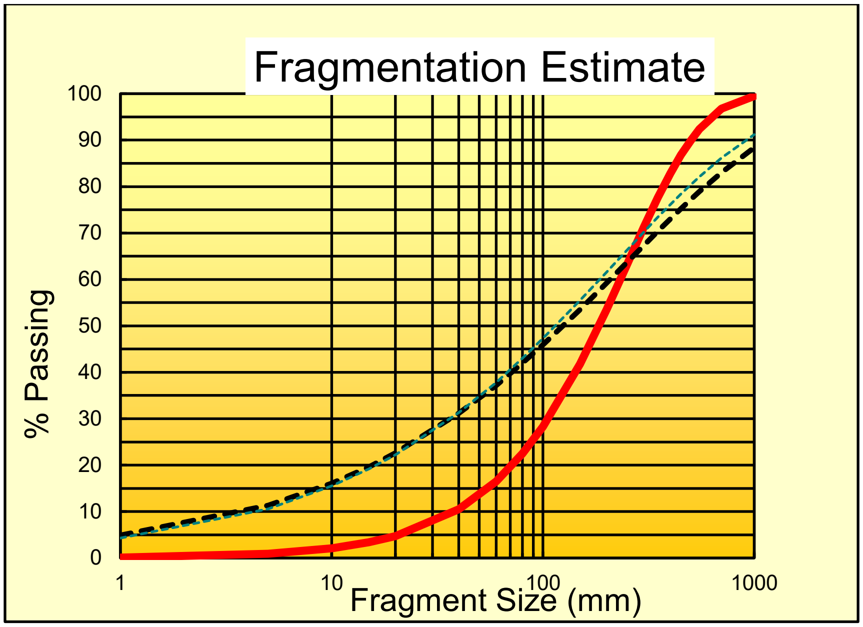


Figure 45 Typical fragmentation distribution curve obtained from a blast compared to a hypothetical desired fragmentation curve indicated in red (Source: Author).

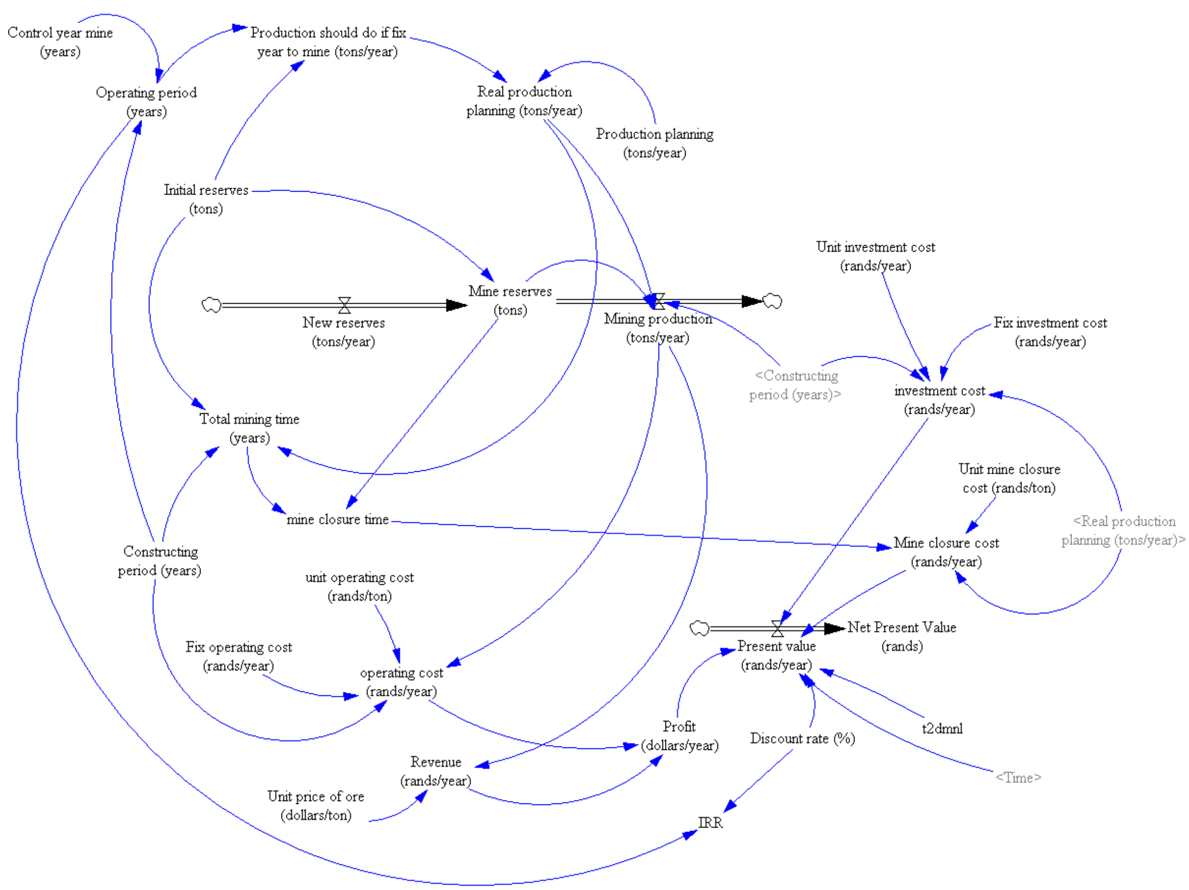


Figure 46 Life of Mine planning model based on iron ore demand

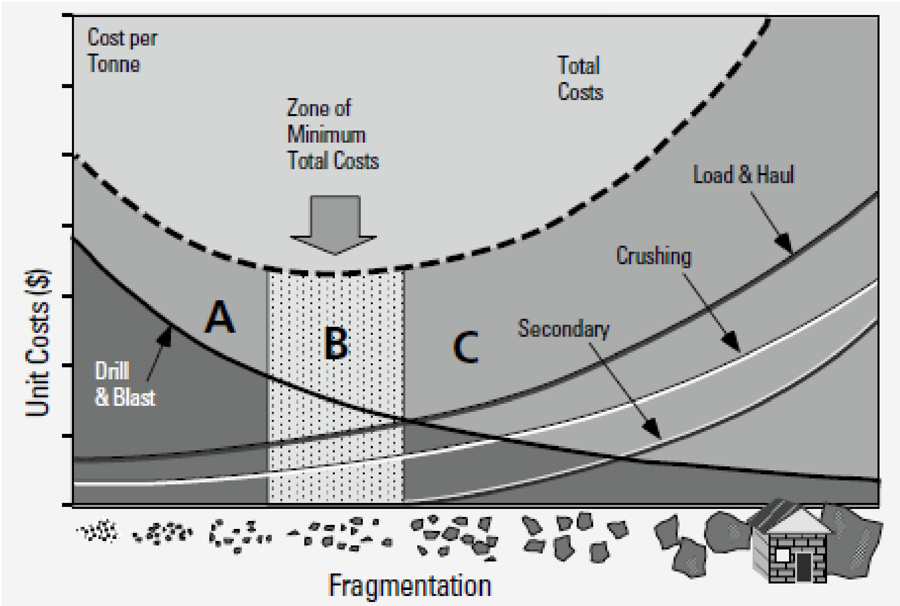


Figure 47 Optimum cost per ton due to increased blasting cost (Atlas Copco)

Claassen et al (2012) summarized approaches to solve complexity in mining, such as the natural tendency to solve complex problems is to break down into various parts and establish the root cause of a problem. Simplicity in a mechanistic world is created by dividing the system into smaller parts therefore focusing on the parts not on the entire system. Simplicity can only be applied once complexity is understood well. Flow and product quality constraints and the like of the key leverage points are paid special attention for value driven operation. Therefore, various measures and targets should be part of the process flow to manage and control a complex system.

Figure 47 (Atlas Copco) is a typical example of to the optimization of drilling and blasting and other mining processes in relation to the fragmentation size classes. This figure is a classic way of representing why drill and blast cost should be high to be more profitable.

A blast management and a cost control system are needed to track the quality of blast implementation and its impact on downstream processes.

Total mining cost is a combination of mining processes, pit operations and people resources such as management and mining crews. Mining processes include drilling, blasting, loading, hauling, crushing. Further detail can be added which are pure mining processes as shown in Figure 48 where shown excludes costs related to equipment purchases, overheads, mineral processing, sales, marketing and freight costs of the final product. Typically, total mining cost is calculated based on the operational and maintenance costs.

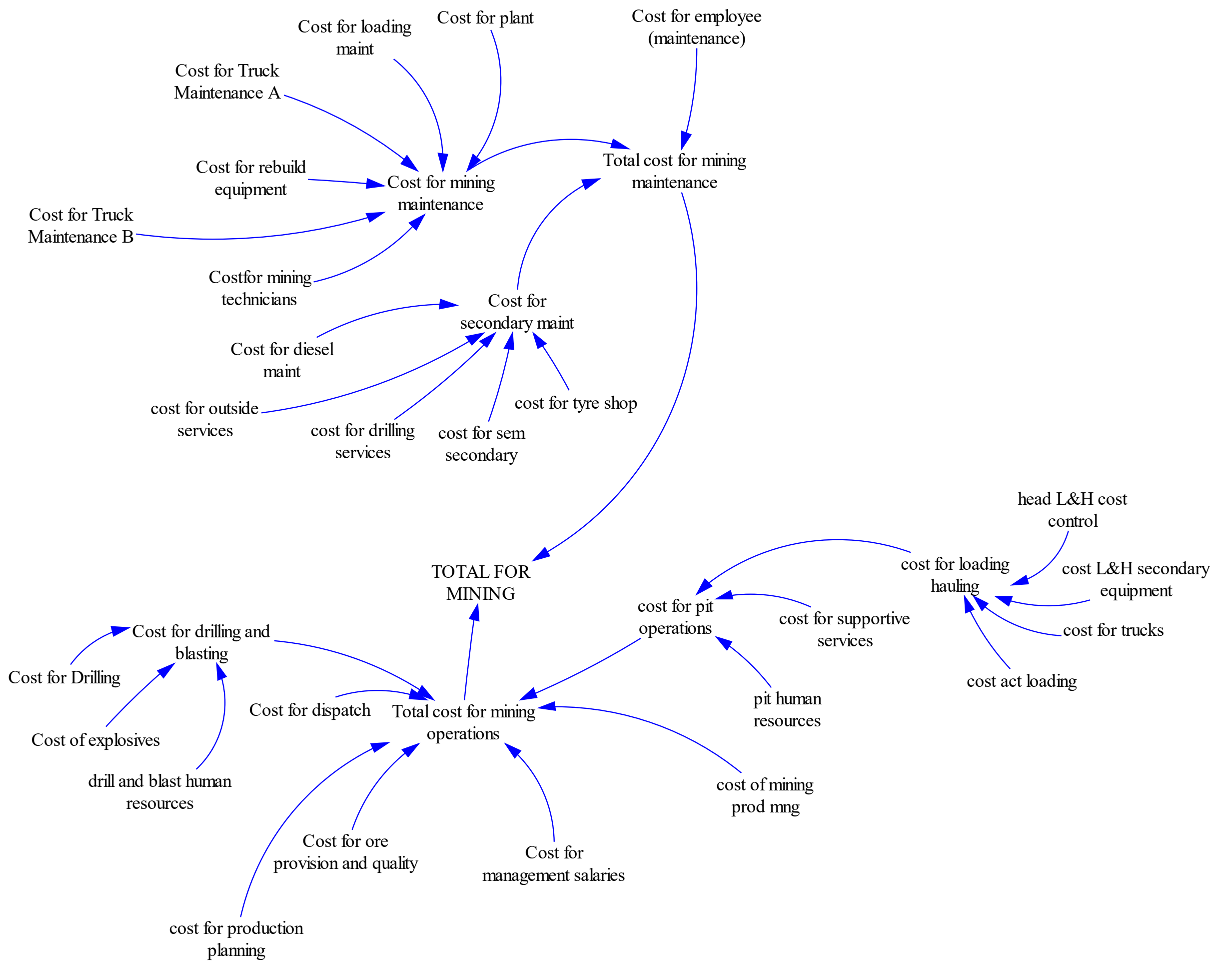


Figure 48 Main cost items at a mine (Source: Author)

Mining companies have been trying certain improvement methodologies such as listed in the Table 8.

Table 8 Typical mine improvement methods

Methodology /Aspects	Cost based approaches	CI/TQM	Six Sigma	TOC	MRTM
Default approach	Reduce cost at all costs	Do things better Equal focus on all parts Technology assist transformation	Manage key business processes Technology assist transformation	Optimize physical throughput/flow	Optimize physical flow and flow of ore characteristic attributes Technology assist embedding principles
Thinking logic applied	Mechanistic	Mechanistic	Mechanistic	Systemic	Systemic
Benefits in mining	Sweat assets	Improvement culture	Analytical and measuring tools and	Flow focus	Focus on variability in ore and ore body morphology as key

			techniques		performance drivers

There are models that are designed to evaluate and quantify mining equipment selection based on the total cost of ownership (TCO) or life cycle costing (LCC) and a stochastic model as discussed in (Gransberg, 2015) and Noorbakhsh (2019). Each one serves a different purpose. It is envisaged that life cycle costing is probably useful for analysis upon the completion of the mine life cycle relatively easy and popular but cost of ownership is a better option to include in a modelling environment as it is dynamic however believed to be not so popular as discussed by Noorbakhsh (2019). The production, productivity and economic outcomes are typically determined based on cost of production loss, cost of energy, operating and repair, cost of sourcing the machinery and acquisition, preventive maintenance, design, disposal, spare parts and operational availability.

Some pre-processing may be necessary for some of the data. Performance of equipment is measured by mechanical availability, operating capacities, operating environment, capital and operating costs, service and support. Time definitions and inputs, OEM technical and equipment inputs are the other parameters. Inputs and outputs must be carefully defined and integrated into the model.

A typical mine has three types of workflows: Geological, mine design and excavation. The geological section deals with mineralogy,

- Structural Geology, Ore Body Model, Mineral Resource Valuation (MRM), Geotechnical and hydrogeology.
- Mine design deals with pit design, haul road design, production profile, equipment for excavation such as drilling and blasting, loading, hauling and auxiliary. Economics deals with Opex, Capex, NPV and IRR. Then there is a labour component in the mine design component.
- Excavation deals with diggable, rippable or drilled and blasted ore bodies.

4.2.6 Production and Productivity as Profit Drivers

Productivity is concerned with producing output efficiently, and it addresses the relationship of output and the inputs used to produce outputs. Profitability in organizations can change for reasons that have less to do with productivity but more on metal prices and it is strongly connected to the creation of value (Humphreys 2020). Measurement is part of the diagnosis and analysis process also to be able to track progress during an improvement program.

It must be emphasized that increased production does not necessarily mean increased productivity. Comparison can be made to previous conditions or to another similar condition departments or competitor. Therefore, it is a relative concept.

Improvement in productivity for human capital means increase in the output of the unit item being produced per employee. However, it should be noted that the increase in labour productivity may be due to many reasons such as change in technology (as explored in this thesis) or due to training or smarter working employees. Therefore, difficult to isolate it in the SD modelling for this thesis for the quantification of the impact of change in technology.

The productivity in a mine depends on certain processes that are identified as high-cost items. They are mostly dependent on time, energy and fuel, consumables, human resources, capacity and unit production per capacity. Some examples of such relationships are listed below (Kennedy, 2000, Loots 2013).

Fuel Burn determined based on:

- Nominal engine capacity (kW)
- Specific fuel consumption (lt/kW/hr)
- Load condition (in percentage to operating load)
- Operating time under load condition

Truck speed determined as a function of:

- Rim pull (propel/retard)
- Total resistance = grade resistance + rolling resistance
- Vehicle weight (empty and full load)

Instantaneous rate of drill penetration (IRP) is determined based on:

- Bit loading/pulldown force
- Rotation speed
- Bit diameter
- Rock strength
- Rock factor

Quantitative outcomes are:

- Fleet sizing
- Total cost of ownership
- Capital cost
- Operating cost
- Machine productivity and production estimates
- Cash flow
- Machine acquisition schedules
- Fuel consumption
- Cycle times
- Production weighted haulage profiles and distances
- Instantaneous rate of penetration per drill rig
- Maximum production rate and real production rate

Productivity units are typically drilled meters (m/hr) and loaded and hauled tons (t/hr).

Most influential productivity parameters are bit loading/pulldown force, rotation speed, bucket/dipper size, loading speed, loader-truck matching, truck payload, rim pull curves, etc. (Gokhale, 2010)

Production estimates depend on operating conditions such as climatic conditions, rock strength, ore body characteristics such as grade distribution and shape, roadway condition and machinery performances (Gokhale, 2010).

4.2.7 Measurement, Metrics, Equipment Effectiveness

The definition and use of overall equipment effectiveness (OEE) over the years has been widely debated (Elevli and Elevli, 2010). It is a measure of total (complete, inclusive, whole) equipment performance – the degree to which the asset is doing what it is supposed to do, OEE measures total performance by relating the availability of a process to its productivity and output quality. It addresses all losses caused by the equipment, including;

- Not being available when needed because of breakdowns or set-up and adjustment losses
- Not running at the optimum rate because of reduced speed or idling and minor stoppage losses
- Not producing quality output because of defects and rework or start-up losses.

- Calculating OEE is possible based on % availability, % productivity and % quality

Improved drilling would lead to more efficient processes such as improving loading and hauling of the ROM (run off mine) material, allowing the increased production targets to be realized. The improved mining efficiencies mean that the required *production rates can be realized with less equipment.*

Efficiency measurement tools need to be considered such as operational costs, labour costs, availability, MTTR, MTBF, total tons moved and the most important one is cost per ton to determine all the areas that a change in process such as automation has an impact on.

The next benefit area is reduction of production losses due to delays. How much of the delays can be attributed to drilling quality and efficiency? This is difficult to quantify as it changes with changing production environment. but it is not easy to isolate drilling related changes in a real environment considering all the other changes within the real mining environment. But this is possible via simulation.

It can be assumed that automatic capture of events during drilling frees operators from paperwork and removes inaccuracies and inconsistencies of manual systems. This is one of the benefits of automated drilling. By how this improves the rest of the mining cycle is not easily quantifiable. The sensible approach to solving this problem is to break down complex issues to common causes, enabling continuous improvement in reducing costs and streamlining operations. (Johnson 2016).

In addition to above, the data capturing processes in a mine environment are not often validated in terms of accuracy. Mines have information management technology for mobile equipment; some of them are captured automatically and some are captured manually. Each department interprets the data differently often leading to some loss of detail in data, therefore interpretations are biased.

The automation also brings automated data capture which helps manage the mining process better. The following has been further added by Johnson (2016).

Data collection –

- Automatic record creation on occurrence of a downtime event or manual entry
- Editing and validation – Adding of additional details such as cause location, classification and cause code.
- Information validation business processes and workflows if desired

- Audit trails (to provide traceability if data is changed) and flexible security (to allow add/delete/edit access to appropriate people only)
- Ability to capture real (where production has ceased) or virtual downtime (slow running)
- Ability to capture time overruns or delays on maintenance or project activities, as well as continuous production
- Complex event capture conditions (such as conveyor is running but motor is drawing low current)
- Ability to automatically assign causes where possible from alarms and other process data
- Ability to split events (e.g., the plant is down for scheduled maintenance, but there are delays in re-starting for production reasons).

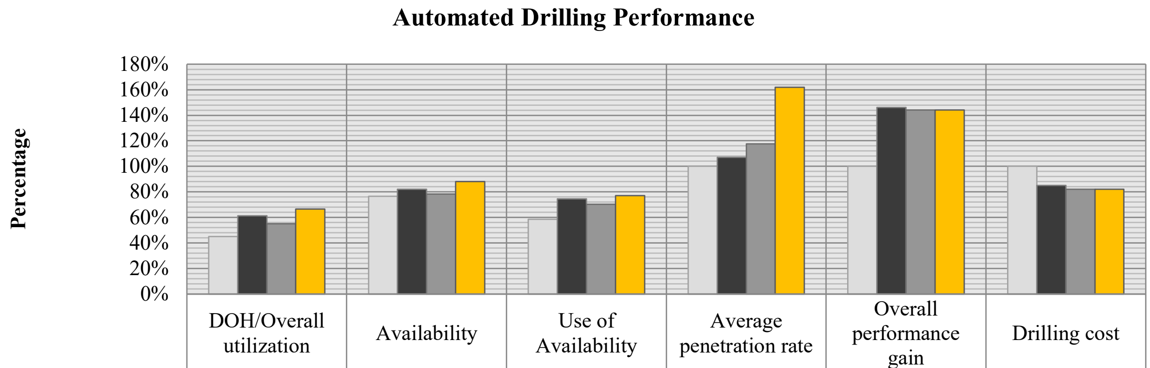
A good mine management system should at least track the following for maximized production in an automatic mode (Modular Mining)

- Crew line-up
- Fuel service management
- Payload analysis.
- Auxiliary equipment performance
- Controlled ore blending at crushers and stockpiles to meet material quality goals
- Tire management
- Shovel hang time
- Unused plant capacity
- Safety and incident records

All of the above are quantifiable measures to be included in SD model.

4.2.8 Previously Measured Performance on the Case Study

The automated drill performance is compared by Henk Pienaar (2015) against the conventional drilling. The original target and the derived target during automation of drilling machines at Kolomela measured is shown in Figure 49. The measurements indicated below can be used to benchmark against the simulated values.



	DOH/Overall utilization	Availability	Use of Availability	Average penetration rate	Overall performance gain	Drilling cost
Conventional	44.9%	76.7%	58.5%	100%	100%	100%
Automated	61.2%	82.1%	74.6%	107%	146%	85%
Derived Target*	55.16%	78.2%	70.2%	117.5%	144.2%	82%
Charter target	66.5%	88%	77%	161.9%	144%	82%

Figure 49 Automated drilling performance summary

The main benefits of automation in the first glance can be listed as follows as discussed by Pienaar (2015):

- “The target DOH (The direct operating hours) exceed the derived levels,
- availability and use of availability over the test.
- The automated drill failed to meet the derived target on penetration rate, but still realized an improvement of 7% over conventional drilling.
- The automated drill exceeded the derived target on “overall performance gain”, achieving an overall improvement of 46% over conventional drilling reportedly this is equivalent to two conventional drills, and
- Drilling cost, even though not reaching target, reduced by 15% from conventional drilling.”

Kolomela mine is looking at three productivity drivers for blast hole drills namely:

1) “Direct Operating Hours” (DOH),

2) “Penetration Rate “expressed in meters per hour influenced by rock hardness, ability of the rig as well as operator, and lastly,

3) “Drilling Yield” expressed in tons per meter, and which is primarily influenced by rock type, required particle size distribution (PSD) and the type of explosives and drilling accuracy.

These three measures are used to obtain tons per annum.

The following has been expressed in terms of drill productivity with all three drivers.

Table 9 Drill productivity summary for conventional and automated (Pienaar, 2015)

Increased drilling capacity through automation				
Capacity driver / KPI	Drilling capacity	Annual DOH	Penetration rate	Yield
Unit	(Mt/drill)	(h)	(m/h)	(t/m)
Conventional	5.95	3883	18.2	71.6
Automated	8.38	5291	19.5	80.5
% increase/(decrease)	41%	36%	7%	12%

The main question here is why as an example drilling capacity increased from 5.95 to 8.38 Mt/drill. This question may be addressed by looking at the parameters that influence drilling capacity. Similar questions can be raised for the other productivity measures listed in Table 9.

The successful execution of mining depends on how accurate the planning and scheduling process is. The planning process depends on effective communication of what has been achieved at the mining block such as rates, qualities and quantities for the unit processes. If the information is delayed in terms of what has been achieved at the production face, then the whole planning and scheduling will be based on wrong feedback or input for the next short-term planning cycle.

Quality combined with wrong planning and scheduling could lead to further production delays or rushed work. Undulating bench conditions, bad fragmentation that results in loading and hauling cycle time increase, environmental damage or unstable final wall conditions are some of the negative effects of inferior quality drilling. For example, the scheduled loading times will not be sufficient due to bad fragmentation. The next round of drilling will also be impacted due to uneven floors, not properly cleaned blasting blocks due to toes or high grounds, etc. In chapter three it was mentioned that x affects y, y affects z and z comes around and affects x. To stop exponential growth of errors additional effort and resources will be needed to correct this error. This may mean either time delays and/or additional costs.

Certain steps need to be followed to quantify the impact of drilling. Firstly, input and output requirements for the model is to be determined. The following are required to capture the core information for a realistic simulation at multiple levels that are used as inputs into the newly developed model:

- Heavy mining equipment and fleet numbers (resources)
- Types of processes and sequences
- Capacities of equipment such as drill rate, penetration rate, bucket fill factors
- Typical geological and geophysical characteristics that will have a direct impact on the mining sub processes and the drill scheduling
- Working hours and schedules

The output of the simulation will be in the form of key performance indicators typically used in mining industry. They are effectiveness and productivity therefore profitability as explained in 1.8.

Effectiveness is defined as a process characteristic indicating the degree to which the process output conforms to the requirements (Liliane and Muchiri, 2008). A production index measures change in output, while effectiveness measures the conformity to the output requirements.

Productivity is the ratio of input to output, this can be measured in terms of time, money and product. Then there is profitability based on output quality and amount within the shortest time possible in simple terms. However, dependencies of sub systems may not call for time efficiency due to possible system imbalances and creating bottlenecks further in the system. This dynamic nature of mining cannot be effectively analysed without proper simulation and visualization

What does operational level data reveal about how effective the processes are running? The argument is that a mine plan defines and communicates goals such as production volume, cost and revenue for a certain period. All the processes have a very closely knitted cause and effect relationship. If correct equipment is selected, haul roads are to a standard, and support services are functioning properly, the mining process should achieve its goal.

Typically, there are four variations of tons moved at a typical mine:

- Planned and mined
- Unplanned but mined ahead
- Unplanned was required to be mined therefore behind with planned and scheduled
- Planned and not mined

Measuring these processes starts firstly at planning level for the following reasons that is typical in a surface mine environment:

- Blocks being mined according to the sequence that is supposed to be mined for maximum NPV.
- Grades and tonnages expected to be found are there and there is low risk of error in estimated grades by the geologist and grade control departments
- Adherence to plan on a long-term medium term and short-term basis are measurable
- Ratio of tons mined to ratio of tons planned. Stemming from this are some sub KPI's that are: Planned drilled meters versus actual drilled meters, blasting planned versus actual blasting occurred on a weekly, monthly or quarterly basis.
- Loading planned versus actual and the link between blasting quality and expected loading rates are established. Correct loading of material is a quality issue which affect the grade and throughput at the plant and stockpiles.
- What if overall planned tons were over 100 %? This may mean that planning has underestimated capacity therefore planning failed to correctly plan. If tons moved are less than 100% and utilization of existing equipment is low this means scheduling as well as planning failed.
- Grade control helps with spatial and geological reconciliation

Time, money and product quality all depend on each other and should not be treated separately according to Lumley and Mckee (2013) mentioned that companies can follow either a cost strategy or a volume strategy. A high focus on costs may mean that fewer but highly utilized trucks versus loaders sitting idle (Lumley & Mckee, 2013). The opposite is true for production volume driven operations where the loader should be as productive as possible to meet the demand required.

Capturing the as is status of a running mine with all critical variabilities can be a way towards a simulation-based solution. Thereafter the status change needs to be simulated by changing the key parameters to simulate the changing environment due to a disruptive technology and changes in processes. The parameters that are most affected due to a change in the mining environment are of critical importance and therefore there must be enough detail captured of that specific parameter.

The first step to capturing the status change when a new system or technology is injected is as follows:

- As-is status is to be modelled with all interdependencies
- Build structures and formulate the whole in the present operation for the future prediction.

- In order to investigate the case all interdependent variables and dynamic interactions of these variables are to be captured.
- Each variable has a stock with inflow and outflow or a cause-and-effect structure. These connections define the behaviour of the system whether it is positively increasing linearly or in a loop.
- To model such a system, there is a need to understand the system constraints as well as buffers, delays, queues, bottlenecks and feedback loops.

There are a few examples of system dynamics application in mining engineering, some are reviewed here for the reader to comprehend how input and output relationship is modelled in System Dynamics (SD). These examples aided this researcher to understand to what detail and flexibility the system dynamics models are created.

Example 1

Inthavongsa et al (2016) developed an approach using SD as a decision-making tool for an open pit mine planning project where a real option decision framework (RODF) with practical applicability to a mining investment project was.

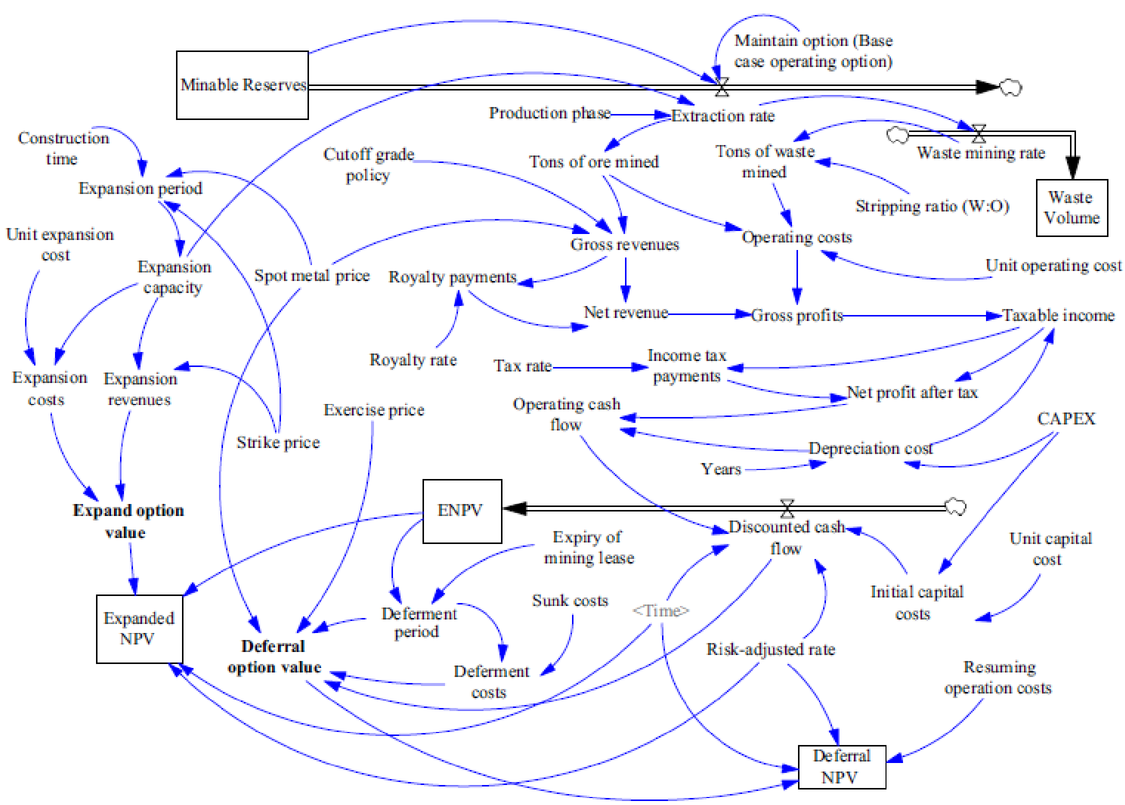


Figure 50 System dynamics models for evaluating economic values (Inthavongsa, 2016)

Their findings suggest that the project has a value to an investor across different time horizons and it is of paramount importance to find an optimal time to exercise a strategic option. Their model can be seen in Figure 50.

Example 2

A study by Ayman (2013) carried out with an objective on general mine planning optimization focusing on low grade big tonnage ore bodies and mostly focusing on mineral processing optimizations with some mining process input. Screen shots are shown in Figure 51 captured from Ayman's thesis.

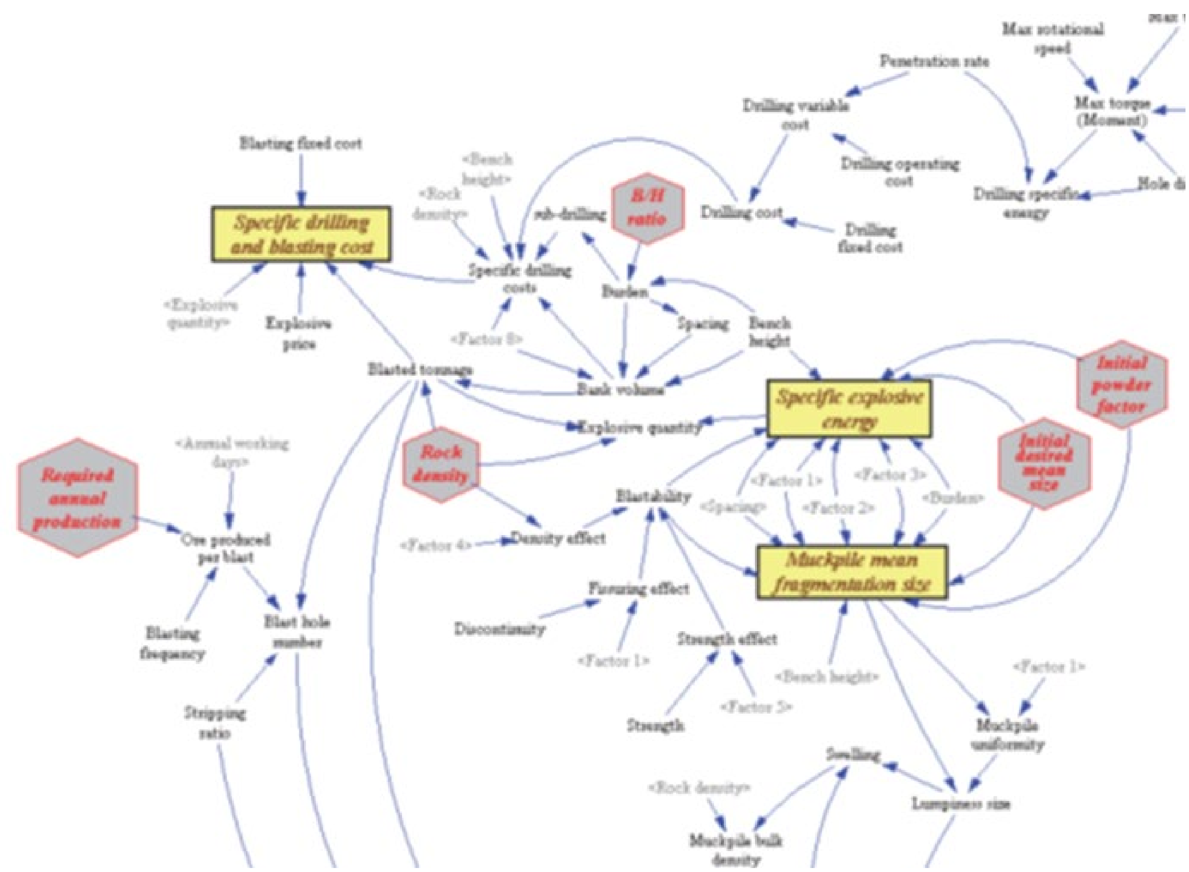


Figure 51 Screenshots for the Drilling and Blasting sub-model (Ayman, 2013)

The author's vision on system dynamics applications was improved with the two examples and Sterman (2000) was the key to make sense of it all on how the modelling should be done for this thesis.

Sometimes, it may be necessary eliciting the data, where no data exists for a model parameter and the cost or duration. Madachy and Houston (2018) explain the way in which these are dealt with in the book called: *What every engineer should know about modelling and simulation*.

Hustrulid et al (2013) state that challenges of complexity urge mining operators to cut costs to increase efficiency. The unit operations such as blasting, loading, hauling, crushing and processing are all impacted by the drilling performance and therefore it is the key factor. Beyoglu (2016) comments on the downstream effects due to the drilling and blasting process. He studied how electric shovels are affected by muckpile looseness. A swollen muckpile with a profile of muckpile tailored to the machinery will yield much smoother and more efficient loading. Although digging performance of shovels are linked to muckpile and fragmentation there are other factors that are used as measures of loading performance. Dipper fill factors, dipper payload, dig rate and frequency of dig cycles have also played a role whether directly related to muckpile compactness or not. The question that arises from this is by how much of the muckpile quality can be attributed to the cycle times is a measurement that needs to be determined, and it is also highly variable from operation to operation. This can also be attributed to the drill rig properties as well as rig operator influence. This has been addressed in this thesis by including muckpile looseness in the model created.

Operator's influence on drill performance has been overlooked by many mines in the past therefore Simpson (2016) developed a system to measure operator efficiency and listed the key performance areas. Simpson categorizes KPI's for manual drilling; they will be useful information for modelling the efficiency of drilling. Most of these are included in the SD model for this thesis except the independent drilling variables.

Chosen KPIs are listed by Simpson (2016) are:

1. Dependent variables:
    a. Instantaneous Rate of Penetration (ROP) (m/hr),
    b. Cycle time (min/hole drilled) (this measure is not inclusive of drilling time or operating delays); and
    c. Accuracy to design (m) (this includes two separate measures for the drill hole collar and toe accuracy to the drill pattern design in easting and northing, x- and y-planes respectively).
    d. Operating time (OT),
    e. operating delay (OD),
    f. operating standby (OS),

g. no scheduled production (NSP),
h. scheduled loss (SL),
i. unscheduled loss failure (ULF),
j. unscheduled loss other (ULO) – all measured in (hr.).
k. availability (% calendar time),
l. utilization (% of available time), and
m. use of availability (% of available time).

2. Independent variables:
    a. Weight on Bit (WOB) (kN); rotary speed (RPM); torque (TRQ) (kNm); Air Pressure (AP) (Pa).
    b. Difference in bearing from design (angle˚), difference in mast angle from design (angle˚).
    c. Shift change (hr.), lunch break (hr.), water or fuel – refilling (hr.), fatigue break (hr.), machine checks (hr.), and accident damage events and duration (hr.).
3. Controlled variables:
    a. Geological domain.
    b. Drill pattern design.
    c. Drill rig; and
    d. Drill bit.

Furthermore, parameters that were seen to be of importance but were either not captured in the data, or found to be unsuitable for application:

Excluded Parameters:

- Drill bit failure modes.
- MTBF (mean time between failure); and
- MTTR (mean time to repair).

These parameters were excluded as they cannot be associated with an individual operator and have been influenced by multiple operators using the same drill. Therefore, they cannot be used as objective measures of an operator's performance in these areas.

Mine productivity index (MPI) is a useful tool based on the four elements of MPI mine Productivity Index) – labour, capital expenditure, non-labour operating costs and production volumes. Mc Kinsey & Company (2016) reports on their company site that there are various trends by companies for improving productivity by spending capital or investing in technology or cutting on labour costs.

If the companies focus on labour productivity, typically measuring the final product output but not the total material moved – per person employed, it will not take cognisance of the geological variability. In addition, if the companies focus on the OEE (overall equipment effectiveness) metric using dispatch data they learn about equipment operating time and delays and availability, utilization, tempo and performance in isolation of each other. That is, drilling section is only concerned about drilling OEE, load, and haul only focusing on load and haul OEE without necessarily understanding the strong interrelationships between the two OEE measurements.

In a mining environment there are various components that should not be evaluated in isolation, which are

- Geological Setting
- Equipment OEE
- Labour productivity
- Maintenance
- Time
- Planning
- Cost

The total productive management (TPM) approach therefore has been discussed recently by Emery, J.C. (1998) that TPM technique as a performance improvement considers ownership as behavioural approach to equipment management, relationship of OEE to the team's environment, i.e., harsh geological conditions and variations.

Dressler (2015) reports that the monitoring and control plan is essential in a project planning environment, and he defines the criterion of performance measurement with three E's: Efficacy, Efficiency, and Effectiveness. The technology adopted requires that three E's are effectively measured for maximum confidence throughout various phases of the implementation, i.e., trial phase, rolling out phase and full switch to the new technology or systems. The full benefit or impact can only be realized once the technology is rolled out to all machines. The automation also brings automated data capture, which helps manage the mining process better. The following are some more of the important points that can be added to the benefits of automation as discussed by Johnson (2016)

Data collection :

- Automatic record creation on occurrence of a downtime event or manual entry
- Editing and validation – Adding of additional details such as cause location, classification and cause code.
- Information validation business processes and workflows if desired
- Audit trails (to provide traceability if data is changed) and flexible security (to allow add/delete/edit access to appropriate people only)
- Ability to capture real (where production has ceased) or virtual downtime (slow running)
- Ability to capture time overruns or delays on maintenance or project activities, as well as continuous production
- Complex event capture conditions (for example conveyor is running but motor is drawing low current)
- Ability to automatically assign causes where possible from alarms and other process data
- Ability to split events (e.g., the plant is down for scheduled maintenance, but there are delays in re-starting for production reasons).

A good mine management system should at least track the following for maximized production. The list also highlights how a change in system such as automated drilling will impact them.

- Crew line-up: Automatically assign personnel to equipment before the start of a shift and ensure that each equipment unit has been allocated a suitably qualified operator. Gather information from various records (roster, equipment qualifications, end of shift) before assignments are made
- Fuel service management: Increase overall haulage efficiency by minimizing refuelling events. Streamline fuel management by allocating trucks to fuel stations only at optimal times and fuel levels
- Payload analysis: Capture real –time payload information from OEM sensors and third-party payload systems on loaders, shovels and trucks. Access information for analysis through standard reporting utilities.
- Auxiliary equipment: Track the status of each piece of auxiliary equipment, prioritize tasks, and assign them remotely to operators. Monitor maintenance performance, plan operation and fleet requirements, and identify problem areas
- Blending: Control ore blending at crushers and stockpiles to meet material quality goals

- Tire management: Detect tire overheating through direct interfaces to online tire monitoring systems. Minimize rock cuts through spillage geo-tagging and automatic assignment of clean-up tasks
- Shovel hang time
- Unused plant capacity
- Safety and incident records

All the above are quantifiable measures that need to be part of KPI's that determine efficiency of an existing system.

4.2.9 Heavy Mining Equipment Key Metrics

Key performance areas are reviewed by Dougall and Mmola (2015). They have mentioned the reasons to measure performance and how it should be used in day-to-day activities. They are:

- To learn and improve
- To report externally and demonstrate compliance
- To control and monitor people.

A basic list of mining KPI's are listed below:

- Average bucket weight
- Average fuel use per machine
- Average loading time
- Average number of dumps per hour/day/week/month
- Average loading time
- Average number of dumps per/hour/day/week/year
- Average payload per equipment
- Average swing time
- Cash Operating costs per unit produced
- Change time between cycles
- Cycle distance
- Cycle time
- Degree of purity and physical characteristics
- Dilution of ore
- Dump time
- Efficiency of metallurgical recovery

- Empty stop time
- Empty travel time
- Fuel usage in lt/hour
- Lifting costs
- Loaded stop time
- Loaded travel time
- Total loading time
- Number of equipment failures per day/week/month/year
- Max Payload per equipment
- Number of holes drilled per day/week/month/year
- Percentage uptime of equipment plant etc.
- Tons of ore feed
- Tons per hour
- Tons per load
- Total minutes lost per shift due to breaks
- Unit variable costs
- Utilization
- Waste per ton
- Waste volume

Machinery History

Involves work hours, work done, fuel used, etc. There are no calculations involved but it is historic accumulation of usage data. The records used are SMU, service history and component history. It is useful for determining the equipment age and factors of life and load factor. It is used for benchmarking against other similar machinery.

Daily Production Rate

It is calculated by mine planning engineers in three levels, long term, medium term and short term. The daily production rate depends on the mine design criteria which are dependent on geological attributes of the ore reserve, mine design factors and planning. It is particularly important as it will influence the equipment selection process. It will also impact the operation by affecting the production requirements of the equipment as well as number of equipment to be used. The equipment efficiency is monitored continuously and optimized through efficient equipment allocation in the fleet and through effective dispatch practices.

4.2.10 Surface Mine Equipment Management and Design Criteria

Mining equipment management metrics should be used based on why it is being calculated, what formulae are used in calculating them, and what the outcome of the metrics are for.

Moreover, Table 10 contains essential measures useful during the SD modelling.

Table 10 Mining Equipment Metrics (Caterpillar, 2006)

MINING EQUIPMENT MANAGEMENT METRICS				
	METRICS	CALCULATION	WHAT WILL I HAVE	Benchmarks
1	Machine History Hours/SMU/Fuel/Work Done	———	**Equipment Age and Factors of Life and Load Factor**	YES
2	Availability	$A = \frac{\text{Operating Hours}}{\text{Operating Hours + Maint. Hours}} \times 100\ (\%)$	**Am I meeting the Mine Plan requirements?** Is performance satisfactory?	92 % new 88 % old
3	Utilization	$U = \frac{\text{Operating Hours}}{\text{Programmed Hours}} \times 100\ (\%)$	**Equipment Use.** Labor/Parts/Burden estimation	90 %
4	MTBF Mean Time Between Failure	$MTBF = \frac{\text{Operating Hours}}{\text{Ns. of Failures or Shutdowns}}$	**Reliability** (Impacts operations & maintenance efficiency) $A = \frac{MTBF}{MTBF + MTTR} \times 100\ (\%)$ **Turnaround** (Return to productive work)	80 hrs new 60 hrs old
5	MTTR Mean Time To Repair	$MTTR = \frac{\text{Down Hours}}{\text{No. of Shutdowns}}$		3 - 6 hrs
6	MR Maintenance Ratio	$MR = \frac{\text{Maintenance Man-Hours}}{\text{Operating Hours}}$	**Effort invested** **Quality of Repairs & Labor**	0.2 new 0.3 old
7	% SCHEDULED WORK	$SW = \frac{\text{No. of Scheduled Shutdowns}}{\text{Total No. of Shutdowns}} \times 100\ (\%)$	**Who is in control!** **Maintenance or the machines.**	80 %
8	TOP TEN PROBLEMS	Pareto Analysis	**PAIN Location & Severity.** How should we response to failure: -Parts Inventory -Maint. Training -Improved Inspections/Tests -Operators/Supv. Training -etc... Fundamental Information for Problem Management	YES
9	SHUTDOWNS PER SYSTEM & COMPONENT	Shutdowns / Component Area Shutdowns / System Benchmark to Others & History		YES
10	SERVICE ACCURACY	$SA = \frac{SHP - SHE}{SHP} \times 100\ (\%)$	**Planning / Scheduling Efficiency** Are Repair Centers following the plan? General planning accomplishment	Within 10 %
11	BACKLOG AGE	———	**Am I standing on a solid base ?** Equipment general condition, Planning department performance, Inspection program quality, Execution of the plan, Opportunity work perfomance, etc...	YES
12	RECORD-KEEPING	———	**Information Source of the Maintenance Operation**	100 %
13	TRENDS	———	**Prognostics** Component Life Mgmt & Enhancement Problem Management	YES Component Life Targets

The method of operation

The operating method is determined during the mine design phase and mine design criteria are used as the data input. This is not tracked or managed continuously however it is important in terms of system boundary establishment in SD.

Mineable Reserve/ Life of Mine

It is calculated based on geological sampling and shaped with the economic influencers such as demand, prices, costs and grade and tonnage. This depends on the confidence level of the geological information. A resource model is the main driver for the calculation of life of mine and needs to be updated regularly after mining starts based on refined sampling. Life of mine may extend depending on the amount of *probable* reserves and they are added to *proven* reserves.

Fragmentation

Size distribution of blasting is an important factor, and it changes the product quality. For example, the fines are not desired in iron ore mines due to reduced selling price of the fines in the final product. Size also matters in terms of loading and hauling efficiencies. It affects the fill factors, digging rates, therefore cycle times of loading and hauling equipment. Size distribution of a blasted muckpile is not easy to determine. It needs to be sieved for estimating the level of fragmentation which is not feasible. But there are methods to predict size distribution of a blasted muckpile. This matter is discussed in 4.2.5.

Bench Height

It is used for equipment size determination and fleet management. It is also used as an input in many calculations from drilling and blasting to loading and hauling. Bench height and berm widths affect the slope angles, rock stability, drilling capability, drilling patterns for effective blasting, Number of drill steels required, and the type of drill rig are also affected with bench height.

Soil and Ground Characteristics of Ore and Waste

Typical characteristics are hardness, abrasivity, digability and geometry of the ore body. It affects muckpile shape and fragmentation quality, as well as slope stability and equipment efficiencies, such as drilling rates, loading rates, boulders and fines, blasthole stability reactivity to explosives, and behaviour when it is damp or wet. Changing rock characteristics or mixed ground conditions may affect drilling and blasting efficiencies.

Legislative Requirements

This defines the constraints to operate within this. May affect blasting times and radius, quality of air due to soil characteristics and dust generation, vibration due to excessive amount of explosive per initiating time, etc. Proximity to urban areas and public infrastructures may pose a risk and need to be managed. In addition, working conditions due to under designed machinery and mining practices may impact on the health and wellbeing of employees. While legislation is there to protect the people and environment it is also a hindrance on mining efficiency due to work stoppages therefore delays in mining processes.

Swelling Factor

Mining efficiencies and/or progress are measured either in mass (tons) or volumes (m3). It is important for surveying practices or reconciliation of tons and volumes produced. During conversion from one unit to another swelling factor becomes very important. The production planning is based on tons and volumes. Truck capacities, bucket size therefore equipment size may all depend on this important variable. It needs to be continuously monitored through sampling and discrepancies can be attributed to swelling factors, however, within limitations based on ranges of swelling factor per rock type. Swelling factor estimation depends on how complex a block of ground is.

Haul Distance

Hauling distance for trucks may vary depending on destination of ore or waste and it depends on the physical dimensions of the orebody and infrastructure layout regarding dumping locations for waste, stockpiles, plant location, crusher location, distance to service bays, pit design and depth. This is a predictable variable based on size of the operation and surface occupied by the mine operation.

Haul Roads/ Infrastructure

Haul road quality has a large impact on the cycle times and fuel consumption due to ramp design and quality. There is also a large role played on safety records of an operation due to relatively higher number of incidents experienced in a mine is attributed to haul road design and conditions.

Fill Factor

It is calculated as the percentage of fill out of the bucket or scoop. It also applies to truck fill factor It has a direct impact on overall productivity and equipment effectiveness. It needs to be constantly monitored through weighing at weigh points per shift. It may indicate to management on drilling loading and hauling practices as well as determines the need for operator training.

OEM (Original Equipment Manufacturer) Specifications

These are technical parameters supplied by manufacturer of the equipment in the mine. It gives an indication of the equipment capacity, rate and output per machine. Type and make, power pack installed (energy demand), manoeuvrability, flexibility, etc are some of the information that is sought after from OEMs. This information can be used initially for equipment selection, later for benchmarking against actual productivity, life of equipment in terms of maximum production expectations. Every equipment needs to be maintained regularly to realize maximum production over its lifetime. Monitoring of health of machinery becomes critical since it has a direct effect on fuel and power consumption and availabilities of these machinery for production. Usage out of stated environments may result in loss of equipment and therefore additional capital costs.

Equipment Size

Determination of equipment size is done through a trade-off study between equipment cost, mining costs, environment related costs, final pit limit, production requirements, dilution and selectivity, flexibility of equipment and environmental impact. Equipment size is especially important for both selection and management. Larger equipment can profit from scale benefits but are accompanied by larger capital and operating costs. If an operation uses differently sized equipment, it is important to match them optimally according to the production requirements and available infrastructure. Larger equipment cost more to operate and should be justified by the amounts or rock that they are able to move within a given time.

Matching of trucks and loaders

Fleet matching can be done with the closed queuing network theory (May, 2012). Or, by plotting the unit production cost for different numbers of trucks per shovel vs. the number of trucks per loader.

Equipment physical characteristics (size, capacity, production rate etc.), unit cost to operate and the number of trucks and shovels in the fleet are the typical input requirements in order to obtain the optimum number of trucks that is matched per loader and the optimum number of loads a shovel needs to fill a truck. This factor impacts the cost of moving material for a surface operation using a truck and shovel system. Also impacts equipment effectiveness and efficiency.

Equipment Dispatch and Allocation Efficiency

Dispatching of production fleet can be done with the Linear Programming (LP) model, during the planning period. This requires remote operating centres and (autonomous) control systems for FMS. The Linear Programming (LP) model calculates maximum shovel utilization and minimizes the number of trucks required for shovel coverage without truck queuing, in real time on a continuous basis. It is used to track and/or improve utilization and efficiency. Allocation efficiency may also help understand the cost of moving material. This is a major component that needs to be managed and tracked continuously to optimize production. Monitoring systems are generally installed that can provide idle time and working time statistics to the person(s) in charge of dispatch. This information is used to calculate optimization of the fleet in real time.

Fleet management systems (FMS) can be used in real time to allocate trucks to shovels efficiently (Afrapoli and Nasab, 2017). The following parameters are a compilation from various mining reference books such as Hustrulid (1999), Kennedy (1990), and Darling (2011).

Flexibility

Flexibility is a measure of how many tasks a single machine can perform and in how many different scenarios, e.g., an LHD is more flexible than a rope-shovel (although it might have a lower production rate). Number of tasks the machine can perform. The infrastructure, haul roads and turning space in the operation will also form data inputs when using the measure of flexibility to match equipment. It does not have to be tracked or managed as it is a measure that remains constant.

Operator Skill

It is important to measure and track skills of operators which in turn helps understand inefficiencies, performance of the machinery and user training requirements.

Mines may need to adopt automated systems to remove the operator related errors. This becomes significant in terms of drilling accuracy, spotting and ground engaging tools consumption.

Maintenance

Predictive analytics and data management can process and analyse large volumes of equipment data to forecast maintenance requirements for equipment. In addition, it uses statistical techniques such as modelling, machine learning and data mining, coupled with advance machine diagnostics. Historical and current machine data are used as inputs. The useful working hours that can be obtained from a machine over its lifetime is dependent on the maintenance quality. It also affects capital and operating cost balances in a mine in a big deal. Machine data are tracked continuously based on hours worked and hours of downtime experienced, as well as information on breakdowns, damage and repair tasks. These informational inputs are used to schedule maintenance activities as accurately as possible to minimize negative impacts on production.

Spare Parts and Consumables Rate of Consumption

This may indicate effectiveness of the tool selection, operation cost, impact on machine maintenance specifically worn out drill bits may cause increased fuel consumption and vibrational body damage to the equipment.

Fuel/Power Consumption

It is calculated as fuel/power consumed per hour and/or per ton mined. Needs to be tracked for determination of costs and efficiencies. Typical inputs are equipment availability and duration the machine worked within the available time

Utilization

It is calculated through equipment effectiveness (OEE). needs to be continuously tracked by measuring the hours equipment spend working within the available operating time. Managed through dispatch and equipment matching exercises.

Availability

This is calculated as Availability = ((Net available time – Downtime losses)/Net available time) x 100. Planned and unplanned downtime for equipment. Inputs are total operating time expected/available, e.g., shift length.

It is continuously tracked by measuring both the scheduled and unscheduled times that a machine was down for maintenance or repair, compared to the available shift time. It is managed by improving maintenance efficiency to strive towards less unplanned breakdowns.

Reliability

Time between failures is measured and tracked which gives a sense of how reliable the machine is or how prone to breakdowns. It has an impact on scheduling and used for equipment availability calculation. It is managed by improved maintenance.

Maintainability

It is a measure of maintainability in the form of:

MTTR = Total Unscheduled Downtime / Number of Breakdowns **Equation 10**

Unscheduled downtime may include number of breakdowns, total maintenance times ease with which maintenance is done or how fast and accessible maintenance is for equipment.

Overall Equipment Effectiveness

OEE = Availability x Performance x Quality **Equation 11**

This is a useful measure to plan for lost time within the operation

Life-cycle cost / ownership cost

$$\boldsymbol{Life - cycle\ cost\ (LCC)\ =\ total\ cost\ of\ equipment/useful\ equipment\ life}$$ **Equation 12**

Ownership cost = Fixed Cost + Operating/Variable Cost **Equation 13**

Capital Cost

Equipment capital costs have a substantial impact on the NPV over the life of mine.

4.3 Realization of the Impact of Automation

Effective blasting is a key business driver for iron ore operations due to poor drilling and blasting has the potential to destroy value by pulverising lump ore that attracts a price premium (McHugh et al, 2010). McHugh et al report the drill automation in terms of key performance indicators such as total production meters, total production hours, penetration rate, average rotary bit life, availability and utilisation. Table 11 indicates the results of West Angela Mine automation compared to manual drills of the same type and environment.

Table 11 Comparison of production and key performance indicators of the automated drill rigs against a fleet of identical production drills at the West Angela Mine (McHugh et al, 2010)

Description	Drill 21R03 Automated	Drill 21 RO3 manual	Drill 21R03 Total	Drill 21R04 Manual	Drill 21R05 Manual	Drill 21R11 Manual	Drill 21R12 Manual
Total Production meters 18/05/2009 to 07/03/2010	99834	48852	148686	207489	163004	146156	109372
Total Production meters 6/09/2009 to 07/03/2010	87863	25099	112962	118649	96605	102454	106172
Total Production meters 18/05/2009 to 07/03/2010	1588	389	1977	1850	1698	1773	2119
Penetration rate (m/h) 18/05/2009 to 07/03/2010	55	64	57	61	54	55	48
Average Rotary bit life 21/05/2009 to 07/03/2010			5950	4750	4550	4350	3050
Availability % 6/09/2009 to 07/03/2010			86	88	83	84	89
Utilisation % 6/09/2009 to 07/03/2010			52	49	47	48	57

McHugh et al, further suggest that on average the automated drill provides a safer operating environment for the driller, produces a more stable hole, produces a more accurate hole angle, drill a more accurate hole depth, is more productive and has a longer rotary bit life.

A comparison of accuracy and speed between manual and automated drilling is also summarised in Table 12.

Table 12 Comparison of accuracy and speed between manual and automated drilling (McHugh et al, 2010)

Description	Manual Drilling	Manual Drilling with GPS	Automated Drilling
Average drill collar error (m)	0.56	0.26	0.14
Average drill depth error (m)	0.51	0.33	0.01
Average tramming time for 7 m spacing (s)	75	43	58
Average levelling time (s)	35	28	40

The article by McHugh at al further confirms that automation does bring benefits in terms of quality of drilling and increased productivity. The paper, however, does not quantify the financial benefits directly resulting from the automated drills. An attempt can be made to quantify the impact of drilling depth accuracy on drilling costs, fragmentation and loading efficiencies. Since incorrect drilling depth may result undulating bench conditions which results in increased effort on bench preparation. Major fragmentation issues regarding boulders in the stemming region may be the direct outcome of the incorrect drilling depth.

Flesher (2018) reports that biggest contributions to higher productivity have come from reducing the mine workforce head counts and boosting labour productivity while modestly increasing output. Employment is reduced by 3% a year while production is increased by 1.8% a year. This may mean reduction in employment leading to more automated processes. Flesher (2018) recommends that mining companies need to ensure more discipline on reducing of operating and capital expenditures especially when metal prices are higher while demand gains momentum.

The drilling automation can be considered as part of a practical management daily in an open pit operation in terms of:

- Production
- Economy
- Training
- Change in behaviour
- Short term and long-term effects

Martin (2015) states that calculating ROI after the implementation of automation solutions is difficult. He discusses that the basic cost of the solution, in terms of price and implementation costs, is easily understood and calculated, but calculating the actual benefit resulting from the solution is a challenge.

Incremental efficiency as Martin (2015) puts it is determined by analysing the value of throughput, energy costs, and material costs to determine the value of the operation efficiency both before and after the project and to calculate the project ROI.

Shortening of life of mine in an operation that uses this technology is also a possibility. Kapte (2009) simulated a tele remote operation on mine life and demonstrated 38% reduction in life of mine. This is due to efficient utilization of an 8-hr shift. Conventional methods provided 5 out of 8 hours that is normally utilized while with remote operation 7.5 out of 8 hours is possible.

Transformation challenges of automation are listed by Prinsloo (2013) as follows:

- "Cultural change
- Fully understanding the application
- Understand and prove the capability of the technology
- It is not a plug and play process since
    - Different product
    - Different environment
    - Different culture
    - Rate of technology evolution affect integration
- Implementation readiness
- Operational readiness
- Support readiness
- Perception of management
- Change management
- Need to persist to take ownership early

The benefits of automation have been listed by (Heiniö, 1999) as:

- Safety: No personnel in the production area, zero LTI's, zero fatalities
- Continuous production and greater utilization of equipment – longer operation hours therefore more tons, more ore and increased utilization
- Real time production reporting for accurate figures
- Real-time condition monitoring
- Production execution and management can be from a remote office environment
- People with disabilities will be able to operate equipment
- Less damage to equipment

Anglo American reports on their public web site (Pienaar, 2016) that the benefits of automation project resulted in:

- 23% gain of direct operating hours
- 18% gain in drill rate (the actual time to drill hole and to move, setup and start the next hole)
- 19% reduction in drilling cost, and
- 70% less injuries and fatalities

"77% of mining specialists think automation is high priority, and 40 % say now is more important than ever" (Somarin, 2014). Mining companies are excited about operational and financial benefits automation technologies offer. Automation of mining processes such as equipment whether fully automated or semi-automated. Benefits are consistent continuous operations, reduced infrastructure and improved communications.

The process of adoption of complex technology into a complex organization may require a formal approach for tracking the effect of the change.

The direct benefits of drill automation can be determined based on the bit life and penetration rates which may vary month to month. Typically, the following needs to be monitored and can be treated as basic KPI's (key performance indicators):

- Engineering downtime
- Operational downtime
- Productivity
- Utilisation
- Meters drilled
- Bit life (megaton/bit)
- Rod life
- Penetration rates (m/hr) per rock type
- Machine pull down forces
- Bailing velocity

Typical reasons for a drill rig standing can be:

- No operator
- Water tank empty
- No pattern
- Travel time

- No diesel
- Blasting process related standby
- Bit change
- Rod/sub change
- Cable move
- Rods stuck
- Deckbush change
- Rod out
- Cable damage

Some of the direct benefits of automated drill rigs that are measurable are listed as follows by Landey (2019):

- Reduction in machine population at a drill site
- Increased monthly drilled meters
- Increased penetration rate
- Increased bit life
- Increased rod life
- Reduction in total drilling cost
- Increased productivity

At a colliery rod change recorded hours per month was 80.7 and this was reduced to 12.1 hr/month after drill rig is automated. For the same colliery average standing time due to no drilling pattern was 694 hrs/month, after automation it was reduced to 80.4 hrs/month. Also, there was a considerable cost benefit since actual expenditure was lower than budgeted which was about 13% less cost per million ton drilled and blasted. Drilled meters was increased by about 27% over a 9-month period.

A typical business would do trade-off analysis to determine the effect of decreasing one or more key factors and simultaneously increasing one or more other key factors for a decision, design or project work. If the objective is minimum cost per ton in a surface mine to maximize profit the following (Figure 52) are valid in determining the cost of the production per ton.

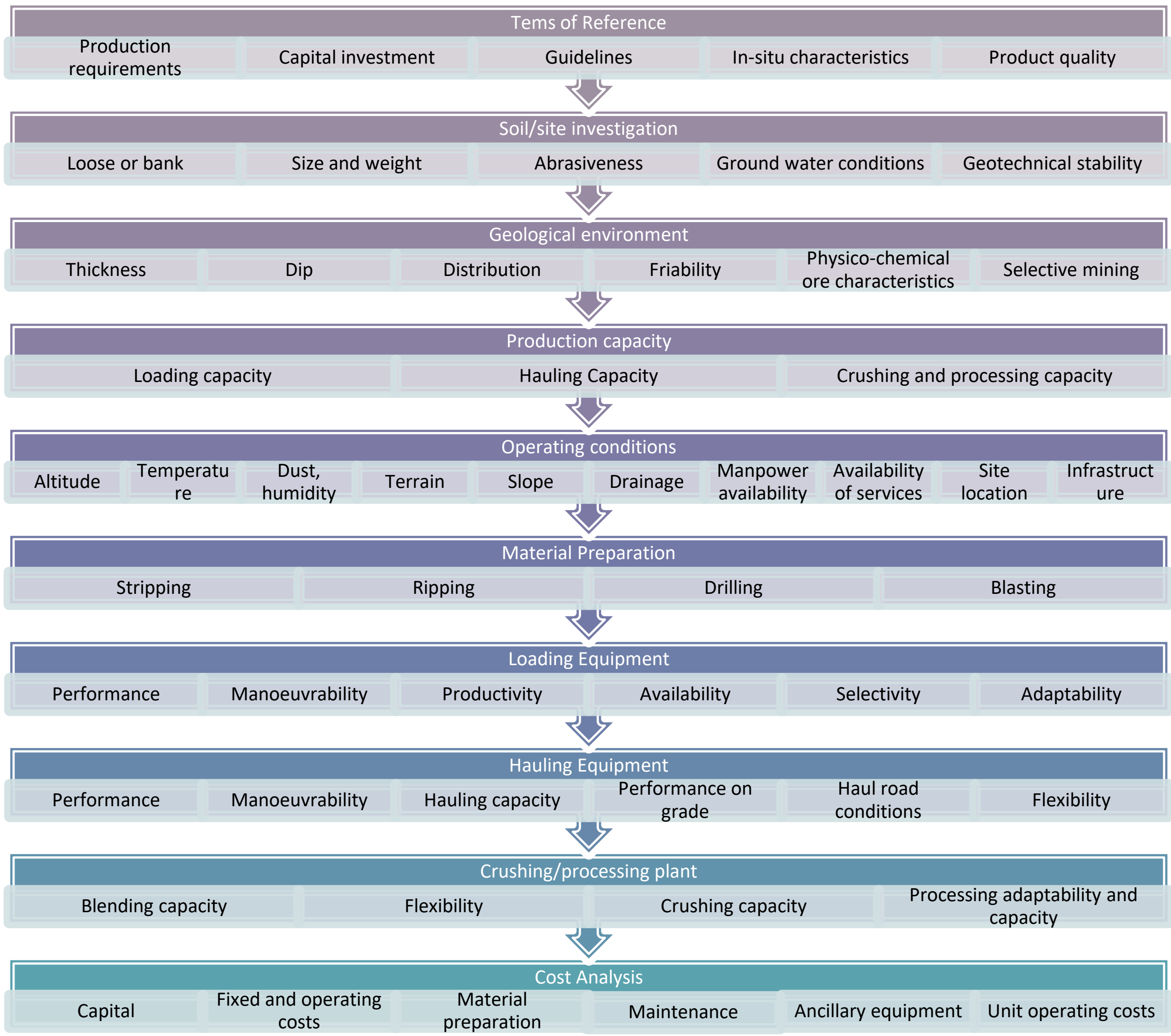


Figure 52 Minimum cost production impact factors

4.4 Chapter Summary

Quantification of impact of technological changes such as drill automation requires analysis of whole mining value chain in terms of impact on downstream processes and causal cycles. The parameters that are needed to build an SD model was discussed in this section. The next section introduces how to start building the SD simulation with the parameters listed in this section.

Traditionally the mining industry does not regard itself as being an operator of a system. It seems to regard itself as being the operator of equipment

Chapter 5

5 CONSEPTUALISATION OF THE SIMULATION MODEL

5.1 Introduction

The steps taken in system dynamics modelling of the mine value chain will be described in this chapter for meeting the objectives outlined in Chapter 1. The parameters that define the processes in the value chain will be explained with the help of an extended literature review and is based on Chapter 4.

The problem-solving process constitute a sequence of activities (Whelan, 1994) that is very similar to SD modelling as in Garcia and Garcia (2018),

- Defining the undesirable situation
- Define the system you wish to model
- Identify key variables
- Describe the behaviour of the key variables (understanding of the situation)
- Assumptions
- Identify stocks and flows in the system and map it out
- Define the flows
- Include quantitative information
- Run the model
- Evaluate the model
- Improve the model

All systems are comprised of components that contribute to a complete and connected system. Each component has its properties, and they cannot be predicted in isolation. In addition, there are many layers in any system as shown by Dominici et al (2018) in Figure 53. As discussed in section 4.3 mining is no different in terms of layers of systems in that environment.

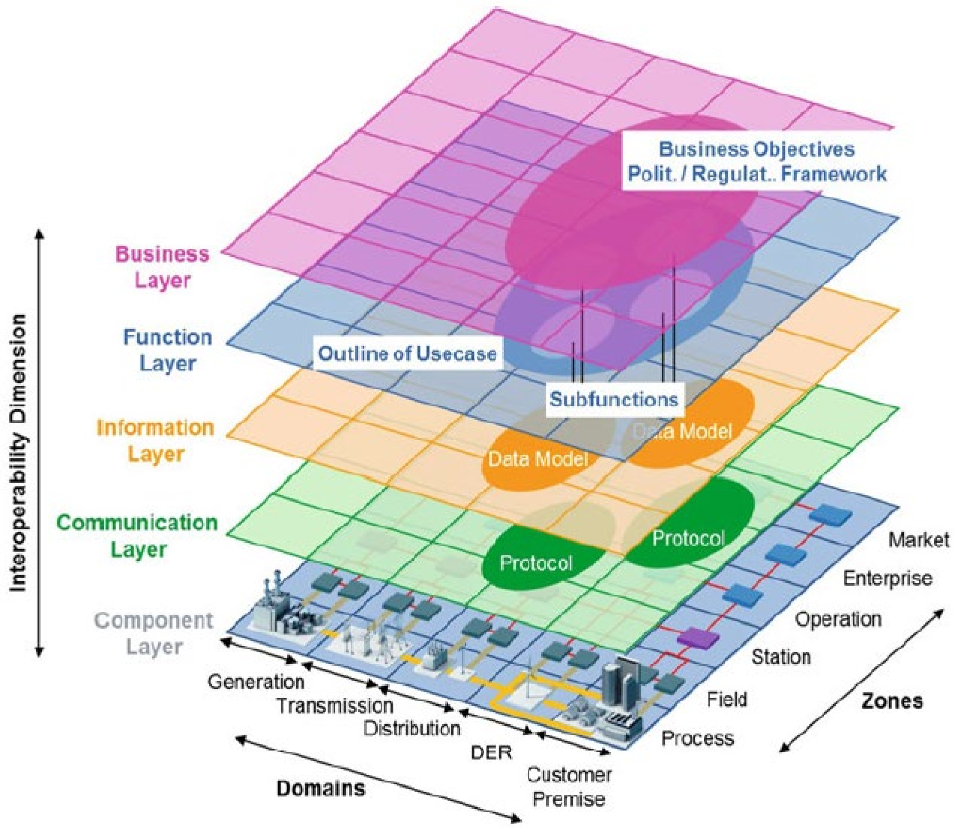


Figure 53 A multi layered conceptual model of smart grid (Dominici et al, 2018)

Simulation input will require the following components and layers for the new model being built in this thesis.

a) Mine resources scheduled converted to mining blocks with grade, size and geological characteristics
b) Simulation of drill and blast processes
c) Loading and Hauling system simulation
d) Crushing Plant/Mill simulation
e) Costing of all processes

Human and environmental factors will be omitted from the cost benefit estimations for this research. Instead, the focus on modelling is going to be on usefulness of the developed model; therefore, a modeller should have acquired some skills according to Whelan (1994) which are listed as:

- Generate a valid model of the system to be analysed and
- enter the model correctly into the computer.
- test the accuracy of the computer simulation

- Make the model clear so that others can understand its structure.

The process that will be followed in this thesis is as shown in Figure 54.

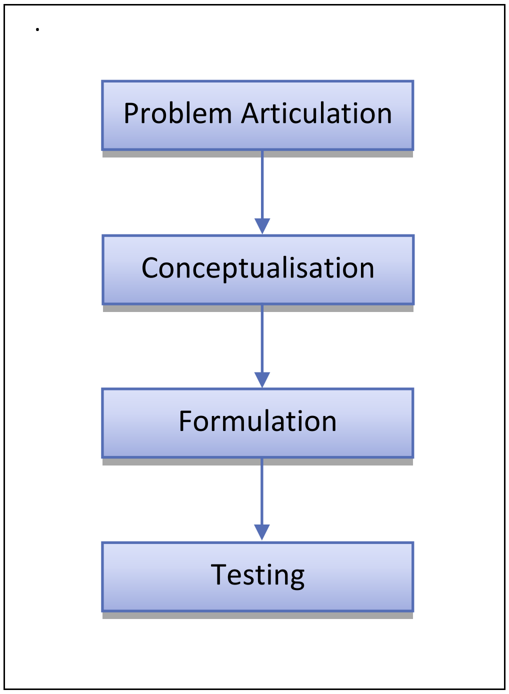


Figure 54 Modeling process followed

A summary of typical SD model creation steps taken by leaders in this field is provided in Table 13) (Wirsch, 2014).

Table 13 Creating an SD model (as listed in Wirsch, 2014)

Randers	Richardson (2002) and Pugh	Roberts at all (1983)	Wolstenholme (1990)	Sterman (2000)
Conceptualization	Problem definition	Problem definition	Diagram construction and analysis	Problem articulation
	System conceptualization	System conceptualization		Dynamic hypothesis
Formulation	Model formulation	Model representation	Simulation phase (Step 1)	Formulation
Testing	Analysis of model behaviour	Model behaviour		Testing
	Model evaluation	Model evaluation		
Implementation	Policy analysis	Policy analysis and model use	Simulation phase (step2)	Policy formulation and evaluation
	Model use			

5.2 Conceptualization

The objective of this research is to quantify the impact of change, i.e., determination of the value due to a new process or change of process/technology in an open pit mining context. This will require an integrated approach for a complete inclusion of major causalities and process dependencies of sub processes. Upon completion of the model, it is going to be modified to be able to capture downstream effects of the introduced changes to key parameters.

Conceptualizing firstly requires a clear purpose for the model, model boundaries and identification of key variables. Then the behaviour needs to be described or drawn with the reference modes of the key variables. The last step is drawing a diagram with basic mechanisms and the feedback loops of the system (Albin, 1997). The model will be unifying micro actions into a macro movement. The success of an entire process is disrupted due to one disharmonized action, and it is possible to demonstrate this behaviour in a simulated feedback loop of a system.

Once the conceptualization is completed the model is converted to feedback diagrams to level and rate equations. Thereafter, estimation or selection of parameter value is done. Finally testing the model and testing the dynamic hypothesis are required which also tests the model behaviour and sensitivity to perturbations (Albin, 1997).

Implementation is the last stage to test the model's response to different policies and translate study insights to an accessible form.

In this section conceptualization of the model for measuring the impact of drill automation is discussed. Ultimately this model should be designed to help mine management to make more informed decisions by understanding the system behaviour when certain changes are introduced.

There are two types of variables that define a model boundary:

- Endogenous – dynamic variables involved in feedback loops.
- Exogenous – components whose values are not directly affected by the system.

Exogenous components will usually be constant or time varying constants and not stocks and flows. Flows are changes in stocks and rates and are measured in units of the stock over time. It must be emphasized here that conceptual models are mental models and do not yet include formulation.

As the model being built mental models can change as there are more discoveries about the system. For example, for a modeller who is trying to generate knowledge about plausible causes and solutions there needs to be a historic reference. This is important since if the model does not behave like historical observations, it is an indication that model needs re-work (Albin, 1997). When there are no historic patterns then the modeller may hypothesize the behaviour such as exponential growth, exponential decay, overshoot and collapse, S-shaped growth, and damped, sustained and expanding oscillations. In mining there is a desired level of production required per year and all sub systems should support the expected outcome as seen in the figure below to

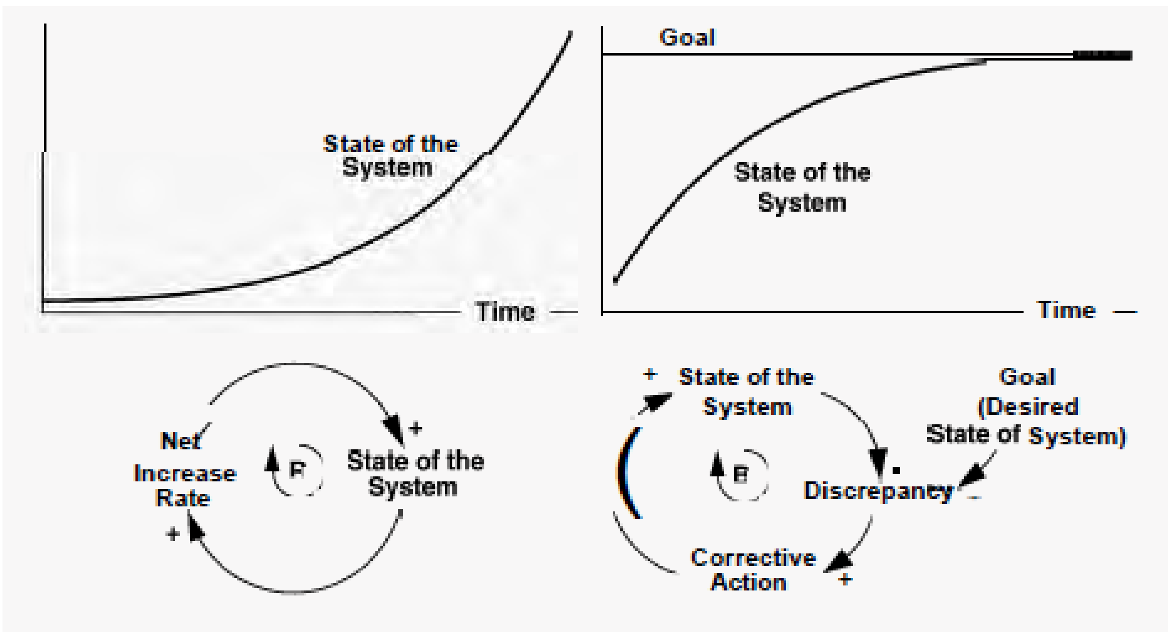


Figure 55 Loops defining states of the system ({Albin, 1997)

Once the modeller has a good enough mental model of cause-and-effect relationship then the system can be formalized into level and rate equations.

The state of the system mapped in Figure 56 gives an idea of how to model and eventually quantify the effect of technological changes in the overall system. The state of the system is however reactive to changes almost instantaneously. The modeller should have an idea or expectation of the outcome of the changes, otherwise it is difficult to demonstrate the proof of concept.

It is necessary to evaluate a given system by defining and accessing the measurement of performance, establishing total achievable performance, discussing alternative ways of improving performance and selecting the best policy for the improvements.

A mine must have some production goals and that there is a perceived state of the current drilling performance. The perceived gap due to the perceived drilling state is calculated as follows:

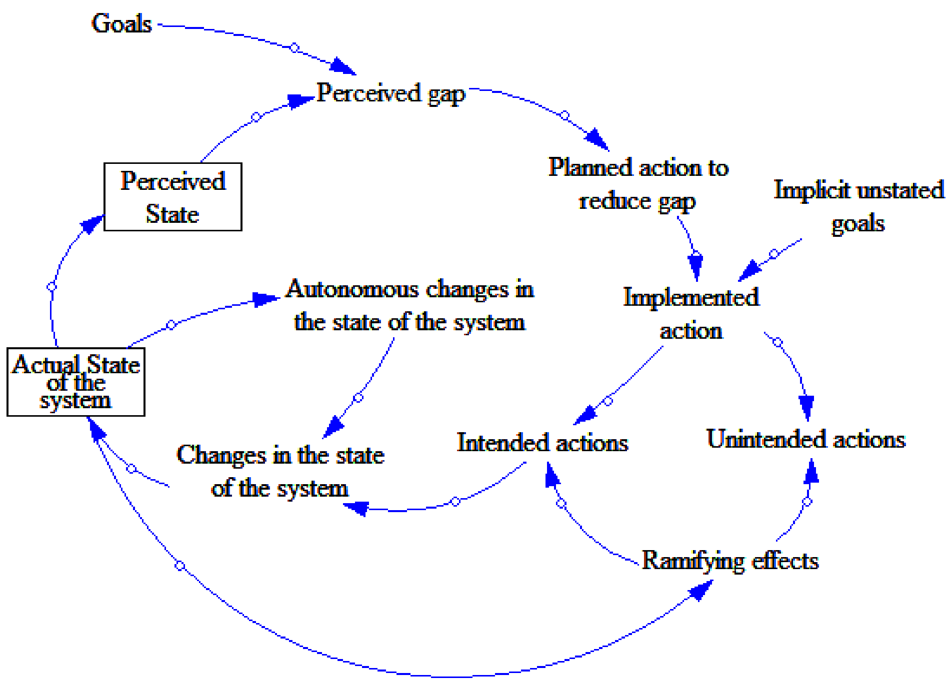


Figure 56 State of the system due to introduction of new technologies or changes in processes

Goal – Perceived State= Perceived gap **Equation 14**

Then, the mine decides to make some changes to the system and some actions are planned to reduce the perceived gap. A mining company may have some work carried out to determine the change the modern technology will create, i.e., an implicit unstated goal which could be increasing the gap and gaining some more drilling meters or improved quality. What this may be described as implicit unstated goal.

The way to calculate this would be as follows:

Planned Action to reduce gap– Implicit Unstated Goal = Implemented action **Equation 15**

The outcome is then both ways; with intended as well as unintended results. After deliberate changes, system is now causing further changes to the rest of the system sometimes creating more demand from the sub systems or capacities. Which may require further adaptation to the new state, almost autonomously the state will adapt to the unique environment (Figure 56) to meet the production demand.

This could be achieved by certain actions within the organization such as improving the unit process where the bottleneck is originating, improving quality, training of the operators, etc. Sometimes additional time and resources are used to close the gap. For example, drilling contractors may be brought to premises to close the gap in required drilled meters.

Conceptualization includes individual quantification for each process and changes caused by the introduction of recent technology should be reflected afterwards. The policies surrounding certain parameters that are most influential will be described in the case study. Processes are then expanded or consolidated based on the relationships assumed in the conceptual cause and effect relationship. In all processes, effects can be either one time or continuously increasing in any of the behaviours described above.

The work is sometimes rushed due to several reasons jeopardizing the quality. This may result in some rework generation. The concept of modelling of undiscovered work and rework generation is explained by Kefalas (1998), and he also shows the basic model of calculating the impact of the rework generated. Kefalas states that *Earned Value* is the most critical concept underlying the utility and integrity of performance measurement since it relates resource planning to schedules as well as to technical cost and schedule requirements. He adds that the management control system must yield data elements capable of providing the information necessary to measure performance on a product or project. The data elements are named as Budgeted Cost for Work Scheduled (BCWS) and the Actual Cost of Work Performed (ACWP). Analysis of these costs and schedule variance data enable a manager to pinpoint problems and quantify them in terms of budgeted cost and determine the reasons for deviation from the scheduled plans. The method he followed will be described below:

Before the drilling rigs were automated the work was carried out with some quality and rework generated at each WBS (Work Breakdown Structure). A demonstration of this is explained in the following simplified model:

Work to do: 25000 tasks

Work force: 80 people

Productivity: 10 tasks/person/week

Quality :1 (means 100 % quality)

Time to discover rework: 2 weeks

Time to reschedule: 4 weeks

Simulation step: 1 week

The model developed by Kefalas (1998) which is adapted for the drilling process is shown Figure 57.

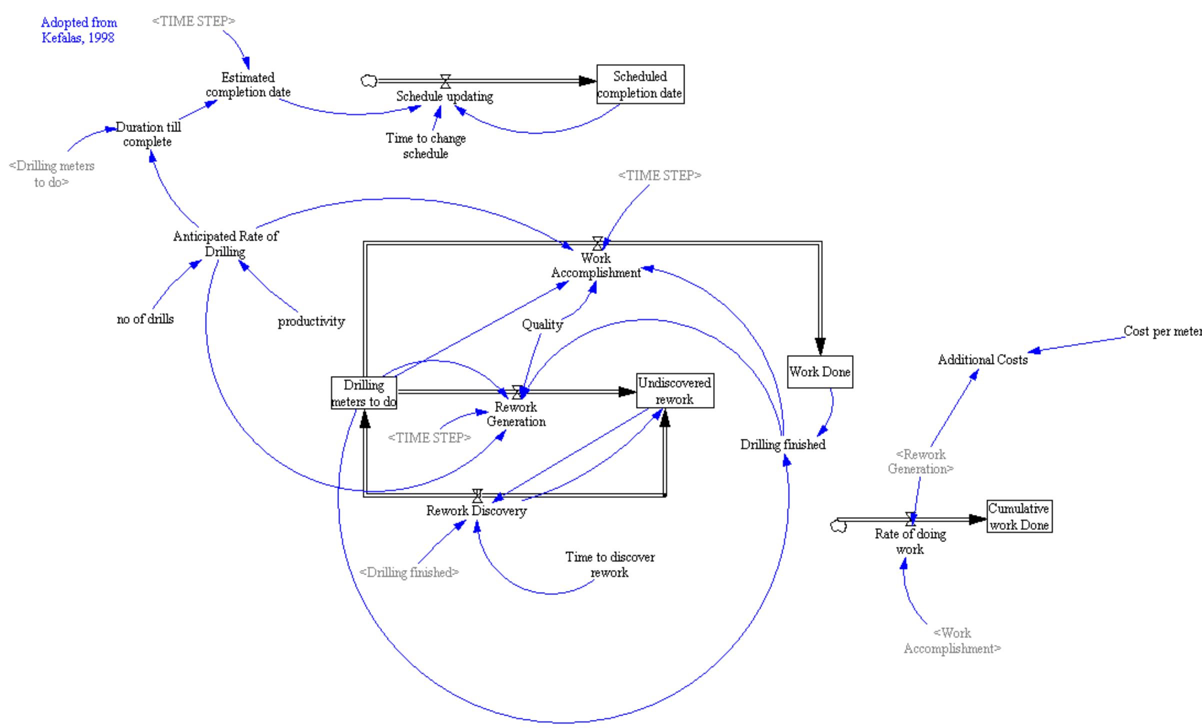


Figure 57 Work accomplishment model for drilling quality (adapted from Kefalas, 1998)

The model will require multiple mining blocks to be modelled with many attributes. Such as the tonnage, volume and work schedule as well as drillability and blastability which are a function of the material characteristics and production processes. Variability lookup tables or Excel type databases may be needed for each mining block for fragmentation and cost impact calculations. The sequence of the blocks to be mined are not continuous but discrete, however, it should be assumed it is continuous since this does not influence the outcomes of an indicative model. The way to manage this in VENSIM is firstly to understand that Vensim can be used to model nonlinearity based on lookups. For example, instead of modelling fragmentation we may use graph inputs such as, fragmentation curves that are based on actual measurements, examples of performance criteria for various production machines, etc. For example, effect of fragmentation on loading performance of the excavators or cycle time studies as lookup variables.

5.2.1 Initial Model Construction in Vensim

Conceptualizing and creating the model for estimating the drilled meters depends on the drilling rate. Drill rate in return depends on many other factors as discussed previously. The feedback rate frequency defines the graph behaviour; therefore, it needs to be set in the beginning. If we run this model over a 24-month period but want feedback every week, then TIME STEP needs to be set at 0.25.

Vensim' s LEVEL type of inputs is work remaining and work accomplished (Figure 58

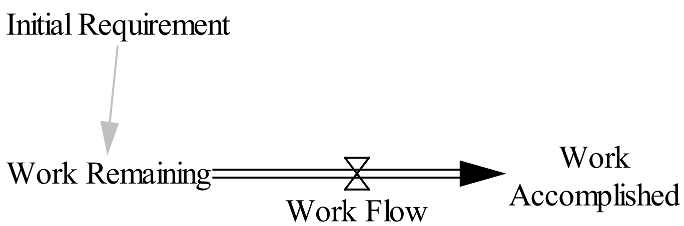


Figure 58 Simplified work flow rate

The units will be set as meters (Figure 59)

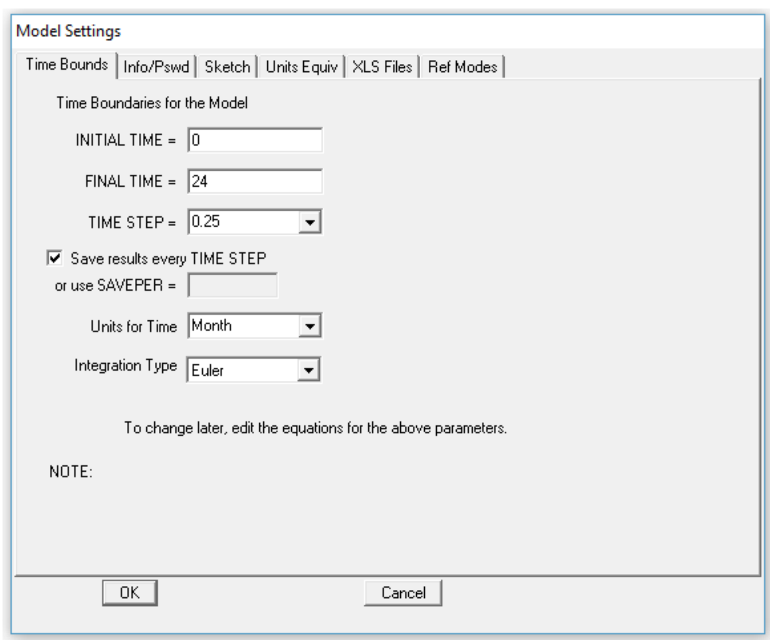


Figure 59 Initial model settings

Initial drill meter requirements are set at 1000 meters and workflow, or rate is set as 100.

The next step is to see when the work will be completed.

Therefore, another node is added to the model which is *work complete* (Figure 60)

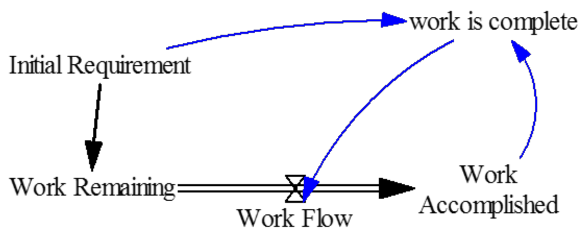


Figure 60 Additional node added to initial workflow

So far, the model code looks like as follows:

(03) INITIAL TIME = 0
Units: Month
The initial time for the simulation.

(04) SAVEPER =
TIME STEP
Units: Month [0,?]
The frequency with which output is stored.
(05) TIME STEP = 0.25
Units: Month [0,?]
The time step for the simulation.
(06) Work Accomplished= INTEG (Work Flow, 0)
Units: **undefined**
(07) Work Flow=
IF THEN ELSE (work is complete, 0, 100)
Units: m/Month
(08) work is complete=IF THEN ELSE (Work Accomplished>=Initial Requirement, 1, 0)
Units: **undefined**
(09) Work Remaining= INTEG (-Work Flow, Initial Requirement)
Units: m

However, on plan this might be true sometimes the mine may achieve more than 100% or less than the required before the finishing time. Quality of the work conducted may slow down the drilling tempo and unintended stoppages of work may decrease the tempo. Re-drilling of certain blastholes may be required to ascertain the blasting quality. Therefore the next step is to introduce undiscovered work such as re-drill requirements.

The equations are changed each time a new variable is added to the conceptual model. The model code now is as follows:

FINAL TIME = 24
Units: Month
The final time for the simulation.
Fraction complete=
Work Accomplished/Initial Requirement
*Units: **undefined***
Initial Requirement=
1000
Units: m
INITIAL TIME = 0
Units: Month
The initial time for the simulation.
Rework discovery rate=
Undiscovered rework/Time to detect errors
*Units: **undefined***
SAVEPER =
TIME STEP
Units: Month [0,?]
The frequency with which output is stored.
TIME STEP = 0.25
Units: Month [0,?]
The time step for the simulation.
Time to detect error lookup((0,5),(0.5,3),(1,0.5))
Units: Month
Time to detect errors=
Time to detect error lookup(Fraction complete)
*Units: **undefined***
Undiscovered ```1rework= INTEG (Work Flow(1-Work quality)-Rework discovery rate,0)*

Units: m

Work Accomplished= INTEG (Work Flow, 0)

*Units: **undefined***

Work Flow=IF THEN ELSE (work is complete, 0, 100)

Units: m/Month

work is complete=

IF THEN ELSE(Fraction complete>=1 ,1, 0)

*Units: **undefined***

Work quality= 0.9

*Units: **undefined***

Work Remaining= INTEG (Rework discovery rate-Work Flow, Initial Requirement)

Units: m

Resulting behaviour of the conceptual stock and flow model above is shown in Figure 61:

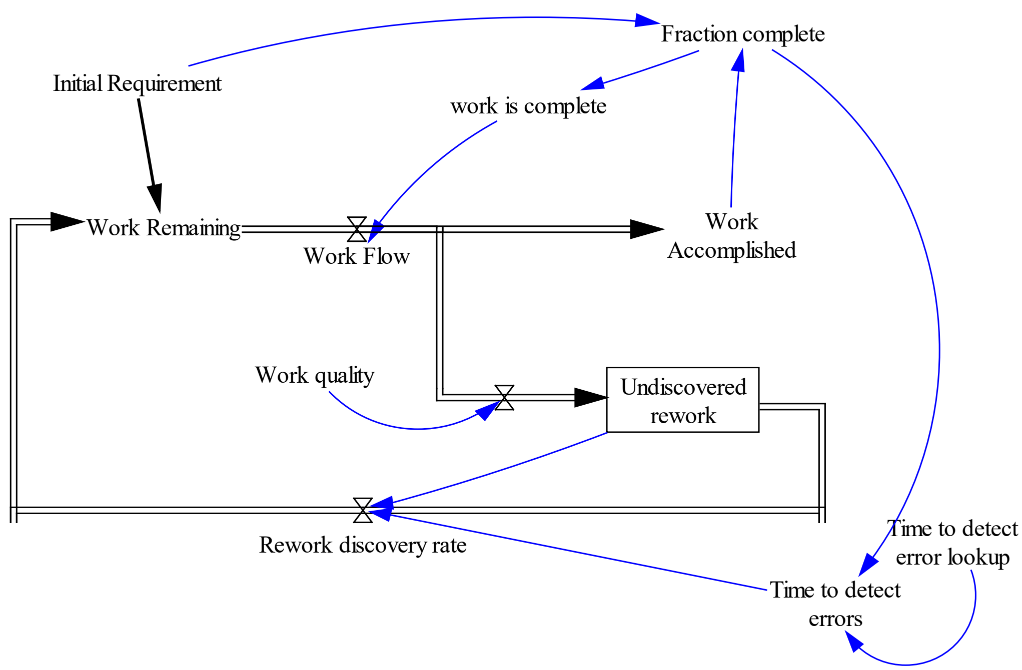


Figure 61 Deveoping the model for determining effect of quality and undiscovered rework (adapted from Sterman, 2000)

The example discussed in Figure 61 is used to study the behaviour of the inferior drilling quality in the form of redrilled blastholes.

5.2.2 Approaches to Solve Complexity in Mining

Claassen at all (2012) state that simplicity in a mechanistic world is created by dividing the whole into parts giving equal attention to each part making sure of the capacities, measures and targets are manageable and controllable. Another approach is to consider the whole system with the thinking process focusing on the importance of the output. In this method, key leverage points and the capacity constraint resources are identified for a balanced flow throughout the system. This approach seems to be a better solution. Comparison of mechanistic and systemic approaches are listed in Table 14.

Table 14 Summary characteristic of a mechanistic and systemic approach (Claassen, 2012)

MECHANISTIC APPROACH	SYSTEMIC APPROACH
Departmental /disciplinary focus	System based
Task driven	Throughput driven
Balancing capacities – align and integrate according to capacities	Balancing flow-align and integrate according to flow needs
Cost world	Flow world
Benchmarking	Focus on system's unique attributes
Average norms and standards	Condition driven standards
Applied to simple environments with little variability and Dependencies/interdependencies	Applied to complex value chains with numerous dependencies/ interdependencies

A summary of difficulties faced in modelling the mining environment is listed by Claassen, which are:

1. A systemic approach is required to design, operate, manage and improve mining value chains otherwise improvement efforts will result in local optimization and the balancing of capacities. This will disrupt the system, and which makes it unpredictable
2. Synchronization, alignment and integration of capacities in the mining value chain requires a mechanistic approach that however increases complexity.
3. Management, improvement, and operations methodologies in mining must consider the variability in ore to simplify complex geological environments and to optimise ore utilisation and system performance. If not considered, then more variability and dependencies will destabilize the system and make it less predictable.
4. A mining methodology definition may enable the identification of constraints, dependencies, and inter-dependencies in the mining value chain. The definition should include the management and control rules, processes and activities based on flow principles.

5.3 Analysis of Unit Mining Processes in VENSIM Modelling Environment

This section describes the process followed to create the new SD mining model in Vensim in a logical and sequential manner by showing incrementally how each part of the model links to the other. Some initial simulation results are also presented for illustration purposes and confidence building in the model. A mine process map will guide how the process flows in a typical large surface mine and how KPI's link the tasks. Each unit level will need to be modelled separately by building all the relationships. This is to make sure that at the smallest detail the model will behave *as expected*. Individual portions can be tested independently for integrity of the model and then added to the bigger model.

Typically, a mine depends very much on the mine resource characteristics. A mine resource is often covered with waste rock surrounding it. Then the ore body itself may not be optimally mineable block by block due to high and low-grade areas. The speed at which the waste cover is removed determines the speed at which the valuable orebody will be uncovered. Therefore Waste/Ore ratio as well as cut off value of the mineable grade is initially determined by the mine planners. It also determines the sequence at which each mining block will be mined. Stripping ratio may change from a small to a larger value as the mine goes deeper. There is normally an overall stripping ratio. A constant waste stripping ratio is to be used in the model not to further complicate the model. Each mining block will be going through the processes as described in Figure 62.

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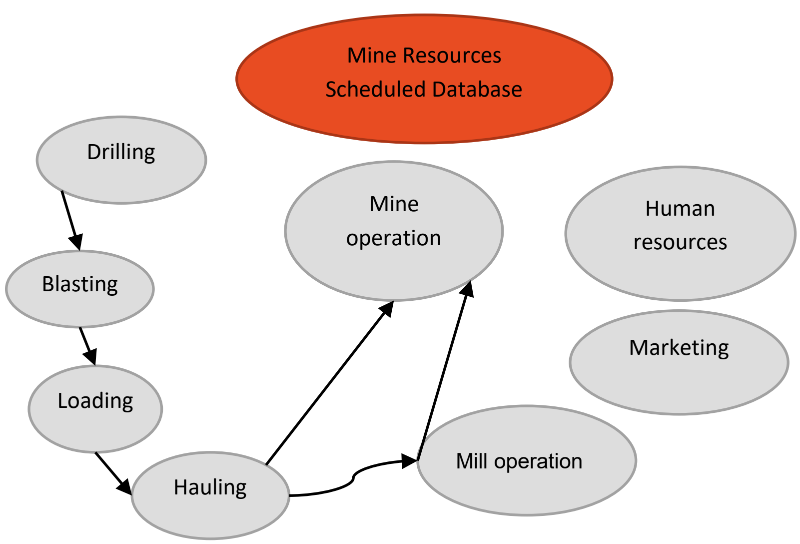


Figure 62 Integrated mine framework for simulation.

A typical mine has long-term medium-term and short-term goals which are based on mine resources. The life of mine is known but may extend. The resources are treated as the existing stock. The stock can be increased with the mines intention of increased production based on resource model that can be converted to mineable reserves based on grades with high confidence levels. This can be added as growth per year in the model (Henderson and Turek, 2013)

There is a certain delay between the sub processes in a mine and follows a sequence that is cyclic. A simplified mine model with those unit processes can be seen in

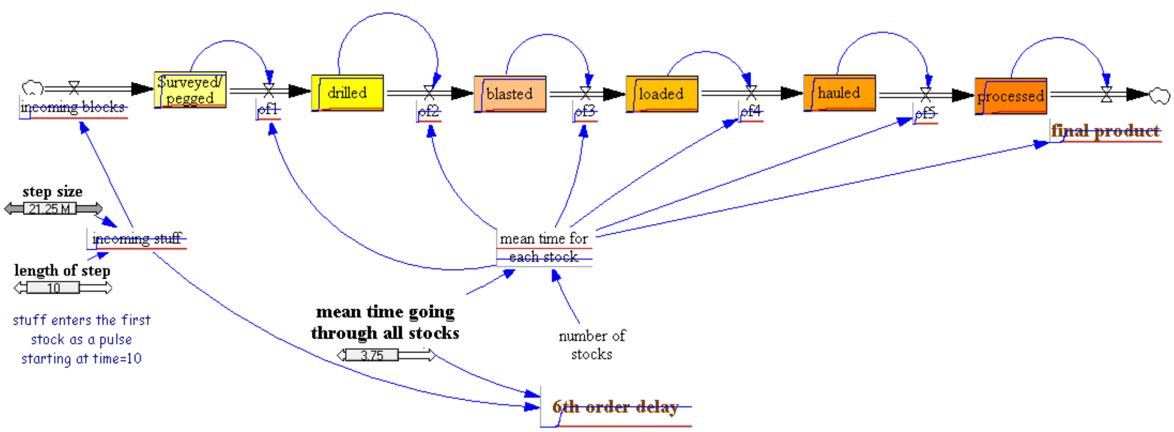


Figure 63. This means drilling must happen before blasting and blasting must happen before loading and hauling. In this way new production benches are created for next round of drilling-blasting-loading-hauling. The model can be constructed as either 4th order delay or a series of four processes with fixed delay or variable rates based on capacity and rates.

In an ideal world the model is simple in terms of time delays between unit processes as seen in the following model created in VENSIM (

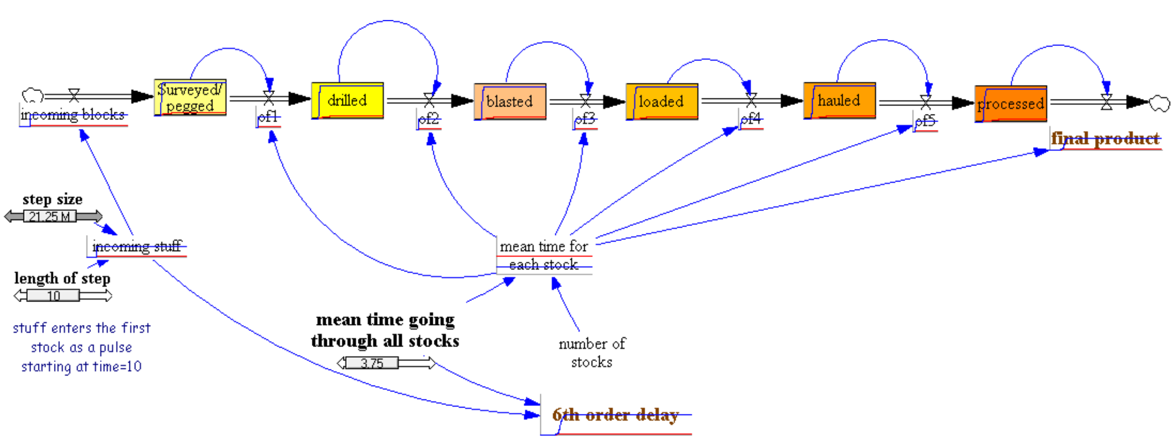


Figure 63). in an ideal world there is a fixed cost at each level and a fixed price for unit production.

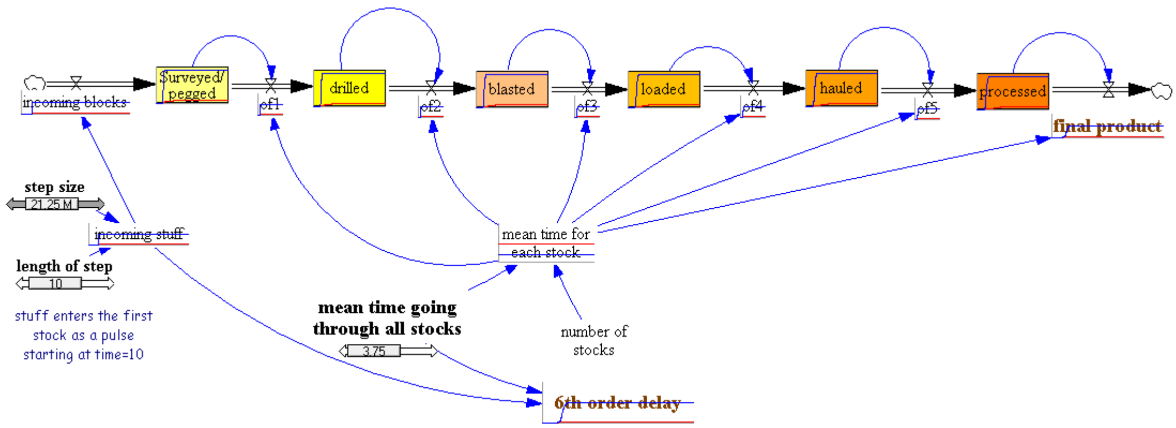


Figure 63 Ideal mining cycle with fixed delays between processes.

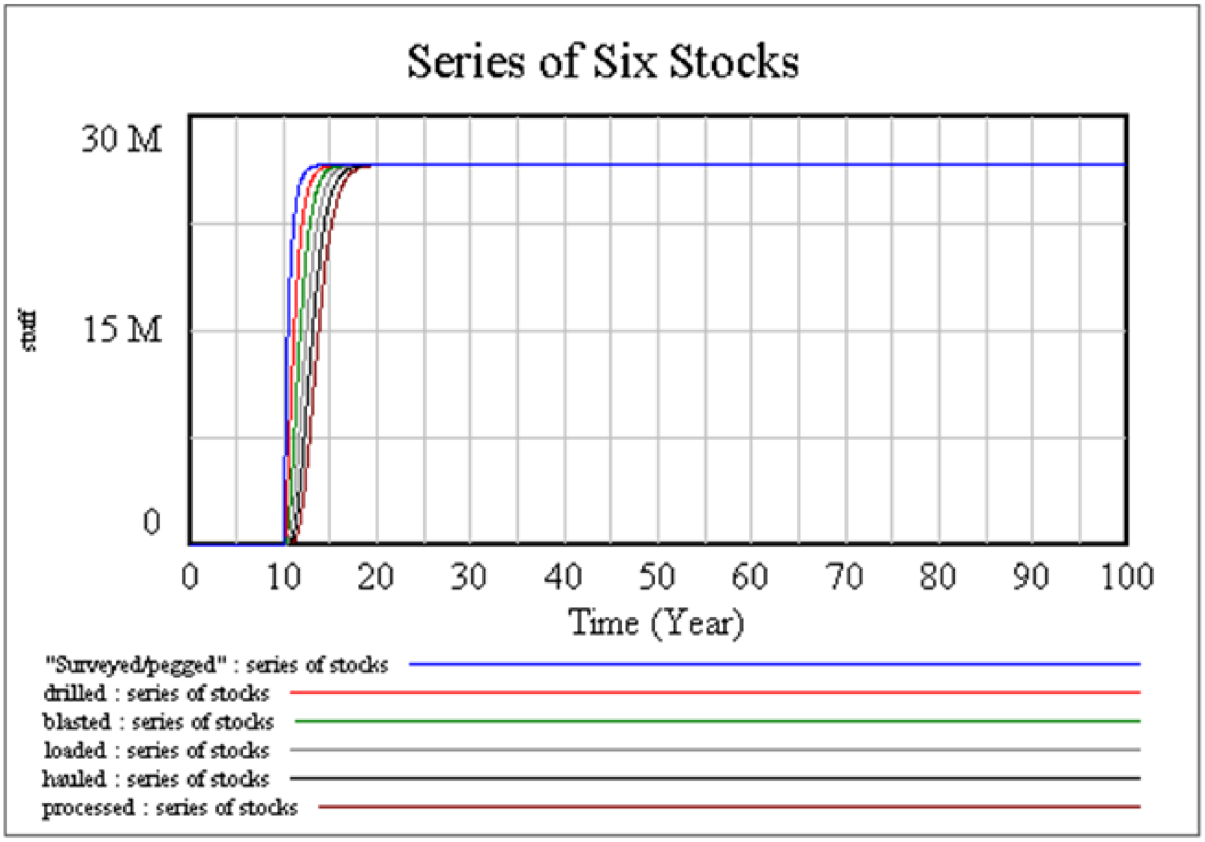


Figure 64 Fixed delay processes with production output per time (year)

In an ideal world the delays will be only for the initial weeks and thereafter due to continuous production it will not matter anymore as seen in Figure 64 which is indicating some result as a run conducted for the model in

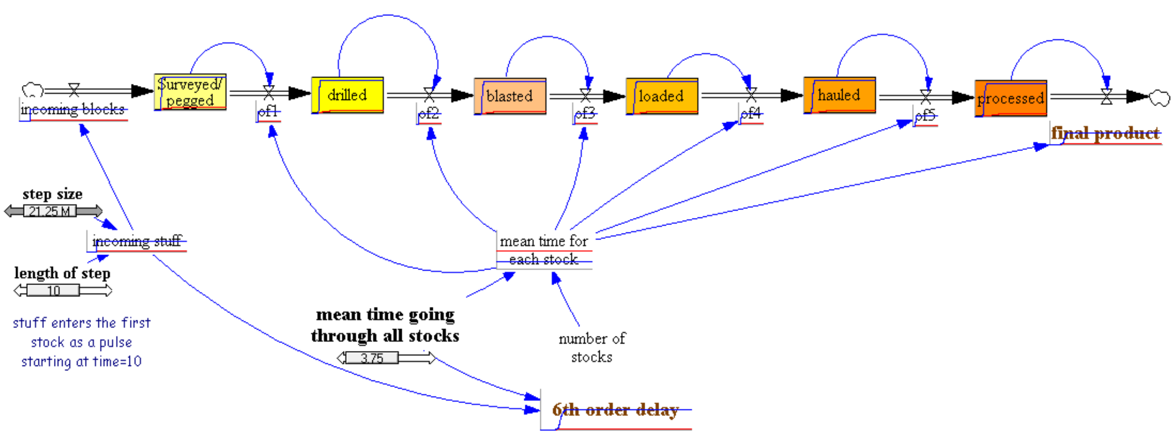


Figure 63. In this graph the delay between each process is exaggerated by the author to make it visible in the graph for demonstration of the concept. It is a day to a week of delay between each process not a few years as the graph suggests.

The first complication in mining is that production is not continuous but discrete and based on number of blocks drilled-blasted-loaded-hauled. Each block once drilled needs to be handed over to blasting personnel. In between there are various quality checks and a handover process. If the blasting team does not approve the quality of drilled holes, they may request re-drill of certain holes, etc. The quality of the blastholes is assessed if collapsed, drilled short or too long or deviated from X-Y location (see discussion in 5.5). There is also angular deviation due to wrong levelling of the drill rig. It needs to be emphasized that once drilled in wrong location some holes will not be re-drilled due to costs and time delays or it is impractical.

If the drilled block has a certain percentage of quality achieved and acceptable by the blasting team, the block is blasted. This means there must be a quality indicator for drilled holes per blasting block. The same applies to loading and hauling. Once blasting is conducted, the blasting could be a success or a failure and this is often noticed during loading process. The loading team only receive a blasting block to be loaded after some quality checks and site prep is conducted such as tidying up loose rocks with dozers, etc. The condition of the muckpile can result in difficult loading conditions that is too high, too low or scattered. If there are “toes” (high grounds between drilled holes not effectively blasted), the excavator may not be able to handle those areas. “Toes” sometimes need to be blasted to make the loading process smoother. In addition, there may exist large boulders that need to be handled separately, i.e., put aside to be blasted later or a hydraulic pecker is used to further fragment the boulders. Sometimes the number of boulders is counted to measure the effectiveness of the blast.

The biggest game changer in mining is variability. A mine does not operate like a factory with fixed boundaries and predictable machinery performances. A simple SD model is now built to demonstrate how variability is a result of simple delays or capacity imbalance in a mining environment.

Assume there is a single mining block that will be staked by a surveyor, then drilled, then blasted and loaded to the trucks to be dumped either to crusher or stockpiles, and waste sent to waste dumps.

The first process in the mining cycle is staking of the blastholes and modelling of this process will be demonstrated for only one blasting block. Resources and block processing time can be estimated. It is a straightforward process; however, it will be demonstrated that variability makes it look complex. The block to be surveyed is queued in the system which is a “stock” in VENSIM terms. The block arrival rate depends on production scheduling and planning of the blocks. This basic model assumes that every day one block will be released to be staked by a surveying team that can finish surveying the block in one day. For simplicity capacity and task are kept at “one” and variability introduced by doubling either the task or the capacity. Sometimes processing time is doubled. The combination of the three, that is processing time, capacity and task are the three important building blocks of a producing system. The reason for choosing simple numbers here is to demonstrate how systems behave too complex due to imbalances or delays resulting in see-saw effect even with one or two production blocks. In addition, it will be used for demonstration of how VENSIM will be used to build a more detailed model from simple steps towards the complex ones.

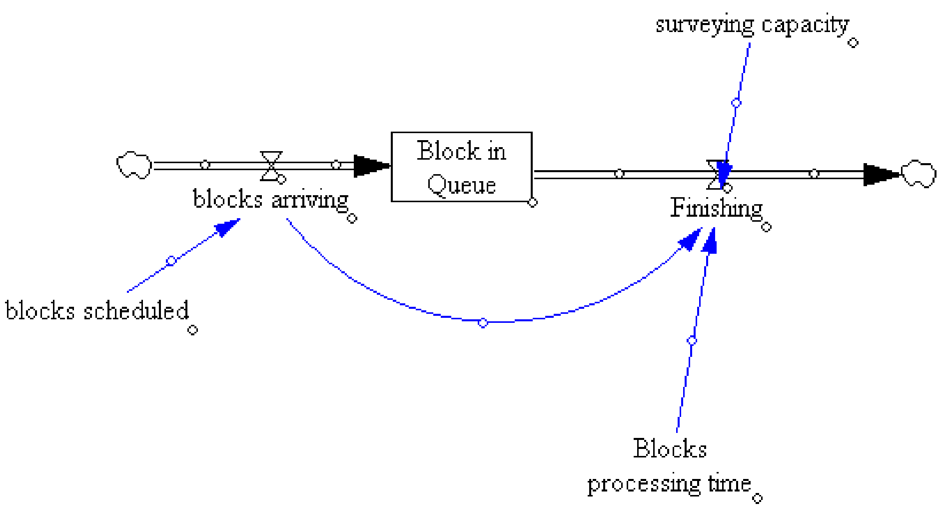


Figure 65 Simple model built to demonstrate importance of a balanced resource, capacity and processing time

This model uses DELAY BATCH function and set as below for determination of finishing rate of the staking process (Figure 65):

DELAY BATCH (blocks arriving, surveying capacity, Blocks processing time, 0, 0, 0)

A series of graphs will be generated for various input variations for which the combinations are listed in the Table 15 for this simple process in order to create variability.

Table 15 Schedule-capacity-processing time scenarios for simulations in Figure 66 - Figure 70.

Figure Reference	Blocks Arriving	Surveying Capacity	Block processing time
Figure 66	1 per day	1 per day	1 per day
Figure 67	2 per day	1 per day	1 per day
Figure 68	1 per day	2 per day	1 per day

Figure 69	1 per day	2 per day	2 per day
Figure 70	1 per day	1 per day	2 per day

The model is tested with various scenarios as listed in Table 15. The graphs are generated by simulating the model in "syntesim" mode entering the scenario variables. Graphs can be seen next to each scenario model. The model time period is set to 14 days to be able to see the work accomplishing story in detail. The objective is not to have an accumulation of blocks in the queue (green slope). It is acceptable to have some variation in the stock but on average it should be stable.

The reader may ask what the significance of this little demonstration is. In mining, variability is one of the issues that result in congestions, unstable stocks increasing – meaning delay in the mining block builds up and stress on the staff who is behind the schedule. Each process needs to be as stable as possible (levelled) since all the processes are waiting on the previous process.

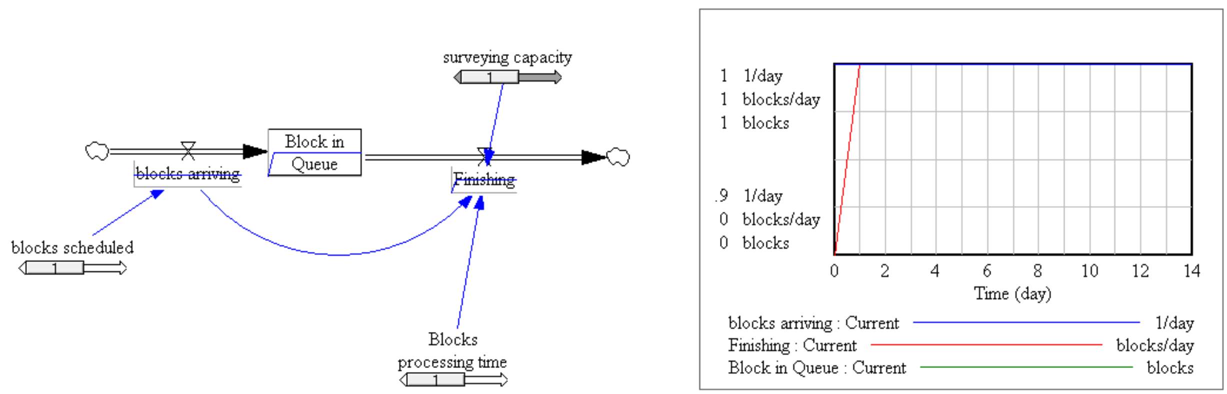


Figure 66 Schedule-capacity-processing time Scenario 1

In Figure 66 the model is stable and shows a blanced steady state flow with capacity and processing time matching the scheduled flow.

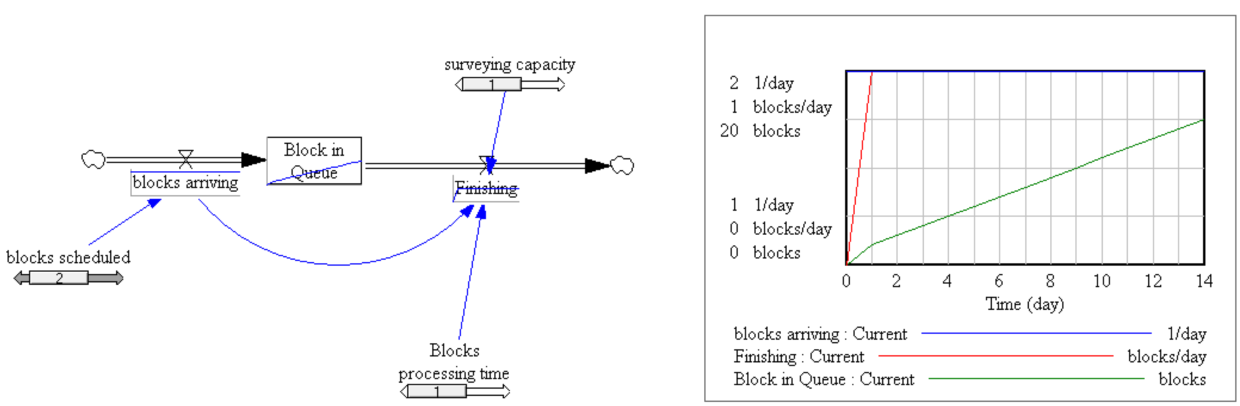


Figure 67 Schedule-capacity-processing time Scenario 2

It can be seen that when work requirement is doubled and capacity and processing time stays fixed then this causes a steady increase in the work lined up or queued. The output is as previously planned.

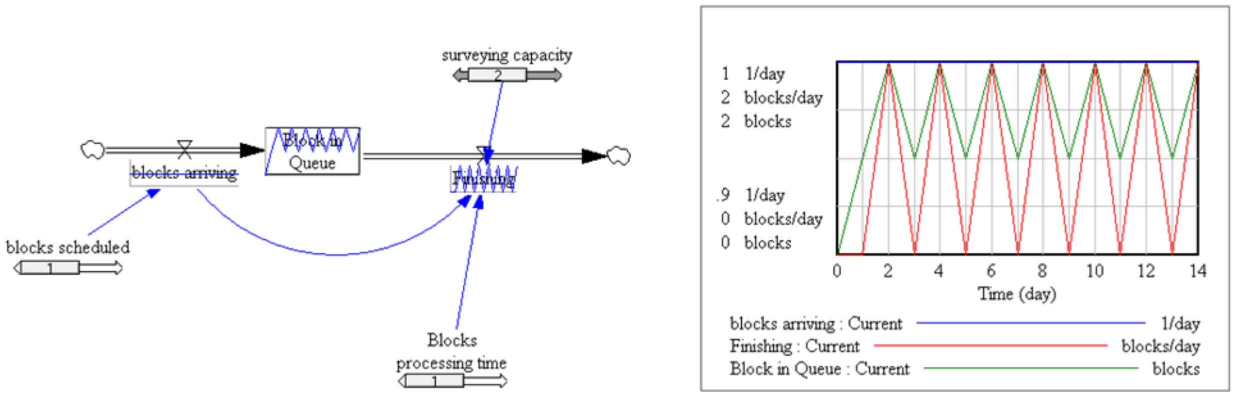


Figure 68 Schedule-capacity-processing time Scenario 3

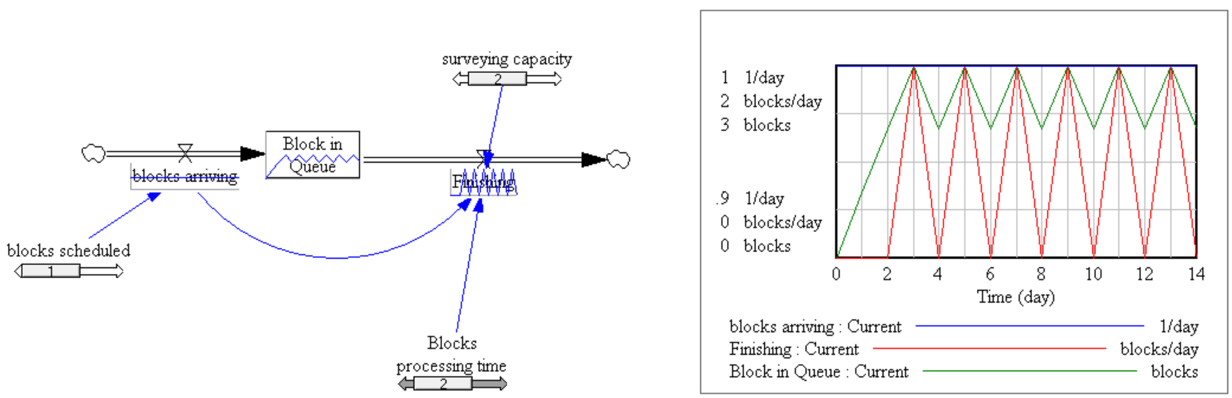


Figure 69 Schedule-capacity-processing time Scenario 4

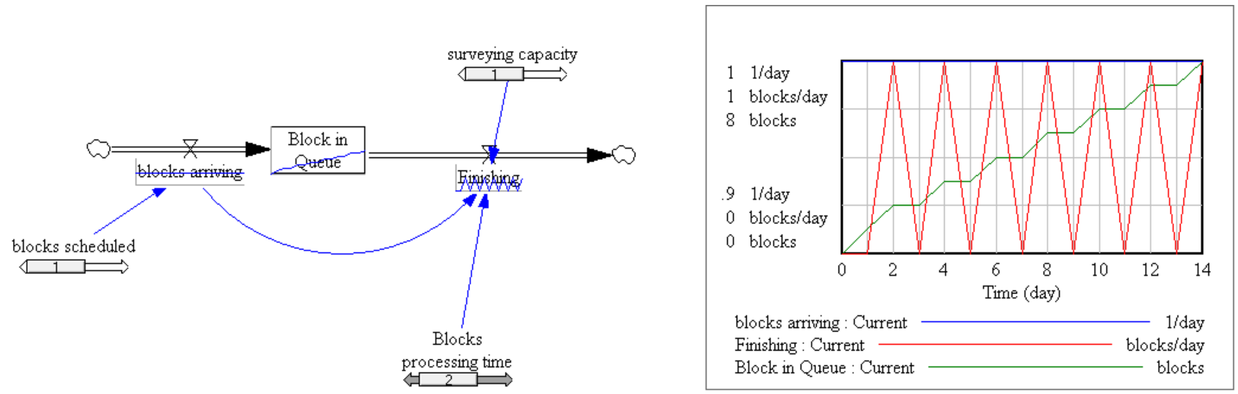


Figure 70 Schedule-capacity-processing time Scenario 5

In Figure 68 work to do lined up stays constant but the capacity to do the work is doubled this results in teams working somedays half days some days full day and some days no work done. The situation may be called as over-resourced. Meaning workforce gets same salaries for less work performed.

In Figure 69 the scenario now changes to less work assigned to the team but processing time doubles together with capacity. Processing time may increase due to many reasons and will not be discussed here further. The two teams now finish work every second day, The input and output will be kept at 1 block per day. The system is stable but there is variability in queued work flow but on average the output is one block per day.

In Scenario 5 only block processing time is doubled (Figure 70). This again causes variability in the work finished and causes list of work-to-do increasing steady state. This setup is useful to analyse delays in production due to delays in processing time, such as cycle times.

The equation box developed for this variable as seen in VENSIM is shown in Figure 71.

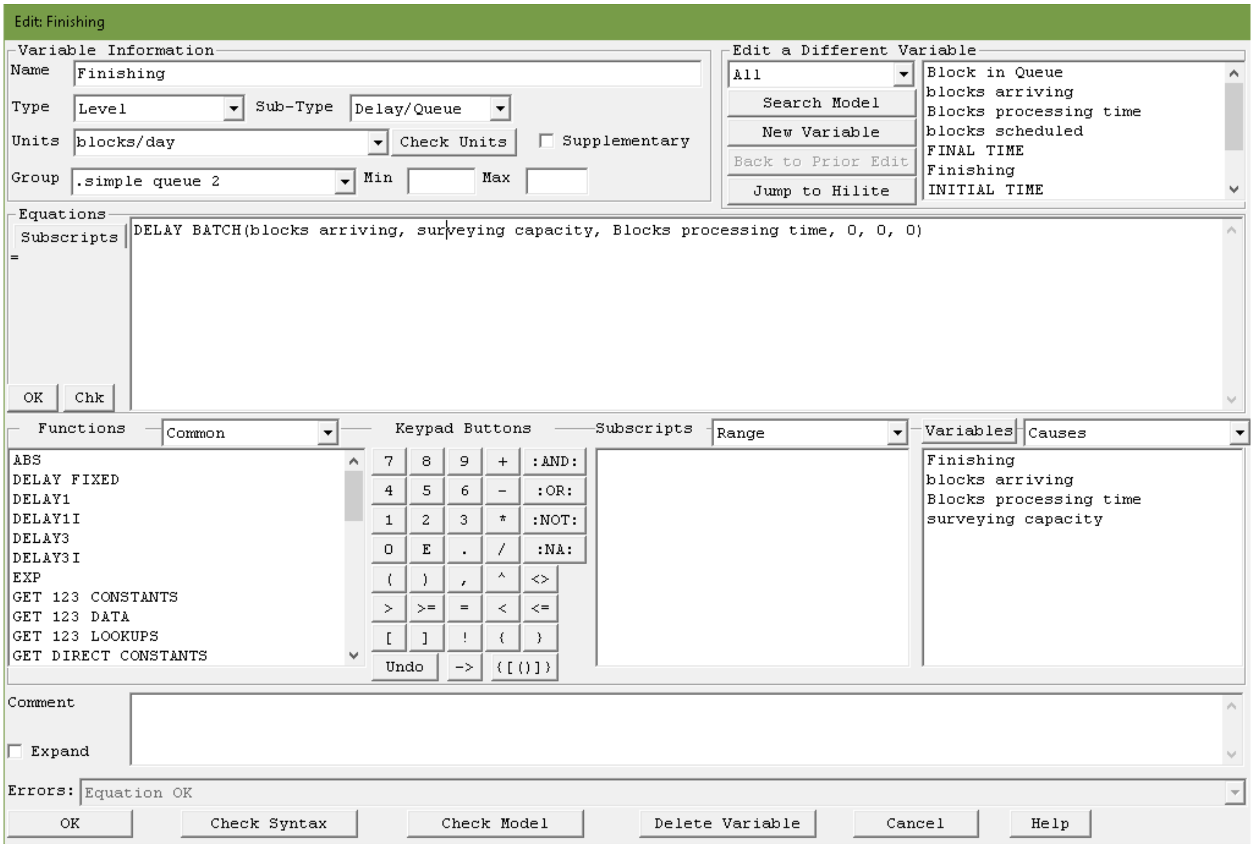


Figure 71 Equation box developed for the variable called "Finishing"

This illustration helps to understand the significance of correct planning scheduling and resource planning. If the tasks allocated to the resource that they are designed for are correct, then system is mature and stable. If a process starts behaving out of the norm in only one step the rest of the processes will feel it in the form of variations or fluctuations.

Once the resources are correctly allocated to the tasks the modelling is then taken one step further by modifying the base model to suit the expectations of the organization, i.e., planned output and resource allocation for each cyclic mining unit process.

It needs to be further emphasized that automation may reduce the variability to some degree by having a predictable and measurable process. If automation of drilling process is adopted, then there is no need for manual surveying process in the mining cycle.

Often the main bottleneck in a mine operation is drilling. If the system becomes unbalanced at any one point in the cycle due to a changing environment, then additional resources need to be deployed to make the system reach steady state. For example, most mines will use contractor drilling, loading or hauling to close the gap due to imbalances in the resources due to variability.

The same simulation pattern needs to be followed for the remaining processes in the mining cycle. This time combined flow is created with the same setup as the surveying unit process. The main processes included are surveying, drilling, blasting, loading and hauling.

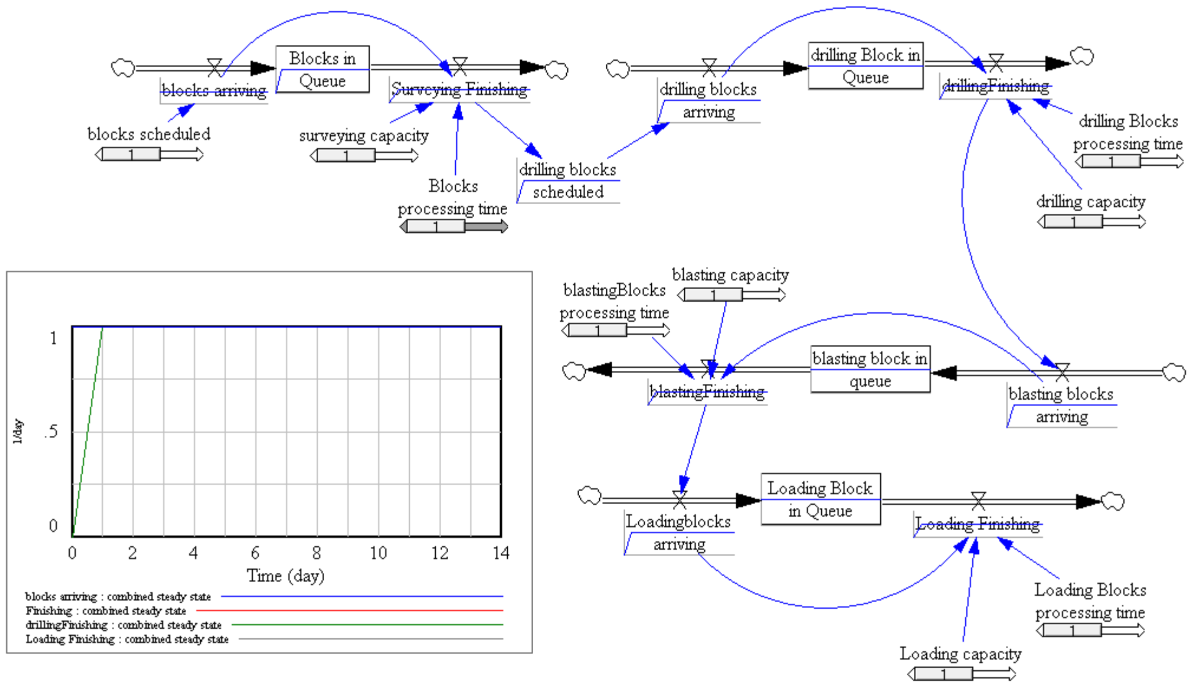


Figure 72 Mining cycle base model with 1 block scheduled per day

The model in Figure 72 is now further tested by " blocks scheduled" and "capacity" doubled, and an additional bottleneck is created by increasing the processing time for drilling. The resulting simulation graph can be seen in Figure 73 for this scenario.

The outcome of this simulation is that there will be a delay in the loading process by the end of the 14-day schedule. However, if the time is extended further all blocks will be processed eventually. The issue here is that customer will have delayed product. The next step is to look at a process to maintain the required output per unit process.

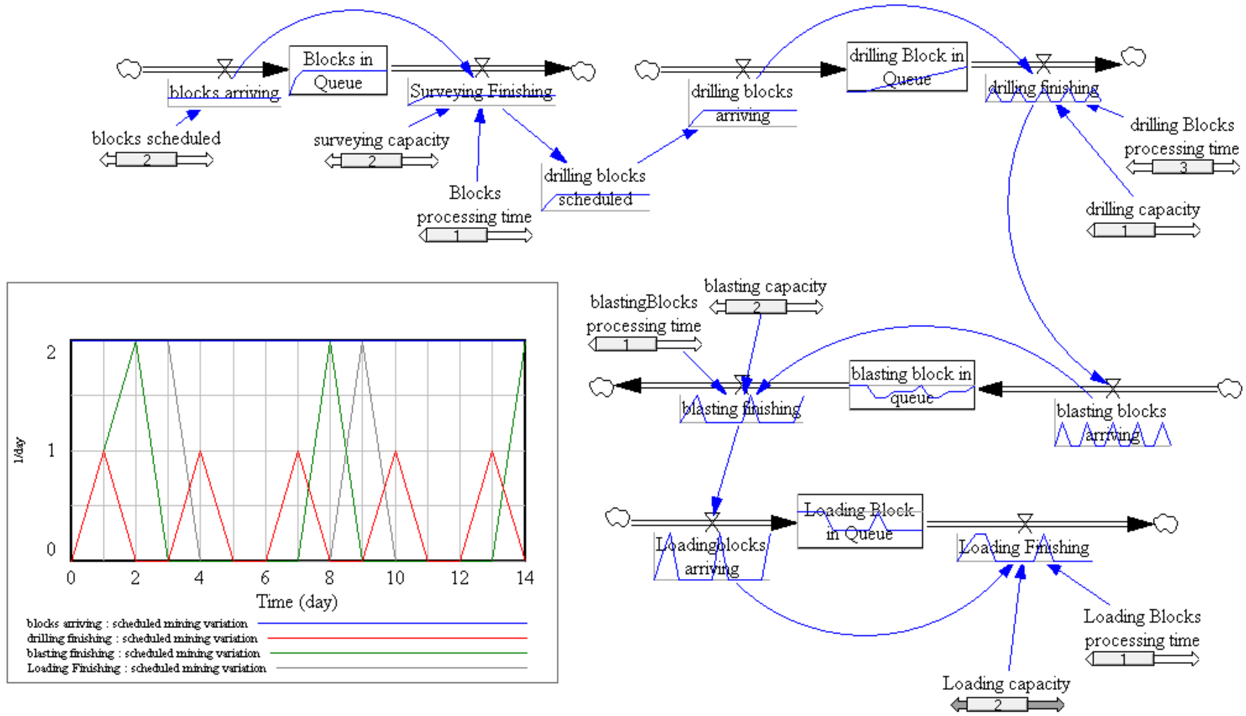


Figure 73 Mining cycle with drilling bottleneck scenario due to increased drill processing time.

There is one limitation that needs to be built into the basic model which is blasting related delay. Loading and hauling as well as personnel need to be removed from site during blasting times. The equipment needs to be walked away from the vicinity of the blasting and moved back on the bench once all safe clearance is given. This period is non-productive. This delay depends on weather conditions, and effectiveness of the blasting team. If blasting times are known, then a delay can be introduced right after the blasting is taking place.

Now that basic mining cycle is modelled, the next step is to introduce mine planning. A mine must meet a certain production demand. Mining blocks need to be scheduled so that production demand is met. In the meantime, the waste needs to be mined out as well. This will be discussed in the short-term planning and scheduling section.

5.3.1 Short Term Planning and Scheduling

Mine planning is a complex process; complexity of mine planning is visible in the process map as in Figure 74.The basis of a typical planning include the processes as shown in Figure 74. A schedule of five weeks of activities is the basis of the short-term planning. This schedule is reviewed on a weekly basis. It is assumed that majority of mines are considered successful if they can achieve 85% of the planned activities on time.

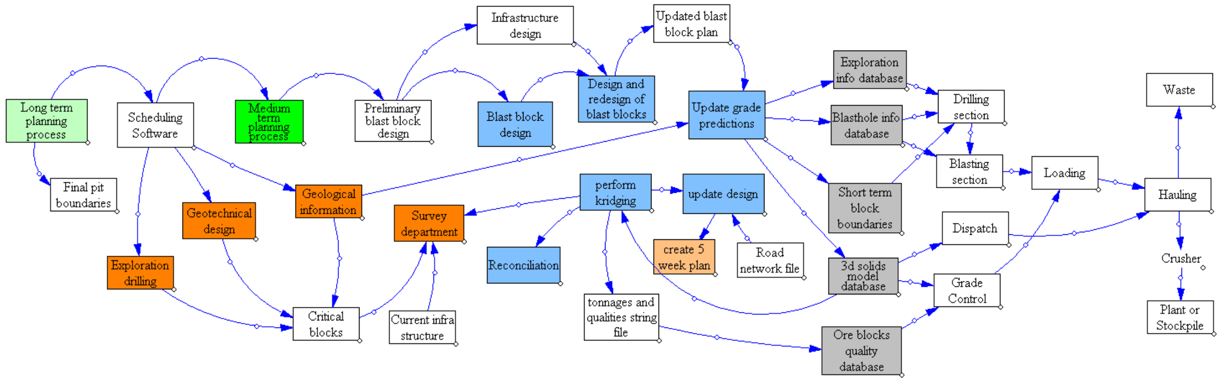


Figure 74 Mine Planning process (own source)

Any mine plan should pave the way to targets defined by management and business model of the mine. Compromises are often made to reach the target due to limitations and bottlenecks in the system and this may lead to errors and lack of quality. The short-term plan model will be established based on the Kolomela mine production targets as reported in their yearend reports openly available. It is a two-way iteration in an operating mine where long term, medium term and short-term production goals are often adjusted based on changing conditions and demand for production. A long term production goal of a mine is based on available proven reserves. Medium term plans are generated based on expected production demand per year. Short term plans are generated to meet the objective in the medium plan and converts it into task and resource-based scheduling of the daily mining activities to meet the target production. Creating a monthly schedule of activities to mine the mining blocks based on the available resources such as drill rigs, loaders and haulers is normally done with the help of scheduling software that is linked to databases such as grades and qualities of mining blocks as well as their xyz coordinates to check if the blocks are available spatially. Some blocks may not be available due to limited accessibility or are being locked due to other mining blocks that are still to be mined. Therefore, following a sequence when mining the mining blocks is important to unlock the other value bearing blocks, in the simplest terms overburden needs to be mined first before reaching value bearing ore blocks.

The parameters involved in planning that can be quantified can be listed as below based on the parameters listed in Kennedy (1990, pg. 398):

- Annual tons of ore and waste combined t
- Annual legal holidays of total shutdown, days
- Scheduled operating days per week, days/week
- Annual scheduled operating days, days/year

- Scheduled shifts per day, shifts/day
- ADT = Average daily tonnage, t/day
- Peek delivery to dumping points, ADT ÷OJE
- OJE= Overall Job Efficiency (45 min, hour), t
- Average mechanical availability of scheduled time, %
- Annual outage factor, %

Rate of overburden mining is dependent on the amount to be mined which depends on the stripping ratio. The overburden planning model is shown in Figure 75.

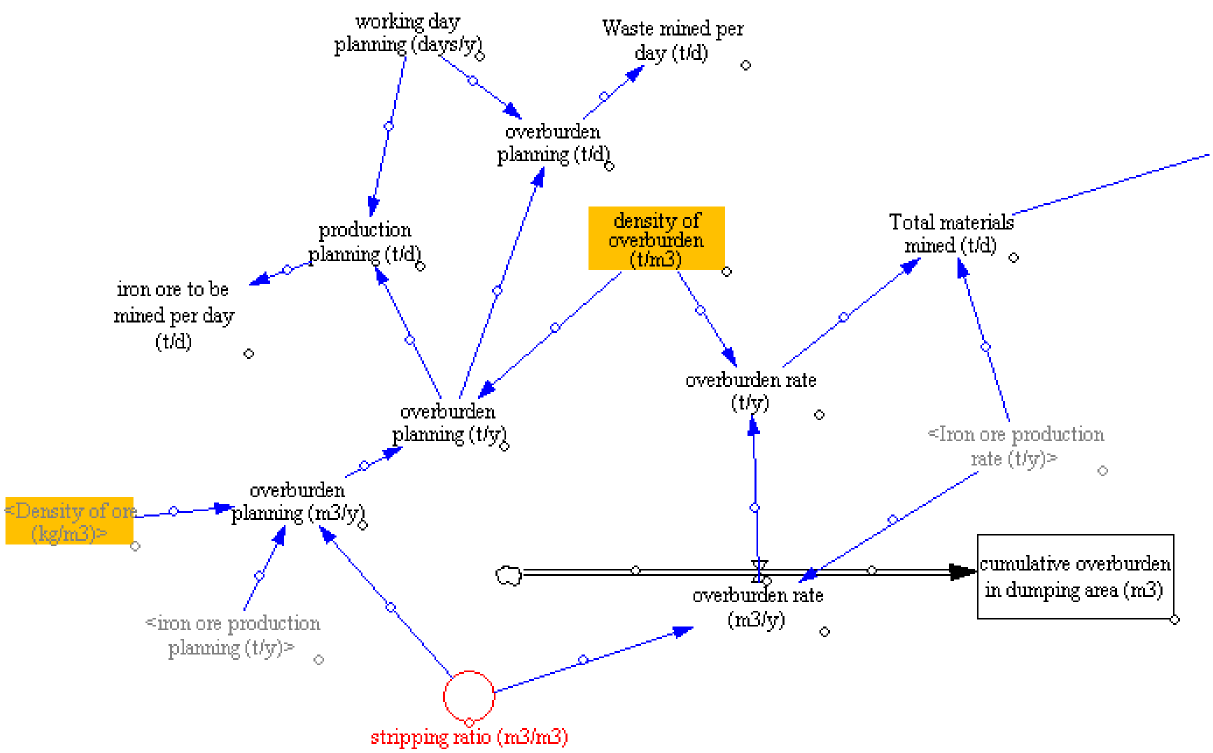


Figure 75 Overburden planning dependent on stripping ratio

The screen capture of the part the model as shown in Figure 76 is ore mining rate calculation based on planned production This part of the model was adapted from Sontamino (2014) with the changes introduced regarding type of commodity, costing model details and stripping ratio, was adapted to suit the purpose of this thesis as seen in Figure 112.

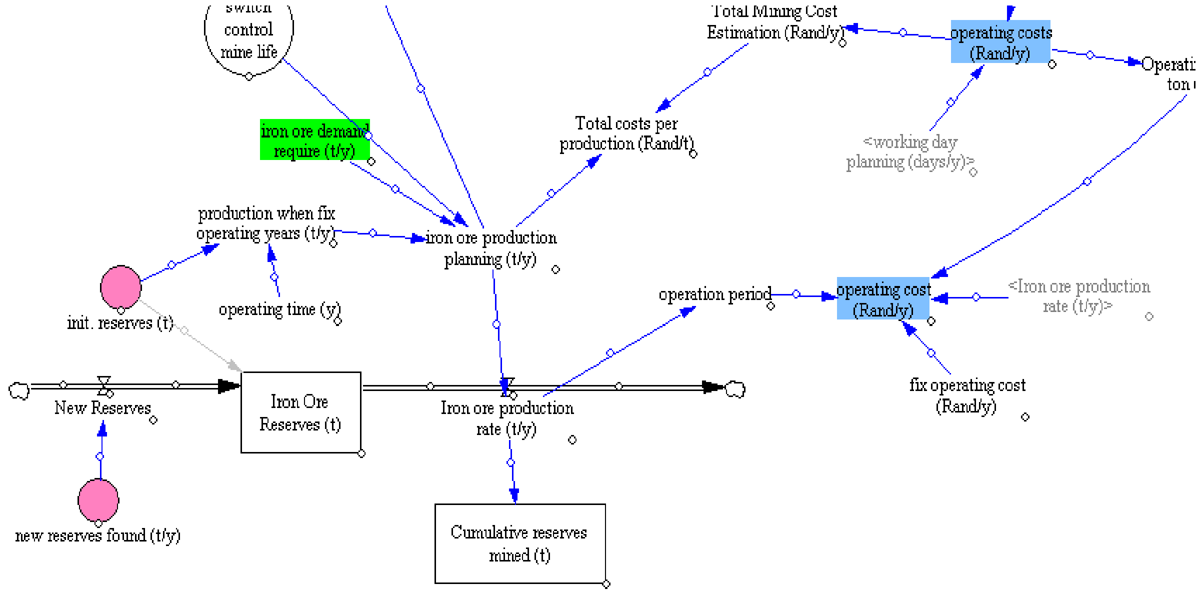


Figure 76 Production scheduling for ore based on annual ore demand (adapted from Sontamino, 2014)

The planning process needs to be dealt with in the simulation model for costs to be simulated based on production. Completeness of the short-term plan depends on effective communication of what is happening on the production floor.

5.3.2 Modelling the Production Schedule

Drilling is accomplished based on a schedule therefore schedule needs to be modelled that adjusts resources.

Drilling rate variability which depends on density, UCS and blasthole diameter in different ground types are important variables that are directly related to drilling efficiency.

The following statements are typical of surface mining drilling:

Rule 1: As the drill diameter increases the drill tempo decreases

Rule 2: As the ground becomes harder the drill tempo decreases

Rule 3: As the ground elasticity increases the rock becomes tougher therefore drill rate decreases

Rule 4: As the ground becomes blocky the drill rate will be slower and the drill steel might bend leading to bit stuck, or drill deviation occurs.

When processes do not achieve targets planned, the impact can be nonlinear due to the dependencies; therefore, the sub model presented in Figure 77 is constructed as a conceptual stock and flow diagram for quantifying the impact of each process. It is an example of a nonlinear relationship that may exist in a cyclic environment.

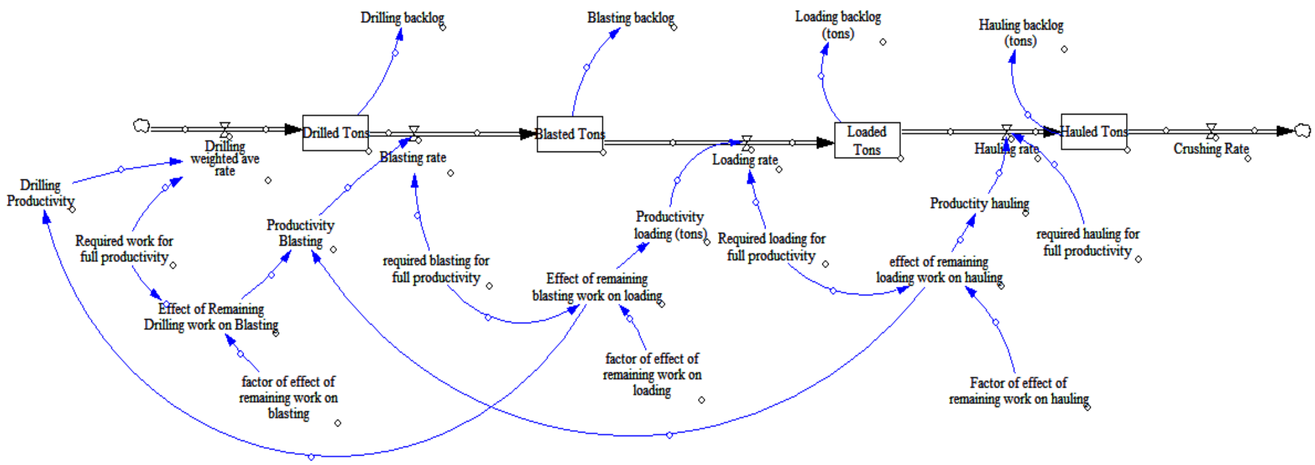


Figure 77 Backlog conceptual model (adapted from Sterman, 2002)

Table 16 Effect of schedule pressure formulae (Sterman, 2002)

The average delivery delay	**the ratio of the backlog to the completion rate**	**Delivery Delay=Backlog/Task completion rate**
		Backlog= INTEGRAL(Task arrival rate-Task completion Rate, Backlogt0) Task completion rate=Min(maximum completion rate, Potential Completion Rate) Maximum Completion rate=Backlog/Minimum Delivery Delay Potential Completion Rate=Net labour*Workweek/Time per task Workweek=Standard Work week*Effect of Schedule Pressure on Workweek
Effect of Schedule Pressure on Workweek	a function of Schedule Pressure	Desired Completion Rate/Standard Completion Rate
The desired completion rate	determined by the backlog and the goal of the organization	Desired Completion Rate=Backlog/Target Delivery Delay
The standard completion rate	represents the throughput the mine typically could achieve	Standard Completion rate=(Net Resource*Standard Workweek/Standard Time per task)*(1-Standard Error Fraction)
Desired Completion rate **Standard completion rate based on resource (labor)**		Desired Completion Rate=Backlog/Target Delivery Delay Standard Completion Rate= Net Labour*Standard Workweek/Standard Time per Task

		Time per task=Standard Time per task*Effect of Schedule pressure on Time per task
Effect of Schedule pressure on Time per Task is a function of Schedule Pressure		

Tasks accumulate in a backlog until they are processed and delivered. However, completion rate, resource limitations and for other operational reasons doing overtime work may not be a solution to catch up for required production due to resource limitations or shortage of time. The factor of effect of remaining work for each sub process is dependent on a specific mine environment.

The following conceptual cause and effect relationship (Figure 78) constructed in VENSIM is applicable to an open pit mine where common inefficiencies are included. Each item in the cause-and-effect relationship is a small system within a larger system that needs to be modelled separately and patched to each other with common parameters, such as cost per ton, capacity, productivity and time.

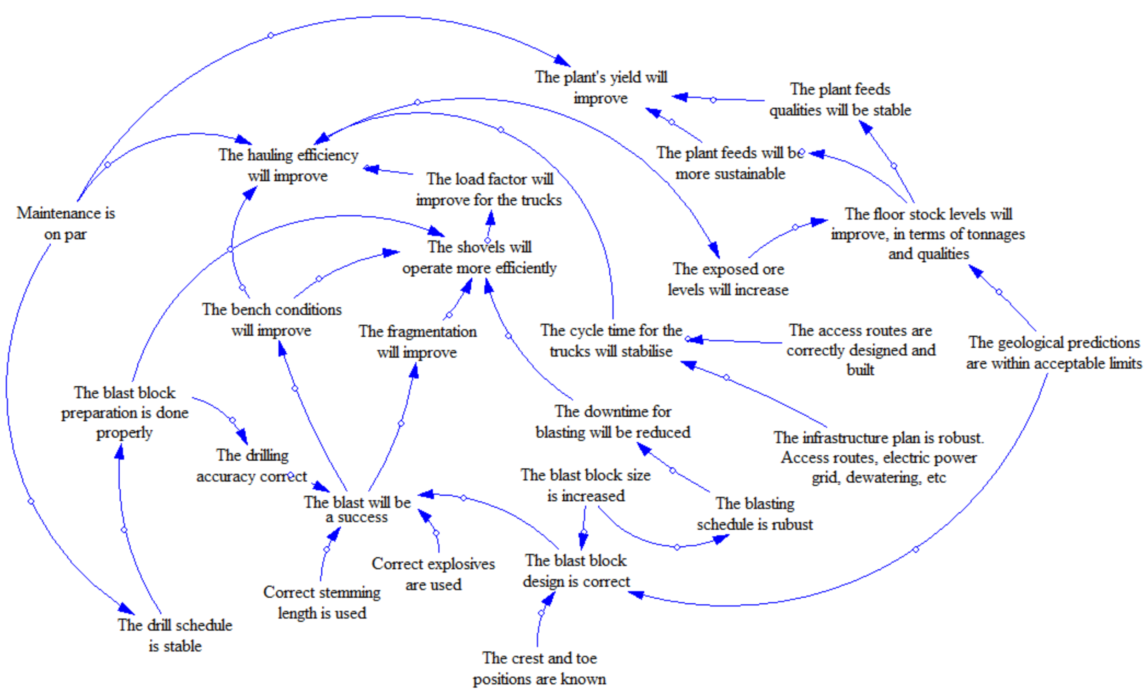


Figure 78 Cause and effect relationship at a surface mine with key impact areas

The most important feature in this diagram is the aspect of stability as variability in a mine is not easy to adjust regarding crushing plant settings. The quality of the material delivered for example if consistent (low variability) will lead to consistent plant settings eventually, therefore consistent qualities achieved at the plant. Variability can be handled better if the reaction times are shorter.

But in the mining environment information feedback is often very delayed and the reaction times are often slow. Therefore, it needs to be managed at the source of the problem, i.e., variability needs to be eliminated.

5.4 Mining Processes and Parameters for Modelling

Modelling will involve using as many inputs as possible and create a base. The variables will then be changed in a simulated mode to see how much it changes from the baseline. Therefore, the baseline should be as realistic as possible.

The model can have multiple layers, such as financial, operational cause and effect relationships, cost input and revenue output. One must remember that there is also information delay at any organization which can also be part of a larger simulated process. This topic is beyond the scope of this study.

Cost estimation for hydraulic shovels and truck fleet will depend on tons per day production requirements. It also depends on hauling distances. The operational costs of hydraulic shovel trucks fleet include 2 cost sections. They include labour operating costs and equipment operating costs. They should be calculated based on the daily production rate. The operating costs should be determined for different haulage lengths in the pit.

It is imperative to remember that the business looks messy with data and what seems to be unrelated may be connected in the real-world dynamics. Therefore, this model should be able to:

- Empower the user by providing a business analysis that can express data in a form that will match business owner’s concepts in their minds.
- Simplify the complexities where possible by abstracting away from tabular relationships.
- Where there is big data sometimes simple sums, rollups and counts are not enough to support multi-level and multi-fact questions that need to be answered.

The following are typical steps at SD modelling based on the system dynamics modelling examples reviewed in the literature.

- Determine the input parameters that are not influenced by any other parameters that will be used in the model and that their value can be changed freely by the user for sensitivity analysis.

- Establish relationships that are dependent on exogenous parameters as well as system generated parameters.
- Create the architecture of causal loop diagrams that the outcome to determine the value qualitatively or quantitatively.
- Analyse the case study with created model and determine the key inputs and outputs for the case study.
- Check the correctness of the model based on the model output in real terms.
- Discuss the results of the case study and the applicability of the model.

Mining has randomness as well as continuity. Sometimes trucks are queued at the loader due to various reasons such as service interruptions, loading difficulties, mechanical problems. Then, the loader will not be continuously loading but will be discreetly servicing the trucks. Sometimes trucks will be waiting in the queue sometimes an excavator will be waiting for the trucks. The truck queues are typical in amine by the loader, waiting to be loaded at the blasted bench. Typically, arrival times of the trucks at the loader depend on the smoothness of the other processes at the mine where there is no waiting at the loader and trucks arrives and serviced and leaves in a continuous manner.

A screen capture of this model is used here as an example in Figure 79 shows the difference in servicing customers discreetly or continuously. All the parameters are kept constant at both models. This shows that just a basic model of the entire mining process is not enough. Mining has a high variability in all processes as evidenced by the demonstration of variability simulations in 5.2.1. It cannot be compared to a running factory for simulation. This is one of the biggest challenges faced during the modelling stage.

Causal loops are part of production environment in mining and a system dynamic modelling tool can handle such situations. Almost all system dynamists face them, and many software packages support their creation and display. However, it has been said that they are inherently weak because they do not distinguish between information flows and conserved non-informative flows. As a result, they can distort direct causal relationships between flows and stocks. It is challenging to determine the behaviour of a system just from the polarity of its feedback loops because, stocks and flows create dynamic behaviours not feedback. For the purpose of this thesis, dynamic loops that are descriptive in nature will be avoided to explain the behaviour of a system but needs to be quantifiable.

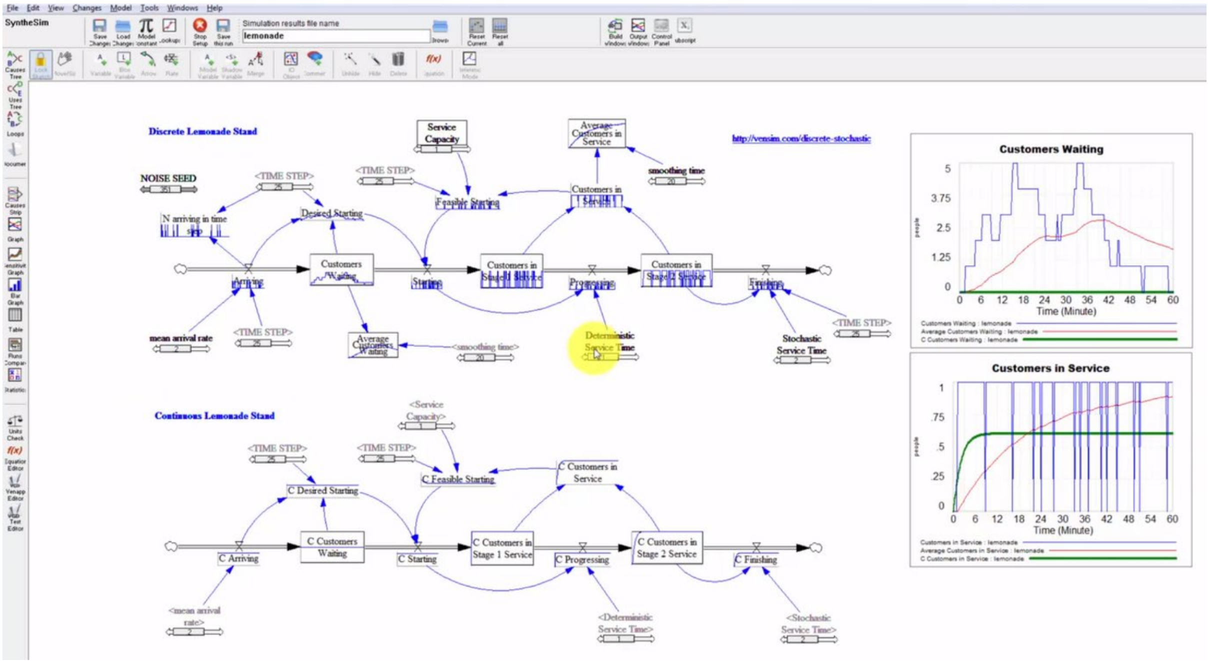


Figure 79 Continuous versus stochastic discrete models

5.5 Conceptualization of Drill and Blast in VENSIM

The drilling process being scheduled and planned for a short term planning includes the following functions:

- Planning data (required annual production, initial desired fragmentation, borehole diameter, bench height, burden, spacing, etc.)
- Ore deposit data (rock mass density, rock strength, hardness, discontinuities or rock mass characteristics from a geotechnical perspective, fissuring, faults, etc.
- Technical and operational data (penetration rate for different rock types, max rotational speed by the drill rig, maximum thrust and torque values, compressed air pressure
- Economic data (cost of consumables such as drill bits, explosives and accessories, cost of renting or buying machinery – capital costs, overheads

All the data is then processed and used for drill and blast short term planning. Short term planning is implemented weekly but planned in two weekly windows, only reviewed weekly to check if implementable based on the field conditions and availability of drills, drilling blocks cleaned surveyed and accessible with ramps, etc. The next step is to allocate drill rigs according to the scheduled production.

Drilling action itself is also an important component in an effective mine and the key areas of an actual drilling process involves the following variable as listed in Table 17. Actual drilling itself will not be the focus of this research but the drilling process in a larger system

Table 17 Productivity and production parameters for drills

Maximum bit loading	Lb (manufacturer units are imperial)
Rotation speed – rated	rpm
Hole/bit diameter	In (often expressed in imperial units)
Hole/bit diameter	mm
Rod diameter – new	mm
Rod diameter – wear	mm
Compressor capacity	m^3/min
Annular area between rod and wall – wear	m^2
Bailing velocity – wear	m/mm
Bench height	m
Sub drill	m
Hole depth	m
Burden	m
Spacing	m
In situ material density	t/m^3
Tonnage generated per hole	t
Productivity	t/m
Pulldown force	t/mm
Rock strength	MPa

Typical information required to determine the number of drill rigs required is as follows (Kennedy, 2000). Although, the list is for determining the number of drills to buy for a new operation the same parameters are also required to determine the short-term planning. For establishing the baseline for this thesis, they need to be mentioned and also placed in the planning of the production as a baseline.

- Hole size, mm
- Bench height, m
- Hole depth, m
- Total hole volume, m^3
- Percent of hole depth filled with explosive, %
- Volume of explosives
- Bulk density of explosives
- Weight of explosives in the hole
- Explosives factor, kg of explosive per ton of rock blasted (powder factor)
- Tons broken per hole
- Total tons ore and waste per year
- Total holes per year, total tons/tons per hole

- Total length of hole, m/year
- Drilling rate while drilling
- Actual drilling time required
- Scheduled annual hours
- Overall job efficiency
- Mechanical availability
- Annual outage factor
- Production utilization, overall job efficiency mechanical availability annual outage
- Actual productive hours, scheduled hours x production utilization
- Drill required, actual drilling time ÷ actual productive hours
- Costing hours, (Scheduled annual hrs. – (Scheduled annual hrs. x 2) -(scheduled hrsx0.05) x2.05

The output of drill and blast process has a direct effect on the loading and hauling. The output is defined with tons and quality (grade and mineralization) as well as metric tons. The drill and blast output are shared with the loading and hauling processes for their planning as well. The feedback can be the grade quality of the ore, tons and volumes of ore and waste, quality of blasting such as fragmentation distribution (muckpile mean fragmentation size), shape of the muckpile, any risks such as misfires, back-break or highwall stability.

Drilling capacity of a mine is the most critical for on time handover of the drilled blocks to the blasting team for the scheduled production. The number of holes per block and the speed at which each hole drilled determines the blasting block availability to the blasting team. A portion of the SD drilling model created in VENSIM is captured and shown in Figure 80. Total drilling time available per week is dependent on the number of drills available as well as the drill rig performance criteria.

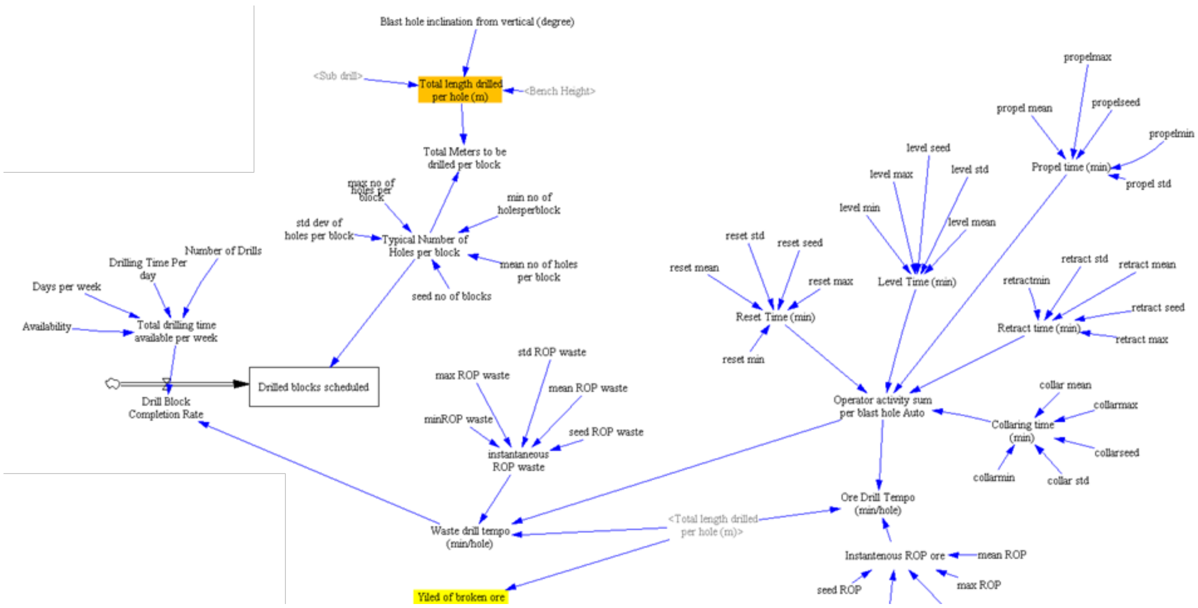


Figure 80 Drill block planning base model created for the the thesis

Drill Capacity Utilization

Capacity utilization is a measure of total capacity being achieved in a given period. The drill fleet not often achieves the maximum capacity available due to several reasons. This will be explored further in this subsection. A basic model found in VENSIM guide on capacity utilization is shared in Figure 81 and will be the starting point of drill capacity and utilization simulation.

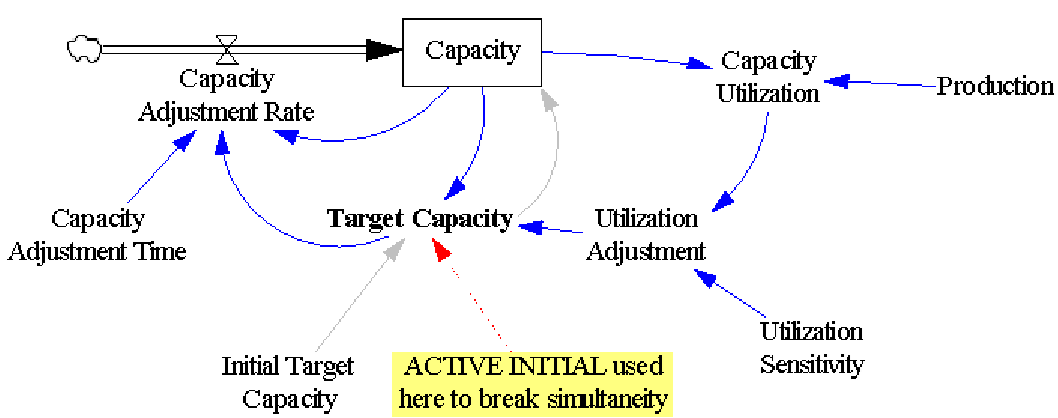


Figure 81 Capacity utilization (Vensim Guide)

Drilling capacity is determined based on the drill tempo which depends on the penetration rates which in turn most importantly depend on the forces applied to the drill bit and the diameter of the drill, etc amongst the many parameters.

5.5.1 Penetration Rates

Instantaneous penetration rates of the drilling rigs depend on the rock type, drilling type and drilling machine variables such as thrust force and rotational speed as well as operator skills. Therefore, it is difficult to estimate the correct penetration rate to be used in the model as an input. Measured real data is available that indicates penetration rates for different rock types. They are listed in Table 18. The statistical description of each rock type may be used to represent variable rock characteristics in the SD model; and a more generalized classification can be used as Ore and Waste and their statistical descriptors also are in Table 18. For this thesis only ore and waste characteristics were modelled to reduce complexity of the model

Table 18 Drilling penetration rates of various rocks as obtained from mine site measurements and the descriptive statistics

Descriptive Statistics of Penetration rates of various rock types (m/min)					
	N	Minimum	Maximum	Mean	Std. Deviation
Hematite BIF	57	.11	.46	.2562	.07693
Calcrete	354	.00	1.27	.6112	.26563
Tillite	82	.03	1.02	.6396	.23933
Calcrete pebble	117	.22	1.08	.7439	.18118
Shale	89	.61	1.69	1.1552	.25631
Hematite	37	.15	1.01	.5864	.17430
Waste	71	.25	1.43	.7952	.24756
Ore	79	.16	.90	.4455	.17520

5.5.2 Blasting Times

It is necessary to incorporate the blasting times into the main model since blasting time is considered as an important delay in the whole mining cycle. A small illustration simulation is constructed and presented here to be incorporated to the main model as seen in Figure 82.

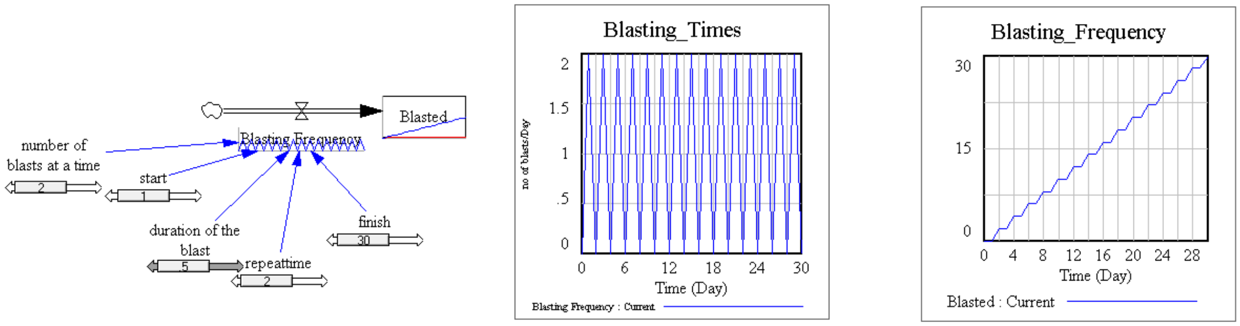


Figure 82 Modelling of blasting Frequency

5.6 Conceptualization of Loading in VENSIM

A typical open pit loading machine is a standard cable operated dipper. More modern ones are hydraulic. Manufacturing specifications or field observations can be used to estimate number of shovels and the available hours for cost calculations for the SD model being created. The following input is the minimum required for a reasonable loading process simulation for the purposes of the thesis:

- Bucket size
- Fil factor (this is a factor affected by fragmentation)
- Average bucket capacity
- Swing time
- Number of passes per minute
- Swell factor
- Weight of bank-meter-cube (in situ volume)
- Loose meter cube
- Weight of loose ore
- Tons per pass
- Annual tons to be loaded
- Annual shovel hours
- Annual Scheduled Hours
- Overall job efficiency
- Mechanical availability
- Annual outage factor
- Production utilization
- Productive hours

- Number of Shovels required
- Shovels in use
- Costing hours

Intrinsic loader performance

Intrinsic loader performance (P_{ah}) is calculated by dividing available time in seconds in a year by mean loader cycle time (C), where mean loader cycle time is measured in seconds.

P_{ah} = Hours per year x Seconds per hour / C

Realistically, certain limiting factors need to be applied to P_{ah} to arrive to a practical productivity. Time factors as described by Hardy (2007) proved to be useful to start putting together the conceptual model for the loader. Some of the factors below are common to other machinery as well such as calendar hours and scheduled hours.

Calendar Hours Hc – solar hours in any given period of observation, e.g., year or day based on solar time – 24 hours per day for 365 (or 366 days per leap year) generally termed "hours" in this thesis.

Scheduled Hours Hs – the annual hours the loader is expected to operate at expected productivity – 8,760 (8,784 for a leap year is generally ignored for convenience). This is then reduced with lost time due to allowances for public holidays, any planned interruptions to operations and daily hours when work is not scheduled, etc

Available Hours (A) need to be determined as the part of scheduled hours when the loader is mechanically and electrically ready to operate. The scheduled hours are basically the time available to the production manage in an open-pit and he is responsible of the available hrs to be used for production. This time excludes service for maintenance.

Utilized Hours or Operating Hours U – the part of the available hours that the loader is actually operating/loading the trucks. This time excludes extraneous lost time due to unplanned interruptions to operation of specific items, groups or all mining equipment and any inefficiency due to logistical mismanagement or operator skills.

A further addition to the factors above is the material factors. *The In situ Density* is one that is used to determine tonnes per BCM (Bank Cubic Meter). This is an inherent property of the material being mined which varies with the geological setting.

Swell Factor is a variable dependent on the inherent structural and geotechnical properties. The mechanisms and degree of disturbance of the material to be mined is measured as the ratio of the volume of a specific weight of material after blasting, ripping and pushing, loading into trucks or dumped at material disposal points where it faces degradation. The disposal places are crusher, ore stockpile or waste dump.

Swell Factor = LCM/BCM = (1 + Swell), (Hardy, 2007) **Equation 16**

- Where LCM = loose cubic meters
- And BCM= Bank – in situ – cubic meters

Equipment factors include Rated Bucket Capacity, Loader Cycle Time, Bucket Fill Factor and Bucket Factor, which are explained below.

Rated Bucket Capacity (Bc): It has two components “struck” and “heap”(see Figure 83. The *the struck capacity* is the volume below a straight line from the cutting edge to the bucket back sheet. It comprises 75% of the rated bucket capacity. The remaining 25% is the *heap volume*. The rated heap volume is based on the height of the heaped material being equal to half of the bucket with.

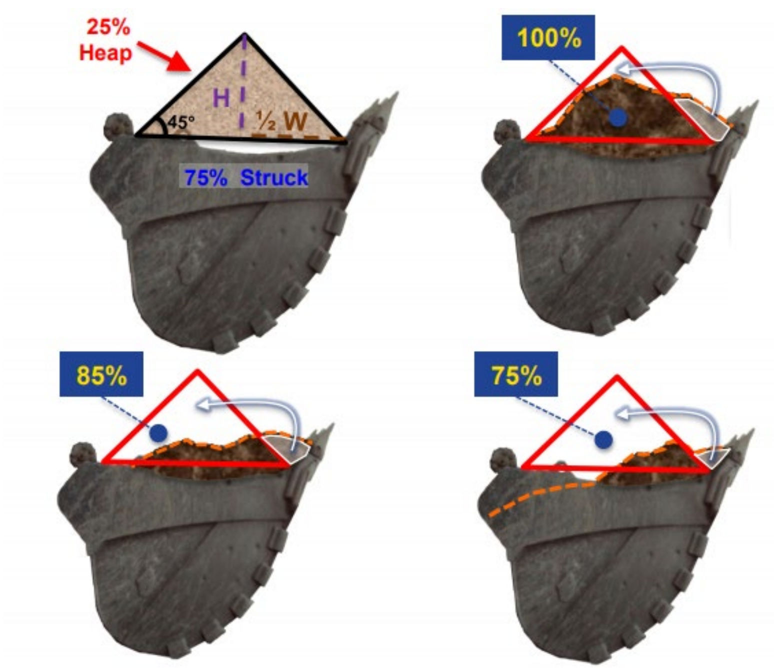


Figure 83 Struck and heap capacity definition by Komatsu™

Loader cycle time in seconds Tc, or cycles per unit of time is useful measure in mining, that includes due allowance for non-optimum digging depth with varying angles of swing – a random variable that is dependent on the condition of material to be loaded, physical characteristics of the loading operation, mechanical and electrical/hydraulic performance of the loader, efficiency of the operator and any extraneous interference that may impose on the loading operation.

Bucket Fill Factor Ff – a variable that typically, for full bucket loads, can be expected to have low dispersion (coefficient of variation of 0.1 or less) considered volumetrically so visual control and measurement is possible and can be effective - generally defined volumetrically as the ratio or percentage of actual volume in the bucket compared with a standard rated bucket capacity that vary for loader types – see next point.

Bucket Factor Bf: A random variable that combines the effects of in situ density, swell and volumetric fill factor generally expressed as a dimensionless factor that can be interpreted in terms of volume or mass depending on the context (Hardy, 2007)

Hardy (2007) adopted dragline productivity to loading shovels and published the following formula:

P_e = Bc x (3600/T_c) x SF x A x U x B_f x PF **BCM/hr or tonnes/hr** **Equation 17**

PF is the propel factor which should be less than 1.

T_C is the average loading cycle time. Digging difficulty can be classified as easy, medium, hard and very hard. Diggability is also a function of fragmentation. The level of impact of fragmentation is difficult to quantify and there is no historic data to indicate otherwise.

Shovel digging depth depends on the bench height and the machine size. According to Hardy (2007) rope shovels optimum digging depth is the same as eye level of the operators. However, optimum digging depth is not a concern for hydraulic shovels.

The following conditions are important parameters that need to be defined in the model:

Swell = [(In situ Volume + Voids) / In situ Volume] – 1 **Equation 18**

= (In situ Density / Loose Density) – 1 **Equation 19**

Loose Density = In situ Density / (1 + swell) **Equation 20**

Swell Factor = In situ Density /Loose Density) **Equation 21**

Load Factor = 1/Swell Factor **Equation 22**

Assessing the degree of fragmentation and shape of the muckpile is possible but time consuming and difficult (Hardy, 2007).

It is hard to establish a quantifiable link between drilling, blasting, loading due to swell factor, load factor and loose density. It is well known that inferior quality drilling leads to blast being *frozen* or compact or too loose to load efficiently. Loading time cycles are greatly affected due to hard digging.

Sometimes raw data may not be so useful for modelling purposes, but statistical bucket load data is of more value in the VENSIM modelling environment for a high level abstraction but still reasonable answers while at the conceptual level. A user may define these parts of a model in more detail depending on the objective of the simulation model. In this thesis the quality of the heaped bucket will be linked to quality and/or level of fragmentation.

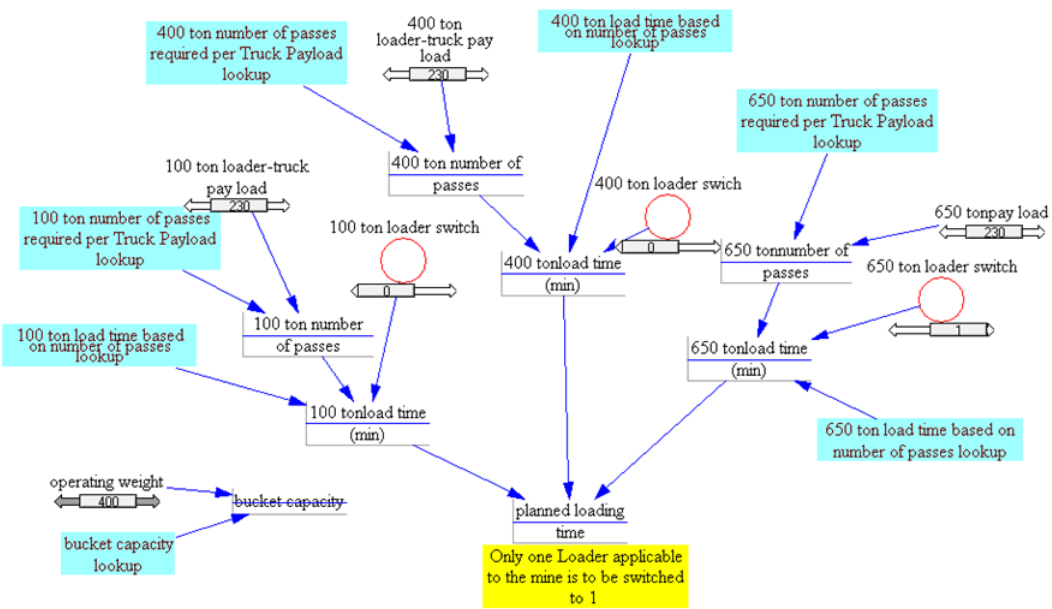


Figure 84 Shovel loading times function based on capacity of the shovel matched to a 230-ton truck.

A small switch function is built to select the matching shovel loading a 230-ton truck which can be seen in Figure 84. For the purposes of not complicating the model further, only one type of truck is allowed for this model, but the payload entered into the model can be varied by dragging the slider per loader on or off options (0 for off and 1 for on) as seen in the red circles in . The payload selected depends on the matching of the loader which is not the subject of discussion for this thesis. The cycle times for loading greatly varies depending on the shovel capacity. They are input in the form of look up tables in the model. The selected truck load payload will have a suitable cycle time matched for each type of loader. An SD model of cause and effect relationship of loading and the impact of fragmentation on the loading process is shown in Figure 85. The bucket capacity of loaders is also an input, and the model determines it automatically based on the shovel size.

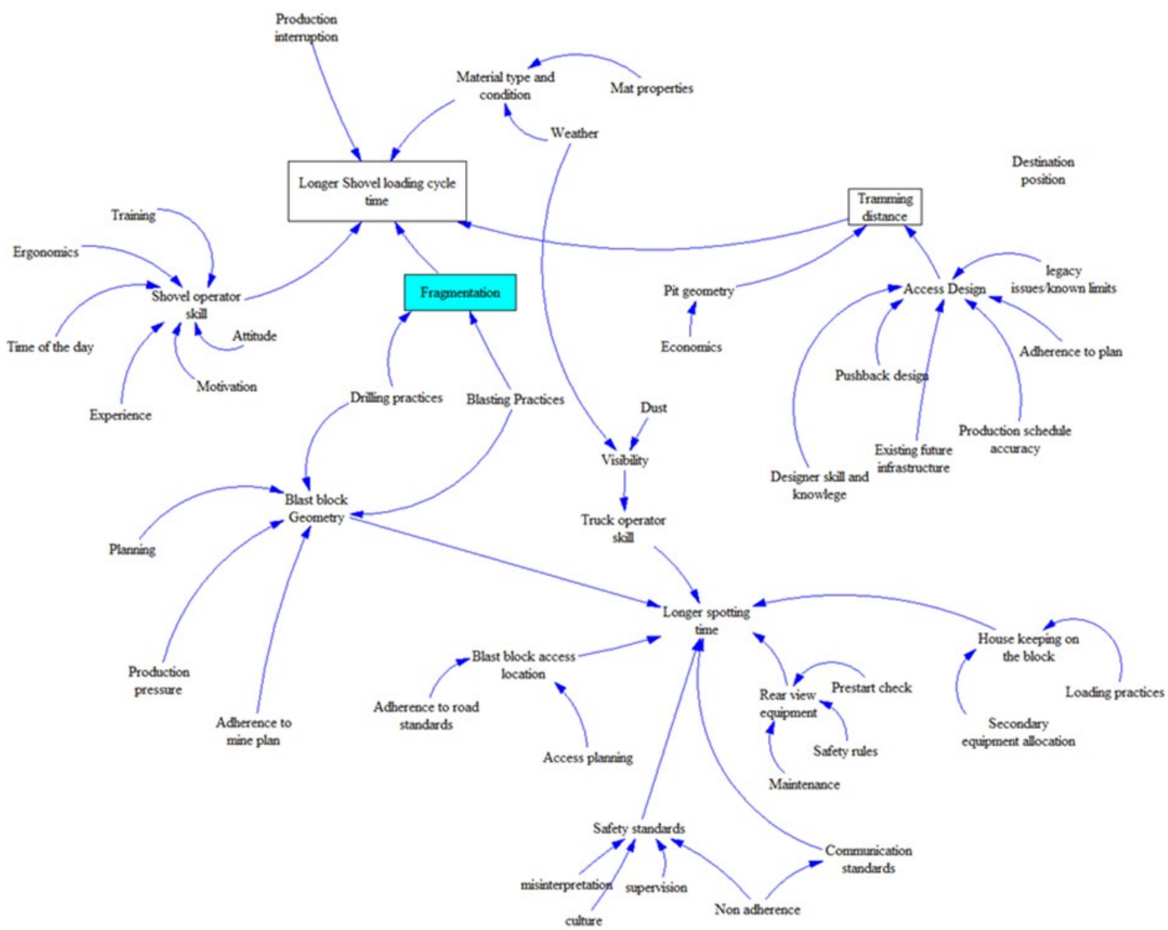


Figure 85 Cause and Effect Relationship of Longer shovel loading cycle time due to poor fragmentation.

The parameters that define the behaviour of loading time are entered into a basic model which shows the effect of fragmentation on the loading time. The model can be seen in Figure 86. The outcome of the simple model demonstrates the impact of fragmentation using p80 values. P80 value is significant as most of the researchers such as Brunton et al (2003), Beyoglu et al (2017) and Zou (2017) tie their empirical formulae to this specific level of fragmentation. P80 is an indicator and does not necessarily represent the whole fragmentation distribution of a muckpile. Fragmentation at P80 level also is used in the empirical formula for power index calculations for crushers and grinders. Therefore, P80 becomes a significant measurement in a series of fragmentation sieve sizes that ties all the other parameters as common denominator. The parameters such as typical values of swing time, dump time, dig time etc are replicated by using statistical values obtained from a typical loading operation. The model can be improved by adding further parameters as shown in Figure 85 where pit geometry truck size, operator efficiency plays a significant role.

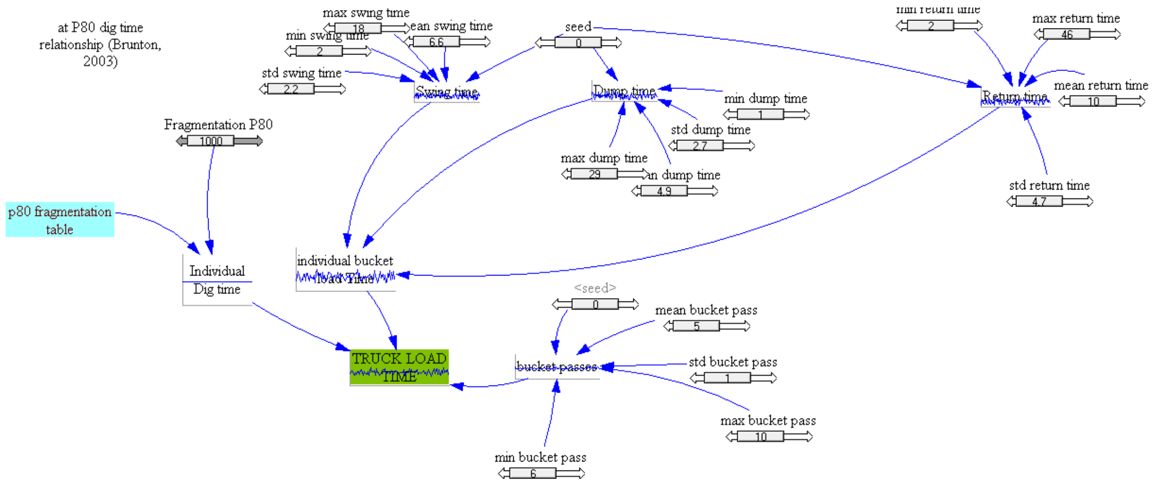


Figure 86 The model being used to demonstrate effect of fragmentation on loading time

5.1 Conceptualization of Hauling Process in VENSIM

Haulage process modelling required are listed below. All initial values used are hypothetical but can be replaced with actuals in the model based on the information collected from a mine being modelled.

- Cycle time
- Trips per hour
- Tons per hour
- Overall job efficiency
- Mechanical availability
- Annual outage factor
- Production utilization
- Scheduled hours per year
- Productive hours
- Annual production per truck per year
- Annual production
- Trucks required
- Trucks in use
- Trucks in fleet
- Costing hours

The key performance indexes that are used for shovels are listed below

$$\textbf{Shovel Utilisation } U(\%) = \frac{\textit{Production time}}{\textit{Available time}}$$ **Equation 23**

$$\textbf{Shovel Availability } Av(\%) = \frac{\textit{Available time}}{\textit{Calendar time} - \textit{Stand by time}}$$ **Equation 24**

$$\textbf{Long Term Shovel Productivity}(LTSP) = \frac{\textit{Tonnes loaded}}{\textit{Calendar year}}$$ **Equation 25**

$$\textbf{Short Term Shovel Productivity} = \frac{\textit{Real Capacity of the Truck}}{\textit{Loading time} + \textit{Parking Time}}$$ **Equation 26**

Long term Shovel Productivity = Short Term Shovel Productivity x U(%) x Av(%) **Equation 27**

A benchmark for world class operations is listed by Caterpillar in Global Mining as seen in the Table 19. Surely the same benchmark can be used for a comparative study

Table 19 Benchmarks for world class mining operations (Caterpillar Global Mining)

Benchmark	BM Value Comments
Mechanical availability	92% (Pre-planned Component Replacement) 88% (Post-planned Component Replacement)
Mean time to repair	3 to 6 hours
Mean time between stoppages	80 Hours (Pre-planned Component Replacement) 60 Hours (Post-planned Component Replacement)
Production utilization	85% - 90%
Maintenance ratio	0.2 Hours (Pre-planned Component Replacement) 0.3 Hours (Post-planned Component Replacement)
% Planned activity	85% to 90%
Record Keeping	100%
Servicing accuracy	95% within +/- 5% of Target

Another benchmark that can be used is cost items (Kirk 2000, cited in Hardy 2007), as listed in Table 20.

Table 20 Cost items as % of Total Mining Costs

Cost Item	KCGM	Ernest Henry %
Equipment Ownership/lease	161	20
Equipment Maintenance	17	19
Drill and Blast	24	19
Fuel and Lubricants	13	12
Tyres	9	9
Total	79	79

Production time corresponds to the time when the shovel was carrying out its main function that is when loading the trucks. The available time corresponds to the time when the shovel is mechanically available, that is, not being services or broken. The standby time corresponds to the time when the shovel is shut down for external reasons such as not being scheduled due to long term planning (Arteaga, 2014)

5.2 Data Representation in VENSIM Modelling Environment

Many mines generate large data, which are often stochastic and partially observable. Input data is based on synthetic data generated by using statistical distributions sampled from an existing mine data and building it in such a way that stochasticity is embedded in each parameter that is being simulated. Each sub-process is to be represented using predictive stochastic synthetic data. There could be various distributions in a mine environment where variables can be represented as uniform, exponential, gamma and normal.

A set of typical probability distribution graphs can be seen in Figure 87 to Figure 89. In order to determine the best fitting distribution function to an existing sample of data a simple statistical distribution fitting software has been used.

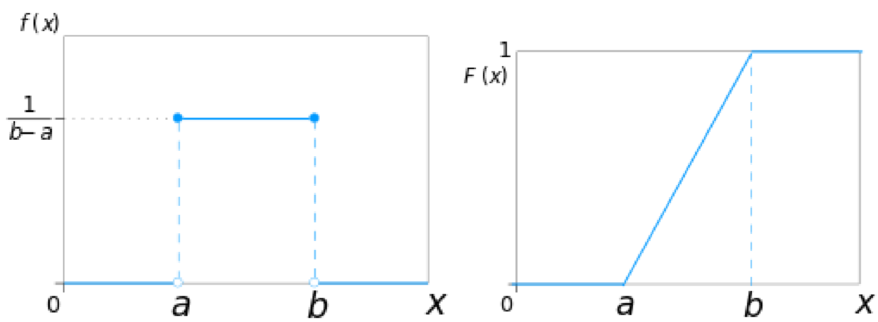


Figure 87 Probability distribution functions – uniform distribution with probability density function (left) and cumulative function (right)

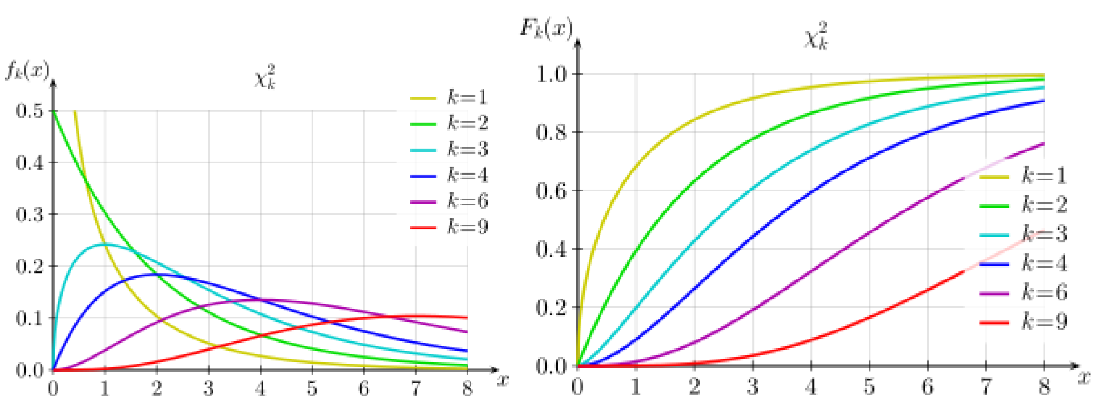


Figure 88 Probability distribution functions – Chi square for probability density function (left) and cumulative distribution function (right)

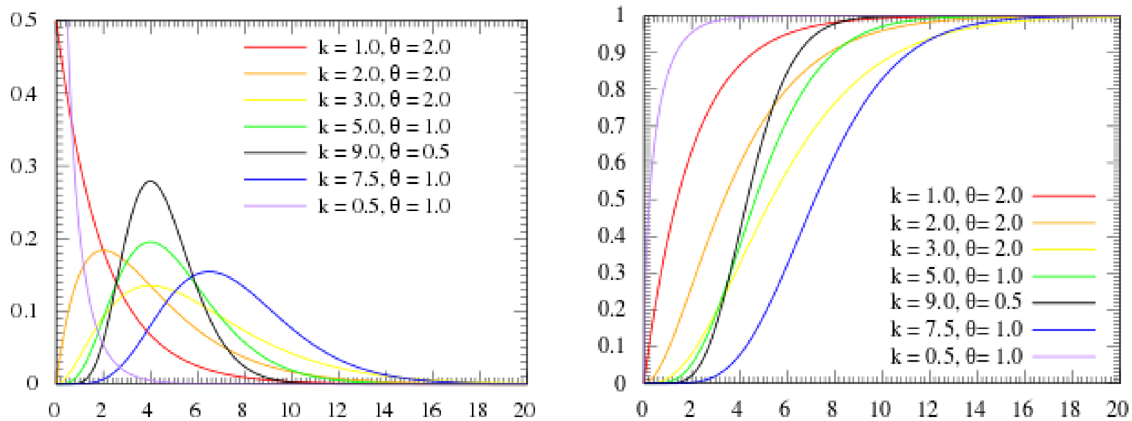
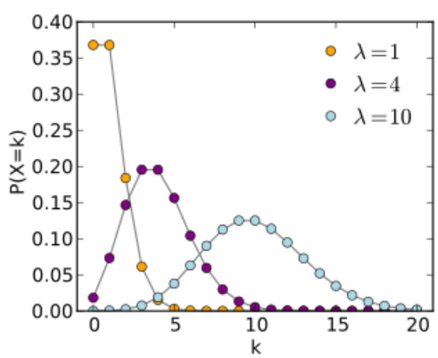
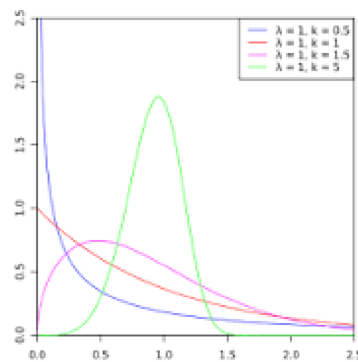


Figure 89 Probability distribution functions – Gamma distribution function for probability density function (left) and cumulative distribution function (right)





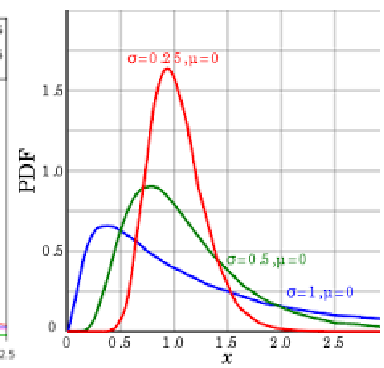
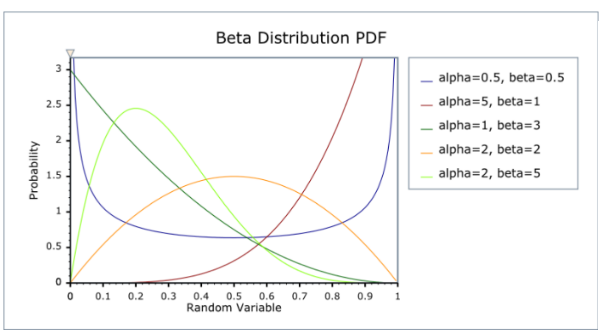


Figure 90 Poisson distribution

Figure 91 Log normal distribution

Figure 92 Weibull



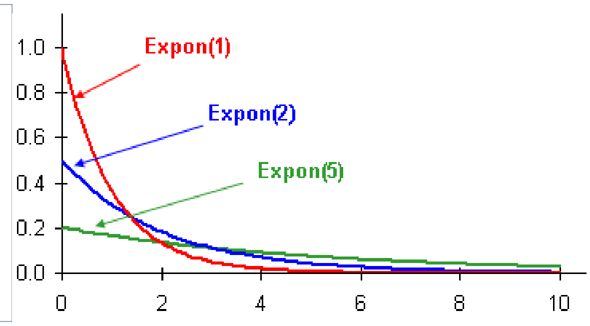


Figure 93 Beta Distribution

Figure 94 Exponential Distribution

The way VENSIM handles random functions and distributions is summarized below in the form of formulae built in VENSIM:

RANDOM Number Functions

RANDOM 0 1( ) RANDOM number between 0 and 1

RANDOM BETA(m,x,A,B,h,r,s) BETA distribution alpha=A and beta =B

RANDOM BINOMIAL(m,x,P,N,h,r,s) BINOMIAL on N trials of probability P

RANDOM EXPONENTIAL(m,x,h,r,s) EXPONENTIAL starting at 0 with mean 1

RANDOM GAMMA(m,x,O,h,r,s) GAMMA with order O

RANDOM LOOKUP(look,m,x,h,r,s) RANDOM number using LOOKUP PDF

RANDOM NEGATIVE BINOMIAL(m,x,P,N,h,r,s) NEGATIVE BINOMIAL N successes prob P

RANDOM NORMAL(m,x,h,r,s) NORMAL with mean 0 and standard deviation 1

RANDOM POISSON(m,x,M,h,r,s) POISSON and mean M

RANDOM TRIANGULAR(m,x,S,P,T,s) TRIANGULAR between S and T with peak at P

RANDOM UNIFORM(m,x,s) UNIFORM between m and x

RANDOM WEIBULL(m,x,S,h,r,s) WEIBULL with shape S starting at 0 with mean 1

Where

m is the minimum value that the function will return. Where necessary the distributions will be truncated to return values above this. Truncation occurs after the output has been stretched and shifted. If the number drawn is below this value, it will be discarded, and another number drawn.

x is the maximum value that the function will return. Where necessary the distributions will be truncated to return values below this. Truncation occurs after the output has been stretched and shifted. If the number drawn is bigger than this, it will be discarded, and another number drawn.

h is a shift parameter that indicates how much the distribution will shifted to the right after it has been stretched (but before being truncated).

r is a stretch parameter that indicates how much the distribution will be stretched before it is shifted and truncated. Note that for the NORMAL distribution **h** and **r** correspond to the mean and standard deviation.

s is a **stream ID** for the distribution to use

RANDOM 0 1() is uniformly distributed on the range 0 to 1. It is obsolete and will return the same noise stream as RANDOM UNIFORM(0,1,0). It is retained to maintain backward compatibility only.

RANDOM BETA(m,x,A,B,h,r,s) provides a BETA distribution with alpha having the value A and beta having the value B before it is stretched, shifted and truncated.

RANDOM BINOMIAL(m,x,P,N,h,r,s) provides a binomial distribution where P is the underlying selection probability and N is the number of draws. Before stretching and shifting, RANDOM BINOMIAL always returns an integer between 0 and N. If N is not an integer it will be rounded to the nearest integer.

RANDOM EXPONENTIAL(m,x,h,r,s) provides an exponential distribution starting at 0 with a mean of 1 before being stretched, shifted and truncated.

RANDOM GAMMA(m,x,O,h,r,s) provides a gamma distribution of order O before it is stretched, shifted and truncated. When O is 1 RANDOM GAMMA is the same as RANDOM EXPONENTIAL. If O is less than 1 a warning will be generated and 1 used.

RANDOM LOOKUP(look,m,x,h,r,s) provides an arbitrary distribution with a probability density function specified by the Lookup function look. Before stretching or shifting the random number will have the same range as the x-axis in the Lookup. This means that the Lookup is the same as the probability density function (PDF) except that you do not need to make the area under the Lookup 1.0, Vensim will automatically adjust for that. The dimensions of look should match m, x, and h. After the random number is drawn from the supplied Lookup PDF it will be multiplied by r then have h added then tested against m and x.

RANDOM NEGATIVE BINOMIAL(m,x,P,N,s) same as binomial except N is the number of successes required so that random negative binomial returns an integer from N to infinity.

RANDOM NORMAL(m,x,h,r,s) provides a normal distribution of mean 0 and variance 1 before it is stretched, shifted and truncated. This is equivalent to a normal distribution with mean h and standard deviation r. The units of r should match m, x and h.

RANDOM POISSON(m,x,M,h,r,s) provides a Poisson distribution with mean M. The value returned is always an integer before it is stretched and shifted. The units for M should match m, x and h.

RANDOM TRIANGULAR(m,x,S,P,T,s) provides a triangular distribution from S to T with a peak at P. You can shift and stretch the triangular distribution by adjusting S, P and T. The units for S, P, and T, should match those of m and x.

RANDOM UNIFORM(m,x,s) provides a uniform distribution between m and x (exclusive of the endpoints).

RANDOM WEIBULL(m,x,S,h,r,s) provides a Weibull distribution with shape S starting at 0 and having a mean of 1 before it is stretched, shifted and truncated. When S is 1 the Weibull distribution is the same as the exponential distribution.

In addition to above, it is possible to do sensitivity simulations (Monte Carlo) with VENSIM and can be useful to determine behavioural boundaries of a model testing the robustness of the model-based policies. An example of how to use it in SD modelling is shown in the screen capture (Figure 95).

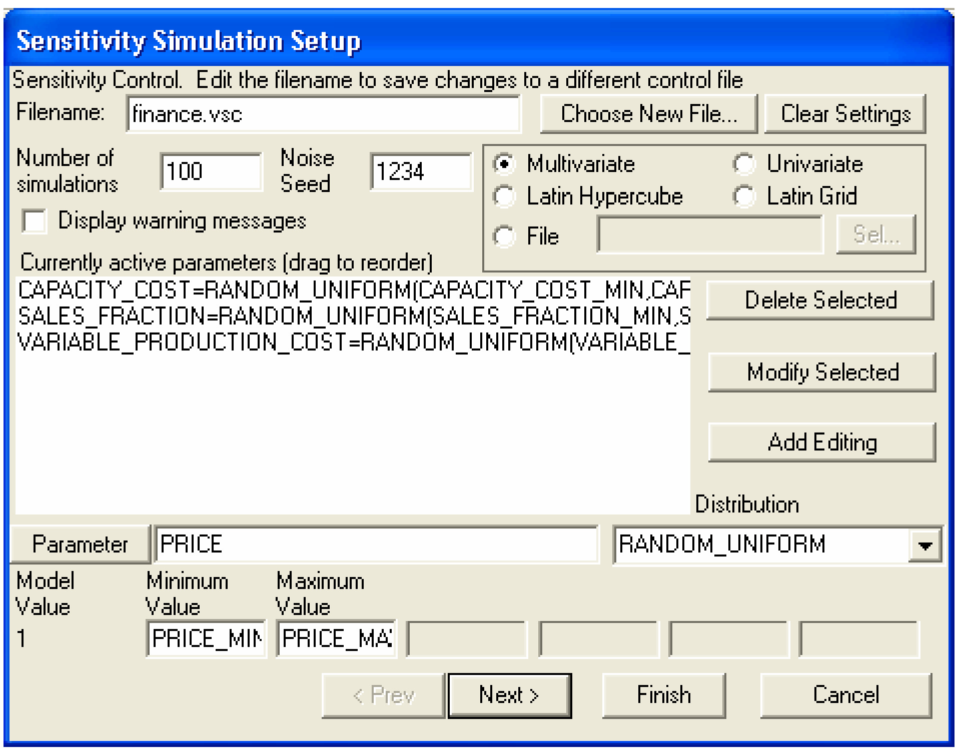


Figure 95 VENSIM sensitivity setup (source:Vensim Guide)

The Monte Carlo sampling technique are frequently used in mining simulations (Kennedy, 1990). Monte Carlo analysis is a computer-based method of analysis developed in the 1940's that uses statistical sampling techniques to obtain a probabilistic approximation to the solution of a mathematical equation or model (Robert, 2004). Monte Carlo simulation can be used for any type of distribution not only normal distribution. This depends on the type of data. To be able to determine the best fitting type of model a sample data is required to determine the best fitting distribution statistically. There is various software capable of testing the data for the best fitting curves. The simplest way to check for the distribution type is using Excel statistical functions or create one's own probability density function fitted to the data. In VENSIM multivariate sensitivity analysis is used for this type of simulations.

Deciding which distribution best fits the several types of data is a tedious task. Once a type of distribution function is determined that fits the data then a test of the goodness of fit is performed. There are various statistical software that automatically fits the data until goodness of fit is no longer improved. The parameters are then determined. These parameters can be used to regenerate similar curves using Monte Carlo simulation techniques. In this way realistic data is generated by simply using the formulae obtained from curve fitting exercise. This makes the model free from spreadsheet type of data for describing the behaviour as is the case for this thesis.

5.3 Model Validation

According to Ford (1999) five validation methods stand out to be deserving special attention which are:

- Verification
- Face validity
- Historical behaviour
- Extreme behaviour
- Detailed model check

5.3.1 Verification

A model is verified when it is run in an independent manner to learn of the results match the published results. This is simply a test if the model runs as intended. Although verification seems like an obvious test, one needs to consider that real world is much more complex. Therefore, certain aspects of the model need to be checked partially if it fits expectations rather than total model.

5.3.2 Face Validity

This test is called a common-sense test. The modellers need to ask themselves if the model structure and parameters make sense. This test relies on a modeller's understanding of the system to judge the structure. For example, certain parameter moves in the negative direction or wrong direction. For complex models it is difficult to see the face value.

5.3.3 Historical Behaviour

One of the most common tests is to set the inputs to historical values. The historical test is informative if majority of inputs are endogenous, and a limited number of inputs are exogenous.

5.3.4 Extreme Behaviour

This is the most revealing test regarding the test of model parameters to extreme values such as 0, or extremely large numbers. If the modeller is satisfied with the result model is considered plausible. However, if the model is constructed to simulate the most likely conditions and it is working within range, this must be stated so to the model reviewers. An example to extreme test is *market shares* example. If the market share exceeds 100% there is something wrong in the model.

5.3.5 Detailed Model Check

As the name suggests this is to be done for fine detail on the formulae and correctness of the units, relationships etc.

5.4 Chapter Summary

The chapter highlighted important information required towards putting together the new system dynamics model to quantify the impact of technological changes in a surface mine by marrying the technology behind each mining sub process and the parameters that define these processes in the modelling environment. The main processes that are discussed with their relevant parameters are planning, drilling, blasting, loading and hauling. The expected behaviour of sub processes is also explained at the unit level to later include in the actual model. The SD model being developed for this thesis is described in the next chapter (chapter 6) and will use the concepts explained in this chapter.

"If the building blocks are so shabby, is it worthwhile building integrated models at all? The answer is clearly yes, despite the present weaknesses of the models. The reason is that modelling forces us to reveal our assumptions and changing those assumptions show how important they are with respect to the outcome."

(Toth, 1995)

Chapter 6

6 SYSTEM DYNAMICS MODELLING OF THE MINE PRODUCTION ENVIRONMENT

The objective of this chapter is to show and discuss the steps taken to develop a new SD model of each mining process using a system dynamic tool that will be used to quantify the impact of any technological changes in a mine. This is necessary to build the argument towards the justification of the objective for calculating the effect of change in the cyclic mine processes. In Chapter 5 mining processes were modelled individually as well as conceptually. It also demonstrated how system dynamics models are constructed and interpreted for the mining environment that is to be treated as a system and defined with an SD tool called VENSIM. In summary how it all comes together is explained in this chapter.

The purpose of the model being created for this thesis is to demonstrate the consequences of changes in the various mining parameters by studying the causality of the processes with the help of a system dynamics tool, specifically, the drilling quality in the interconnected mining production system. The main unit processes to be included in the simulation model in a typical mine are drilling, blasting, loading, hauling, crushing.

Data collection, reduction and organization are often bottlenecks in simulation studies as stated by Barnett (2003). He also states that tendency to model everything and collect as much data dooms many simulations. This thesis taps into any data measured previously by others to establish the base model. The model attempts to include the unit processes in terms of functions and relationships in a mine rather than data driven causality modelling most traditional approaches do. Thereafter it is up to the user to include mine specific data to get accurate results for that specific mine. The principle of causality needs to be captured in the form of formulae for a properly running SD model. Therefore, the data will be used rather to check if the model is correctly setup and creates sensible information.

6.1 DRILLING and BLASTING

6.1.1 The drilling Performance and the Effect on the Downstream Processes

This chapter is also dedicated to discussing challenges in the drilling operation and how the drilling performance affects the rest of the mining processes. This in turn justifies automation efforts in a mine, therefore, it is important to understand how blasting quality is dependent on the accuracy of drilling. This will be discussed in the next section.

The drilling process is simplified in the schematic as in Figure 96 will be the guide to the reader.

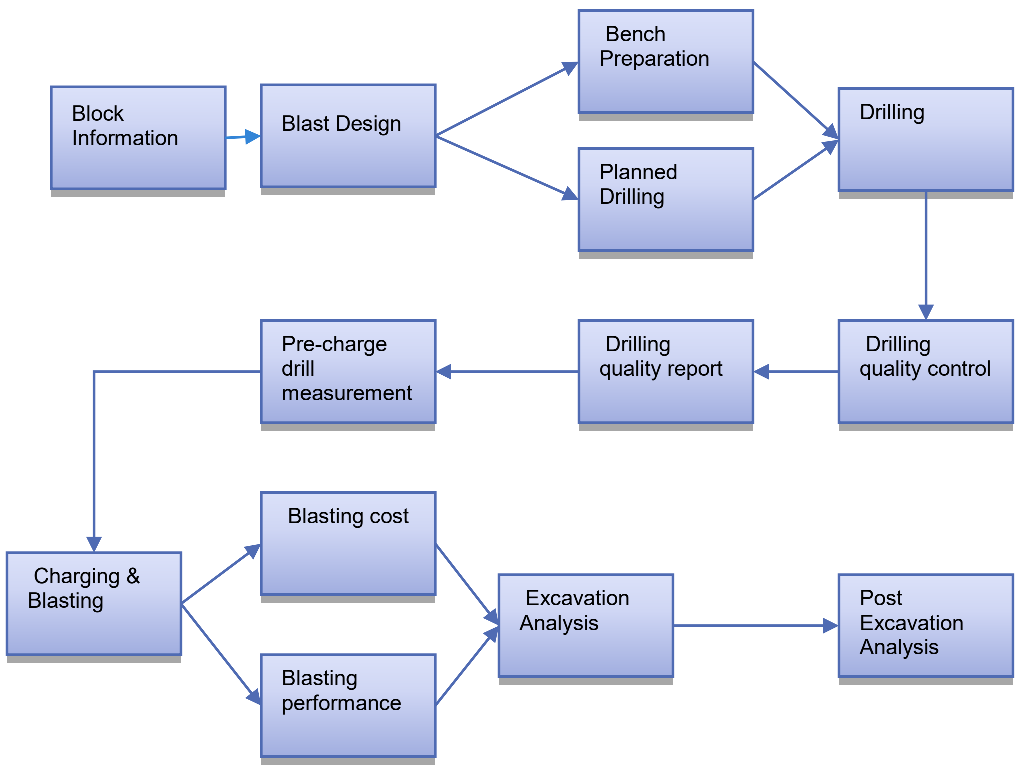


Figure 96 Drilling process map

A blasting block is a piece of stock that goes through multiple processes and the information captured at each stage is to include the following as seen in Figure 97.

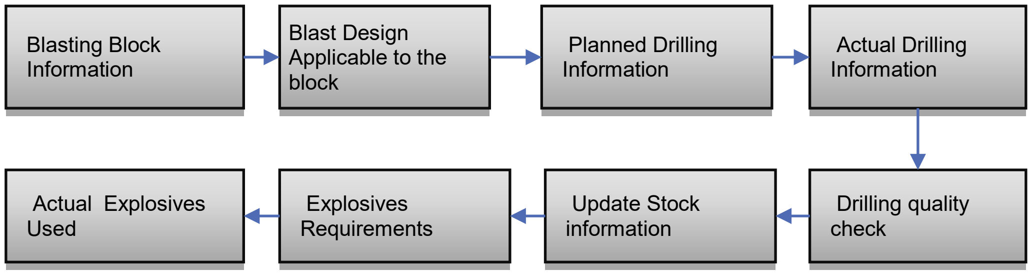


Figure 97 Steps in the quality management for mining processes

Then the next step is to check performance of blasting due to the drilling as indicated in Figure 98.

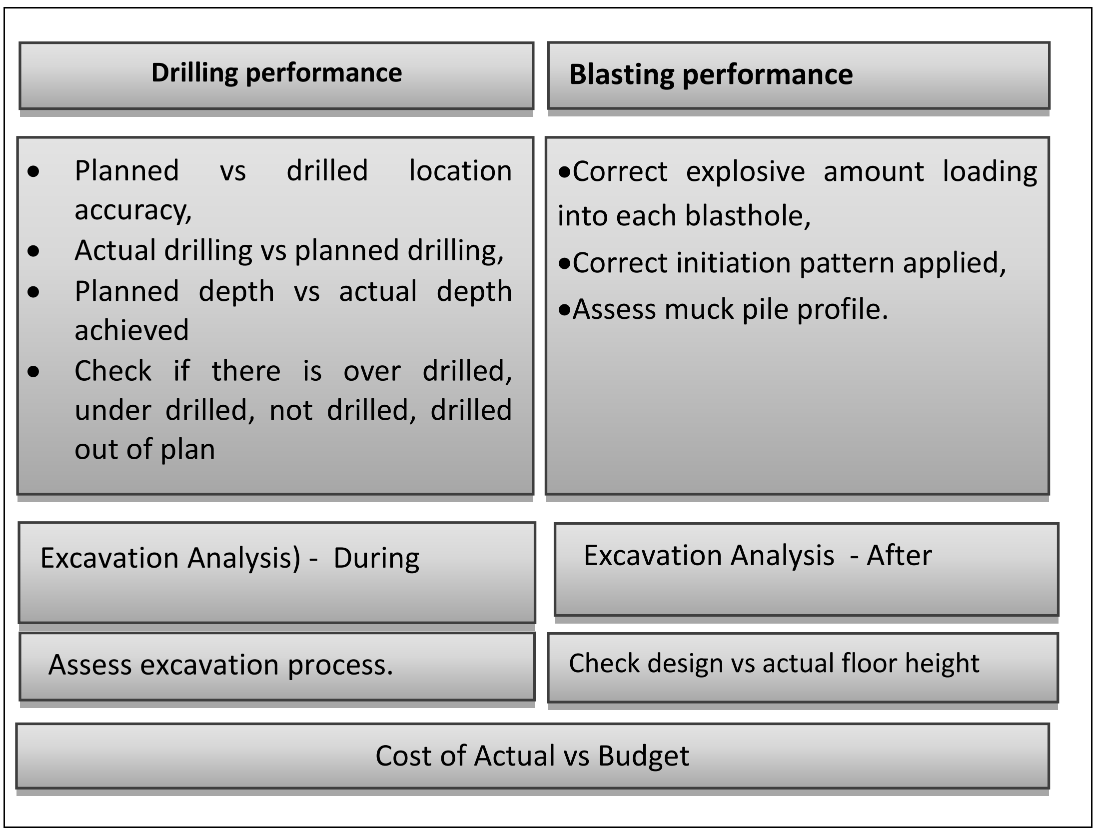


Figure 98 Excavation analysis during or after blasting

The following is expected from a good blasthole drilling process and communication to the management on the progress or non-progress of drilling in terms of meters per time together with drill quality reports.

1. Pre-drilling activities planned to match blasting requirements and loading and hauling requirements.

2. During drilling activities to ensure that blastholes are drilled in exact location and at the required rate as planned
3. Post Drilling activities to make sure that drilled holes stay open free from falling debris and capturing of the quality of drilled holes such as drilled to required depth without deviation from the required angle and at required X-Y coordinate

In addition, the following may be added:

4. Capturing of drilling data while drilling
5. Drilling performance measurement
6. Rock prediction while drilling
7. Tracking of consumables
8. Real time QA/ QC

The following scenarios are possible during the drilling operation

1. Drill pattern is not staked according to plan
2. Drill pattern has missing holes
3. Drill pattern is not completely drilled as planned with missing rows
4. Drill block is half drilled to quickly provide blasted material to the loading team by blasting half block. This may cause double effort in terms of blasting times for the same block of ground and adds to lost production time. Smaller blasting blocks should be avoided. The effect of blasting smaller blocks will be demonstrated in the simulation modelling.
5. Shorter drills
6. Closed off drill holes
7. Collapsed drills

Consequences of all the above points are somehow difficult to incorporate into a simulation model to capture the effects accurately. But a degree of quality can be built into the model in the form of rates from 0 to 1. To demonstrate how to incorporate such effect into the model operator efficiency factor was used. Figure 99 shows operator efficiency and how it can be incorporated into the model. The direct influence of the operator would be on the drill time itself. If the user wants to see how it changes overall drill performance and cost implication it is a matter of simply dragging of the input box slider when in "synthesim" mode within the model.

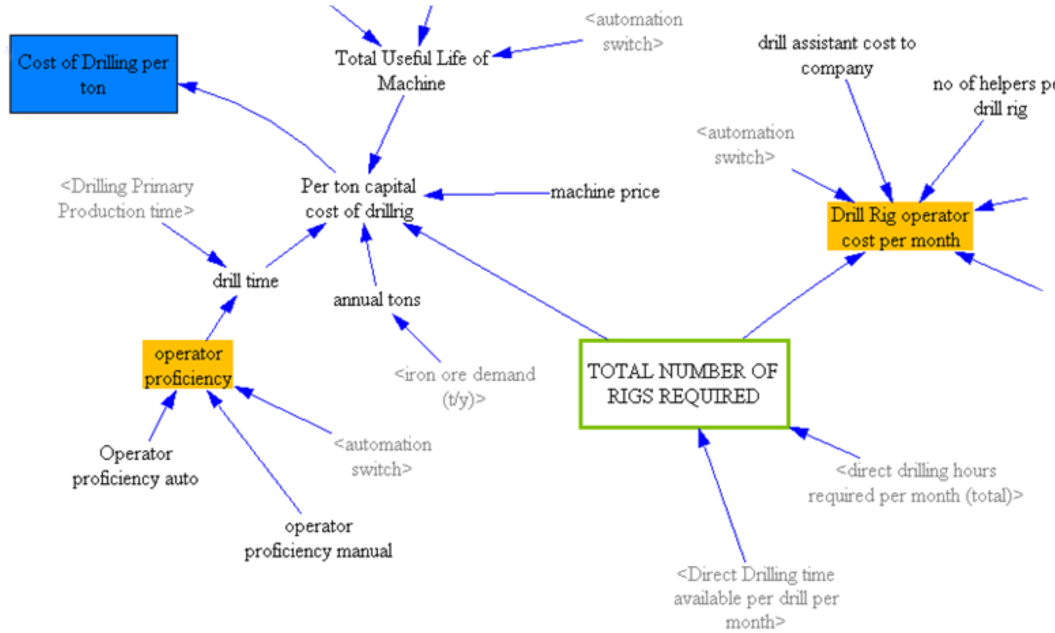


Figure 99 Incorporating causality in the form of percentage point

In addition, an automation switch is incorporated to make the selection dependent on the mode of operation which is seen in Figure 99.

When the automation switch is set to a value of zero then the function selects the manual proficiency value, when the automation switch is set to 1 this time the value for automatic is assumed. Therefore, the accuracy of the model depends on how accurate the user wants to be in terms of the numbers used.

Often assurance of the drill quality requires inspections during and upon completion of the drilling of the blasting block and subsequently handed over to blasting task team. This requires additional manual workforce, with GPS navigated automated drilling this would have been captured automatically while drilling. Effective communication between the drilling and blasting teams is vital to identify serious errors and either rectify the situation or if not, blasting engineer needs to adjust explosive loading plans. Normally there is a signoff procedure by the blasting engineer before drilling team hands over the drill block to the blasting team. This manual process does not always guarantee accurate information passed on to the blasting team. Once the block is blasted the blasting will be of substandard quality leading to the usual difficulties of loading hauling, fly rock or fragmentation problems as was discussed in Chapter 4. Once blasting happens the reasons or evidence of the substandard quality will also be destroyed with blasting. Measurement while drilling (MWD) becomes important for on time information sharing between the two teams.

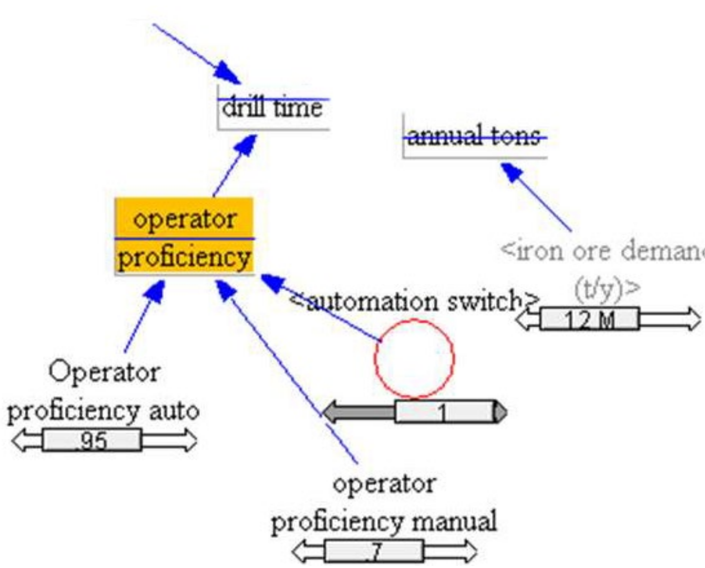


Figure 100 Interacting functions built into the model for input changes during the study visible in the form of sliders

An inferior quality drilling effects on burden, which in turn results in locally choked blastholes or, floor humps. The resultant profile of the floor affects the loading machines' performance therefore loading capacity is reduced. There is also a risk of drilling into undetonated explosive column. If a misfire is identified, it is necessary to drill next to the undetonated holes which is a risky process due to possibility of hitting a live detonator. The precision of the drilling therefore is critical in these high-risk zones. Sources of drilling errors is summarized in the Figure 100.

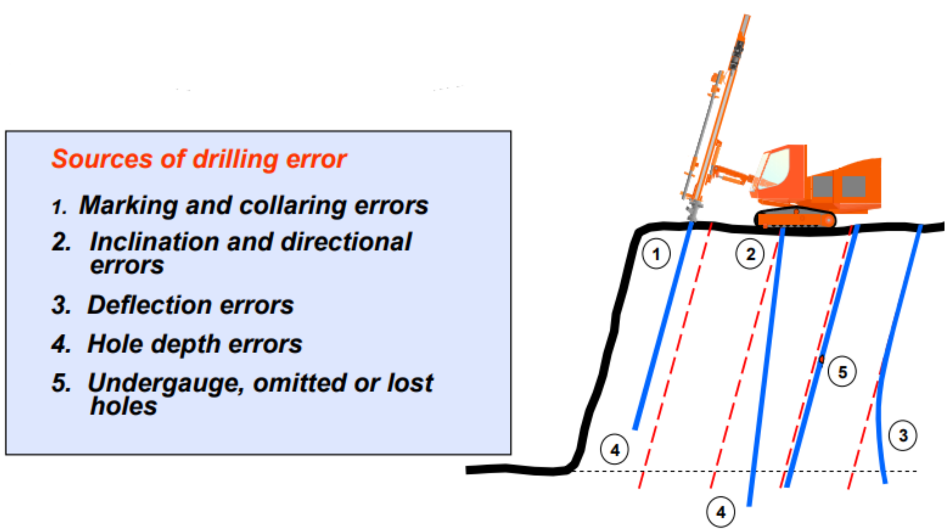
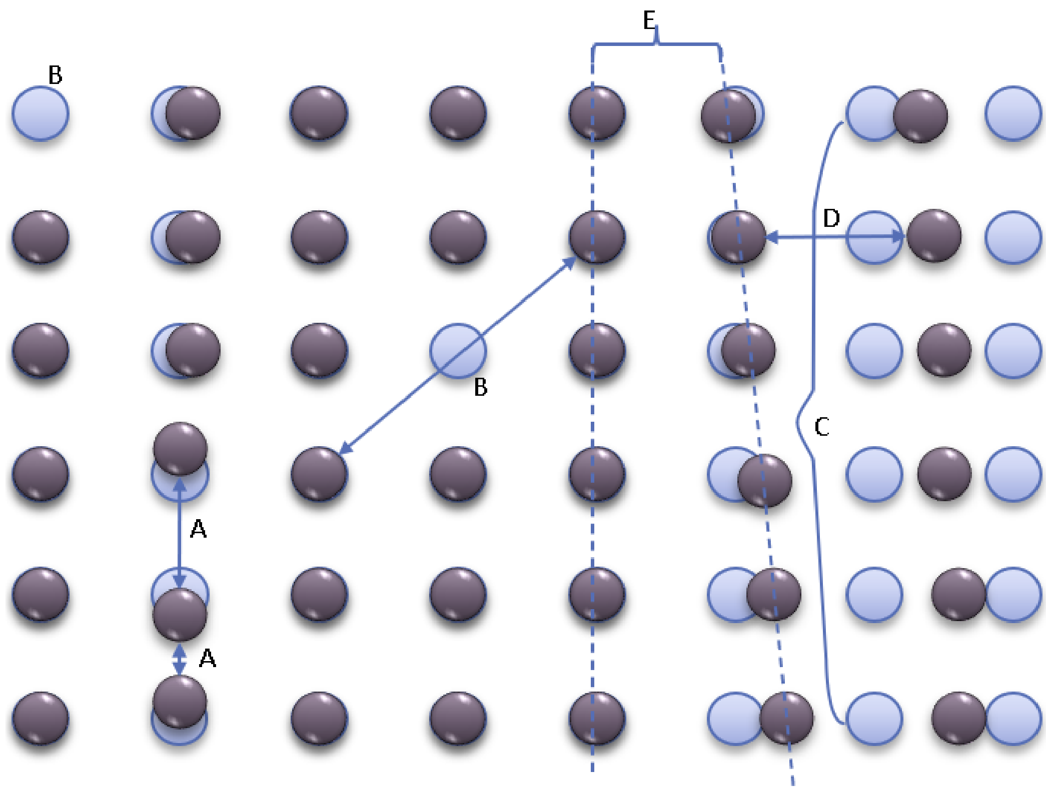


Figure 101 Drilling errors (source: Sandvik)



A- Misaligned holes, too close or too far apart (wrong Burden distance)
B- Missing holes
C- Whole row not drilled (results in multiple problems for the blast)
D- Last row is doing the job of two rows of blastholes (in the field it could be perceived as "no missing row of holes" but in fact one raw missing
E- Field surveyor throws the line in the wrong angle.

Figure 102 Multiple drilling error types due to X-Y location misaligned on the bench

Consequences of incorrect drilling on the XY plain is summarized in Figure 102. Explosives energy is designed based on the Burden distance. If the blast holes are separated too far away, i.e., too much burden distance then the explosive will not be effectively breaking the surrounding area. If the blastholes are too close to each other than the surrounding area of the blastholes are pulverised, also it means waste of explosive energy and waste of drilling effort, adding to costs and increasing cycle times. This is not desired for iron ore due to increased fine fragments. When a complete row of blastholes that was planned but not not drilled will either increase the number of boulders or high "toes: on the bench as demonstrated in Figure 102.

High toes could be a result of shorter drilled holes as well. This picture is just a small depiction of what could go wrong if blasthole is not drilled at the required location. The errors regarding Z dimension (depth of drill holes) will be explained with the help of the drawings in Figure 103 and Figure 104.

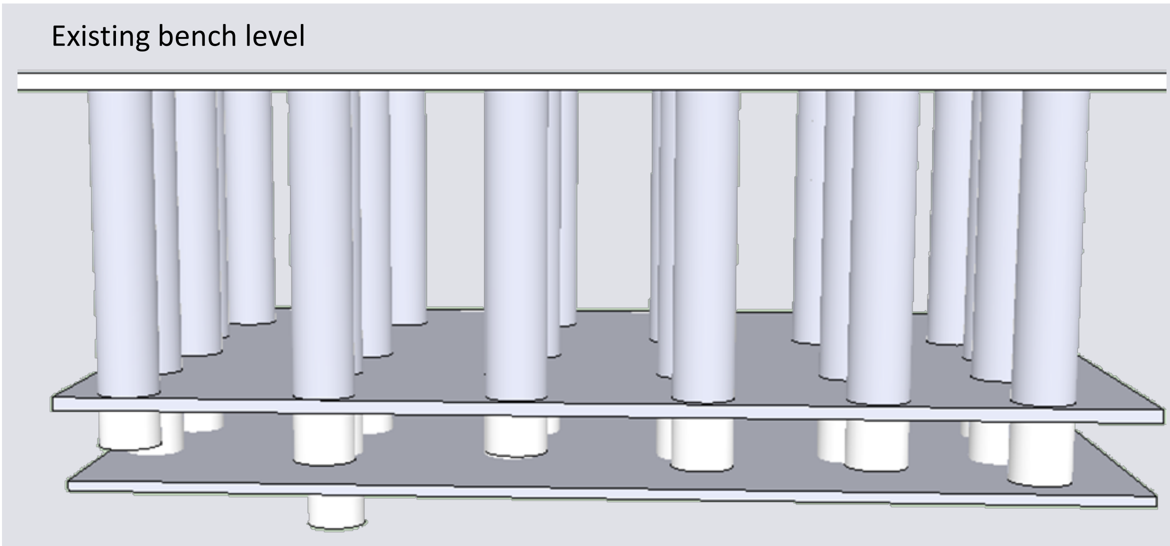


Figure 103 Depiction of a block of ground drilled showing designed versus actual elevations

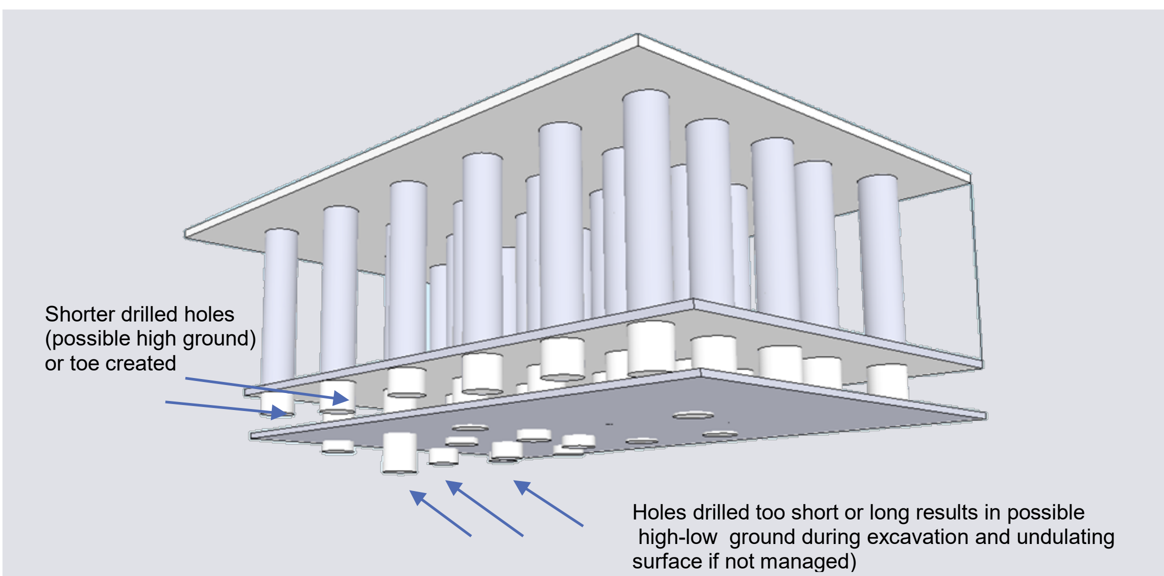


Figure 104 High ground or low ground creation due to incorrect drill hole lengths

Before the drilling process starts, surveyors are informed of the drilling pattern approved by the blasting engineer. Once the drilling pattern is obtained the pattern needs to be staked in the field by the surveyors. The holes are marked either with wooden poles or small concrete blocks.

Each drill hole needs to be drilled to a certain depth predetermined by the surveyors so that a guaranteed even level is obtained after the block is blasted and loaded out. This information should be clearly visible on the stakes for the drill operator, who then drills to the required depth. Sometimes this information is not captured correctly. The bench preparation quality determines the quality of the blastholes as well.

The depth of holes even if it is drilled to the correct depth may still be in error if the staked hole area was dug too deep by the previous loading action. Naked eye will not detect a meter of low ground easily on the field. A manual drill system will setup the drill rig assuming it is at the correct elevation, and drill required length of drill but due to low ground from previous bench activities it will be 1 meter deeper. In addition, explosives loaded on the blasthole will not be loaded at the similar elevation to keep the bench even for the next round of excavation.

Sometimes due to production pressure high grade areas need to be drilled and blasted much quicker for plant demand for high grade ore etc. Thereafter additional drill rigs are put on the drilling block to speed up the drill block handover to the blasting team. This would cause congestion on the block. This also leads to drilled holes are trodden over with the rigs walking over already drilled holes and the drill chippings and debris falling into the drilled blast hole and therefore shortening of the drilled hole. If the block of ground is in a congested high traffic area, this poses a problem in terms of increased interaction between mining crew and other machinery adding to the safety risk. Ideally, a blasting block should not get more than 2 to 3 drill rigs per block for safety. A mine site is show in the picture in Figure 107 where blasting block is allocated with three drill rigs, but for a larger block where drill rigs are comfortably away from each other, and the block is not confined. It is not always the case.

Drill collaring errors can sometimes be extreme as seen in Figure 105 where two blastholes drilled too close to each other. This can pose problems in terms of sympathetic detonation as well as fragmentation problems amongst many others. The drill chippings piled around the blasthole collar can fall into the blast hole due to wind rain, or high traffic if stands open for too long. This can result in blasthole being shortened. In addition, may result in contamination of the explosive in the toe area causing weakening of explosive and therefore deflagration instead of detonation. This further adds to fragmentation problems as well as toes and stiff muck pile which is difficult to dig by the shovel.

Figure 105 Drill collaring errors

Figure 106 An example to drill deviation (source: Mining Magazine, 13 Feb 2015)

Correct Presplit: Integrity of a pit wall relies on the drilling accuracy to get good quality half barrels. The straightness the barrels left behind is indicative of the quality of the presplit. These barrels, visible only after the blast takes place, is often inspected to check the completeness and quality of the presplit drilling as a measure of the quality of the drilling. Presplit drilling quality is critical to the highwall and bench stability.

Figure 107 A view of a drilling block with three drill rigs allocated to it.

6.1.2 Cause and Effect Relationship for Improved Drilling and Blasting

The cause-and-effect relationship for a drilling and blasting process is mapped out in Figure 108 that shows how processes are interdependent, therefore quality of one process leading to improvement in the other processes leading to many other benefits including plant yield. This causality is a typical miner's experience in general but never quite easily quantified in a holistic manner. The intention is to look at the individual processes and detail the causality at a quantifiable level so that real effect can be realized by the management.

The following are to be incorporated in value simulation:

- On time drilling
- Coarse fragmentation
- Drilling consumables are reduced

The blasting site will be neat therefore undulations will be minimal, this will lead to efficient drilling.

- Cost of secondary blasting.
- Improved cycle times
- Reduced maintenance
- Reduced operational cost
- Increased productivity

- Increased revenue
- Reduction of lost blasts
- Increased efficiency of internal processes
- Increased staff productivity
- Reduced operational cost
- Reduced energy consumption
- On time feedback
- Improved quality of blasting
- Improved fragmentation
- Improved wall stability

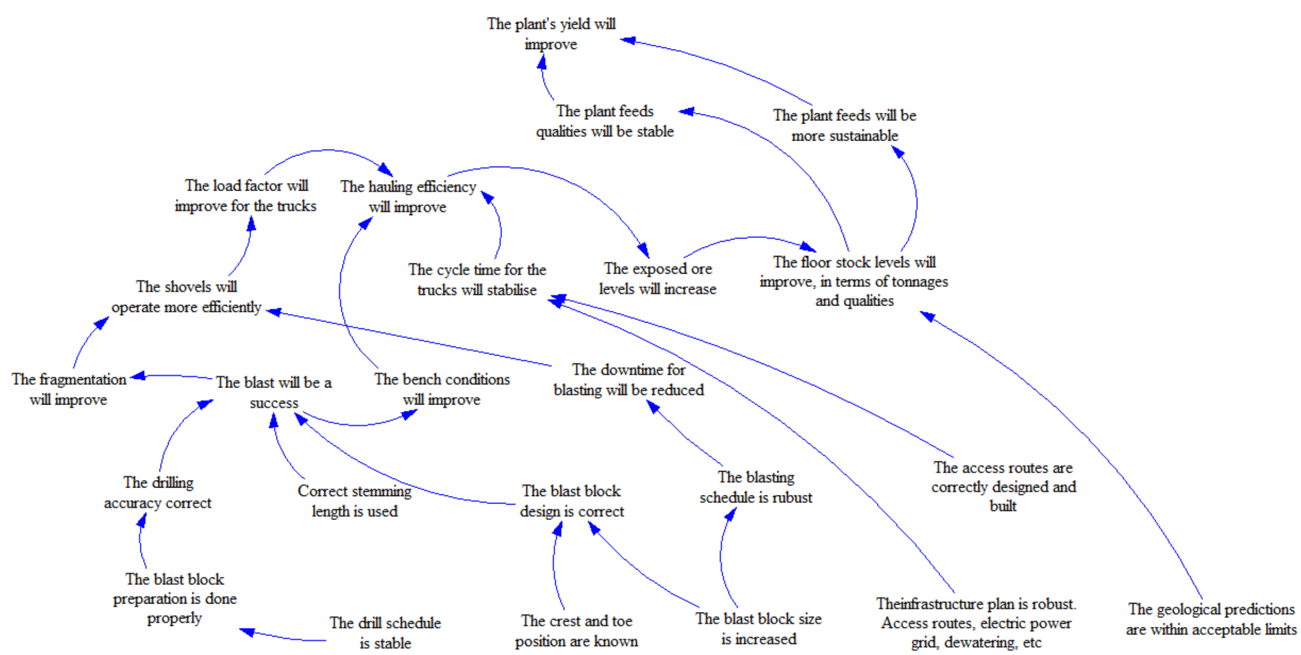


Figure 108 Cause and effect relationship of improved drilling and blasting

6.1.3 Impact of Drilling on Fragmentation

The relationships between fragmentation size distribution and loading and hauling processes and crushing and grinding are all important in assessing the quality and performance of blasting. An integrated approach has kept the researchers busy based on the hypothetical cost function of increasing average particle size (Dinis da Gama 1980). Mean size of the fragments of the blasted muckpile can be consistently achieved only if the designed blasting parameters are adhered to, i.e., burden and spacing distances, depth of the blasthole and the powder factor used for the blasthole.

Consistent "Burden" and "Spacing" distances are important for uniform fragmentation. The following are possible due to an incorrect setup of a drill rig.

- Increased burden distances leading to oversize
- Decreased burden distances leading to fines generation
- Shorter drilling leading to high toes requiring additional effort and causing increased cycle times of the loaders
- Longer drilled holes in the previously blasted bench becomes the problem of the next round in the stemming area being damaged in the level below therefore generating big boulders due to inefficient explosive confinement in the stemming zone.
- Uneven floor leading to decrease in loader performance and increased bench preparation activities

A typical cost per ton versus fragmentation study is represented in Figure 109 (Cardoso, 2015). The figure clearly provides evidence of lower cost effect of smaller mean fragmentation size distribution.

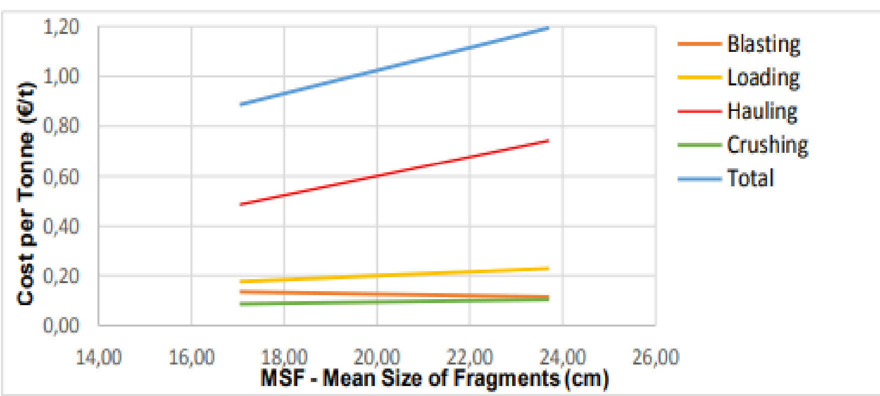


Figure 109 Relationship between the cost per ton and mining processes based on mean fragment size (Cardoso, 2015)

The most significant cost item in Figure 109 is hauling, followed by loading. The figure did not include crusher and mill power consumption, but they are also significant cost items. Shovel's performance depends on the fragmentation. The haulers will be delayed if there are delays at the shovel and therefore the crusher and the plant will not be fed at consistent levels. If there is a queue at the shovel due to hard digging conditions, this probably means drilling and blasting quality was not at its optimum, attributable mostly to the drilling performance.

Many researchers studied digging performance versus fragmentation of the muck pile. In addition, make and design of loading equipment, operator efficiency, loading trajectory, swell and muckpile shape and looseness are the main factors of the loading efficiency. If the mean fragment size is known, there is a better chance of predicting the rate at which hauling cost increase which can be determined via modelling.

But the mean fragment size is not easy to determine practically. The most accurate way of determining fragmentation size distribution of a blasted muckpile is to do a full-scale sieving. However, there are only a few of these studies due to practical difficulties and cost. Therefore, some relied on photographic techniques such as Split Engineering software. It was introduced from 1980's onwards where two-dimensional photos of a muckpile are analysed to obtain a fragmentation distribution curve. However, it only gives insights about the muckpile surface. Photographic analysis is therefore indicative but not close to the reality. It is believed that prediction models can give a better indication of the muckpile size distribution compared to the photogrammetric techniques.

The amount of fines in a blasted muckpile is often miscalculated by photogrammetric methods, if the measurement of the amount of the crushed fines in a blast is desired to be accurate then direct sieving of the muckpile is to be done which is costly as well as disruptive for large-sized mines. Therefore, researchers have been working on a more accurate prediction of the fragmentation size distribution of the resultant muckpile. Rock fragmentation by blasting is a complicated but attractive field by many researchers such as Kanchibotla (1988), Kuznetsov (1973), Cunningham (1983), Cunningham (1987), Cunningham (2012), Djordjevic (1999), Chung and Katsabanis (2000), Esen (2003), Onederra (2004), Ochterlony (2009, 2017), Faramarzi (2013). Cuninngham's Kuzram model has been the most widely used and discussed amongst these researchers.

Kanchibotla et al (1988) pointed out that Kuzram model underestimates the contribution of fines. The coarse part of the distribution is predicted using conventional uniformity index based on blast design parameters and this model was originally proposed by Cunningham (1987). This is the part that this research will be using for cycle time performance calculations of loaders and haulers that are impacted by substandard breakage of rock therefore, correct prediction is critical. The fines portion of the fragmented rock is also important to be predicted for iron ore mines as the revenue for fine ore product is less compared to lumpy ore product.

To understand the impact of fragmentation one needs to be able to predict particle size distribution. All the above researchers have detailed ongoing studies for prediction of the size distribution.

The size distribution curve should be ideally used to correlate with loading difficulties, crusher energy consumption, losses due to fines, or coarse fragmentation leading to additional processes or drilling and blasting cycles. There is no direct calculation of the impact of fragmentation yet. Sieving analysis can be costly and biased towards a certain part of the blast as variable rock characteristics increase variability on the excavation quality for each process. Therefore, based on rock conditions, perhaps prediction can still be the best tool to give an indication of the success of a blast. Quantification of the impact of fragmentation is critical. For the purposes of this study, the most practical prediction model will be selected and incorporated to the SD model.

Below, a brief description of fragmentation prediction theories and models is given, of which one of them can be incorporated to the SD model.

Kuz-ram Fragmentation Prediction Model

The Kuz-Ram is an empirical fragmentation prediction model in terms of mass percentage passing a given mesh size. Based on field observations Kuznetsov (1973) formulated a semi-empirical equation that relates the mean fragmentation size with the applied blast energy per unit volume of rock as a function of rock type (Morin & Ficarazzo, 2006).

$$R_x = exp\left[-0.693 \cdot \left(\frac{x}{x_m}\right)^n\right]$$ **Equation 28**

Where Rx is the fraction of material retained on screen at x size, x is the screen size, x_m is the mean size and n is a constant called uniformity index.

Where:

A = Rock factor

V_0= Rock volume broken per blast hole in m3. (Burden * Spacing * Bench height)

Q = Mass explosive used in the blast (kg)

Also expressed as a function of powder factor:

$V_0/Qe = 1/K$ **Equation 29**

$$n = \left(2.2 - \frac{14 \cdot B}{d}\right) \cdot \sqrt{\left(\frac{1+S/B}{2}\right)} \cdot \left(1 - \frac{W}{B}\right) \cdot \left(\frac{|BCL-CCL|}{L} + 0.1\right)^{0.1} \cdot \left(\frac{L}{H}\right)$$ **Equation 30**

From equations (4) and (5) the mean fragmentation size Xm for a given powder factor can be calculated as follows:

$$x_m = A \cdot K^{0.8} \cdot Q^{1/6} \cdot \left(\frac{115}{RWS}\right)^{19/20}$$ **Equation 31**

The factors that the Kuz-Ram model does not take into consideration are (Cunningham, 2005)

- Variable rock mass properties,
- Jointing of rock strata,
- Number of blast holes and number of rows,
- Bench dimensions relative to drilling pattern,
- Timing between detonating holes,
- Velocity of detonation,
- Decking,
- Water, air and stemming in blast holes,
- Previous blast conditions, results and geological influences.

The accuracy of any modelling and simulations that explore the effects fragmentation and blast design changes have on downstream operations depends extensively on the accuracy of rock mass characterization and the ability to model and measure fines generated during blasting (Kanchibotla et al, n.d.)

The Kuzram model does provide for good fragmentation estimates but there is still possibly room for improvements. Spathis (2004) addressed a common mistake that is the median size calculation was mistaken for mean size.

Gamma Based Blast Fragmentation Model (Faramarzi, 2015)

A blast fragmentation model is developed by Faramarzi et al (2015) based on a gamma function which takes care of errors of fines and coarse prediction by combining three models where one predicts fines portion better and the other predicts coarse portion better. By stitching at a reasonable point (in this case 40% passing size) he believes he has obtained a better distribution curve closer to reality and it is not a complex process to do that. Costs of primary crushing should always be analysed as part of a cost prediction model since it greatly influences the total production costs.

Swebrec Function

The Swebrec function (Ochterlony, 2005) is given by

$$P(<x) = \frac{1}{1+f(x)}$$ **Equation 32**

$$f(x) = \left[\frac{ln\left(\frac{x_{max}}{x}\right)}{ln\left(\frac{x_{max}}{x_{50}}\right)}\right]^{b}$$ **Equation 33**

Where P(<x) is percent passing, x is the rock fragment size, x_{max} is the largest fragment size in the distribution, x_{50} is the size at 50% passing, and b is a curve modulation factor. To best fit the curve x_{max}, x_{50} and b are to be determined.

Practical application of Swebrec function is described by Ochterlony (2005). Determination of x_{50}, and b is described below combined with the functions listed above. x_{max} however can be estimated based on the burden, B, parameter in a blast. The difference of this model to Kuzram is that Rosin Raimmler function and an equation are replaced with P(x) and x_{50} functions.

$$x_{50} = AQ^{1/6}(115/s_{ANFO})^{19/30}/q^{0.8}$$ **Equation 34**

A=0.6(RMD+RDI+HF) **Equation 35**

$$b = 0.5{x_{50}}^{0.25} ln\left[\frac{x_{max}}{x_{50}}\right]$$ **Equation 36**

Ochterlony (2005) explains only one short coming in this model is that he does not yet know how x_{max} depends on the specific charge. It could also depend on burden, spacing and timing of the blast.

Gheibie et al (2009) modified Kuzram model by modifying the “Rock Factor” mentioning that his model proves how important he rock mass condition is in terms of fragmentation prediction.

For modelling purposes initially Kuzram will be incorporated into the simulation model because of simplicity and widely accepted practical applications. For this study which method has been used will not matter too much now but, in the future research, it could be addressed with the field measurements. This is beyond the scope of this research. The indicative impact of fragmentation rather than replicating actual reality will be the outcome in this research if Kuzram is used. Reality is too complex and variable, therefore whatever attempts are made to model fragmentation as close to the reality it will not be near to the real values; rather it is going to be some form of approximation to show the impact of correct drilling on other processes via fragmentation estimate.

Kuzram prediction model will be used for indicative purposes for simplicity of modelling and incorporating the effect of fragmentation on the rest of the processes. Due to time constraints the other fragmentation models have not being studied for the SD modelling.

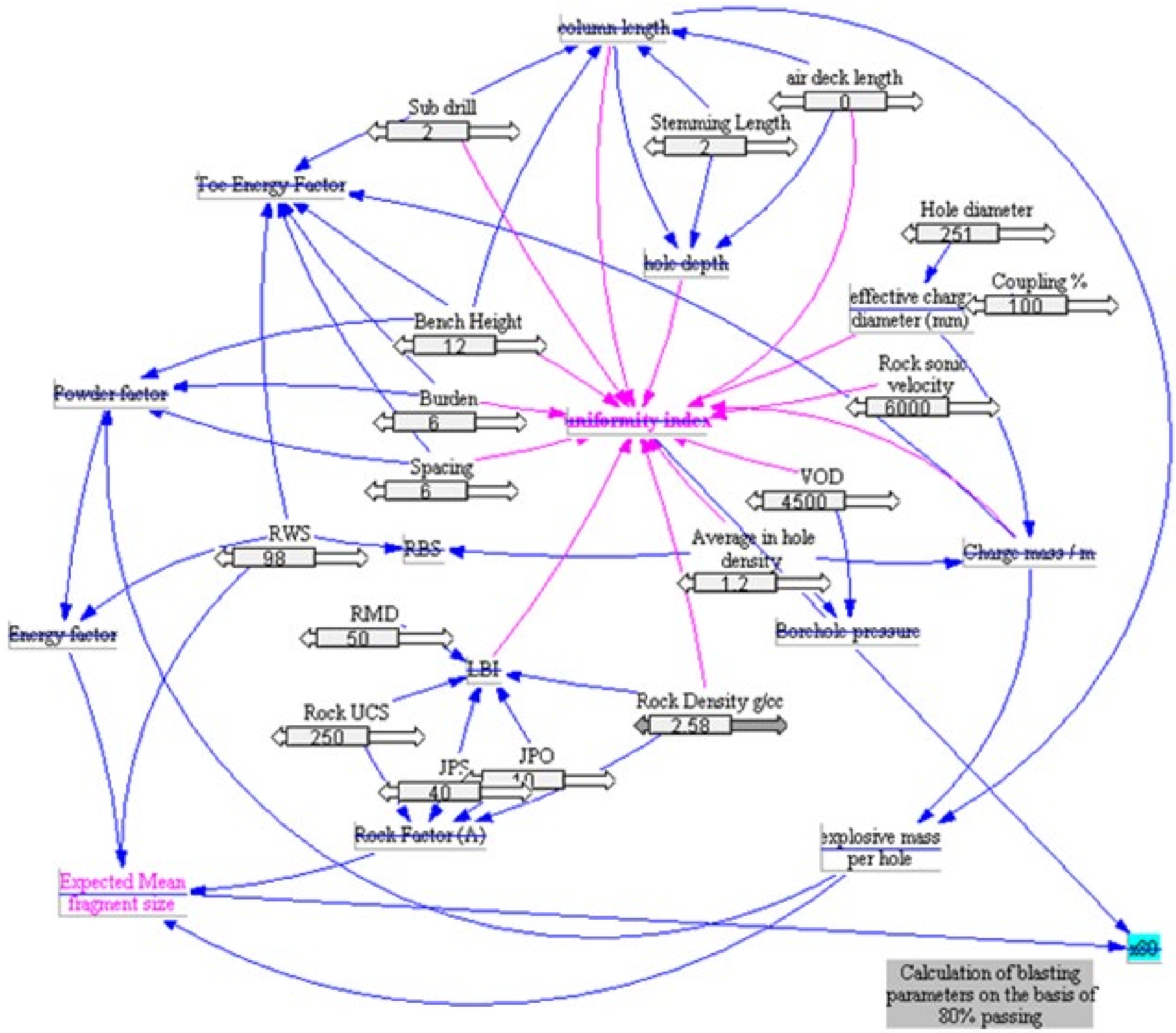


Figure 110 Fragmentation model created using Vensim

The basic fragmentation formula using Kuzram has been implemented in the system dynamics SD model using Vensim as seen in Figure 110. This model is integrated into the larger SD model considering all factors in the production, such as loading and hauling, crushing power consumption based on P80 fragment size, effect of fines on revenue, etc. In this model 20 sieve sizes can be selected for determination of the percentage values of a muckpile’ s fragmentation distribution. The model is such that whatever sieve sizes are selected the total of the fractions are always unity therefore this flexibility is great considering each operation will have a distinct set of size fractions.

Each rock type as well as blasting parameters will also result in various fragmentation levels. Therefore, model will adopt to any surface mining environment.

Incorporating the Kuzram prediction model into the larger SD was achieved successfully despite the complexity surrounding the formulae. The complexity is not in the formulae but in determining size ranges that automatically adjusts to the modellers preference of the sieve sizes, since every crushing plant has different sets of sieve sizes designed for that specific plant. Therefore, the following setup has been put together as seen in Figure 111. The modeller using this SD model is free to choose sieve sizes by dragging the sliders to the correct levels. The set of sieve sizes (in mm) used here are the ones used at Kolomela mine that has been selected as the case study. The left side of the simulation are inputs of sieve sizes, and the right-hand side gives the fraction of muckpile in that size range. Therefore, all the fractions have to add up to unity.

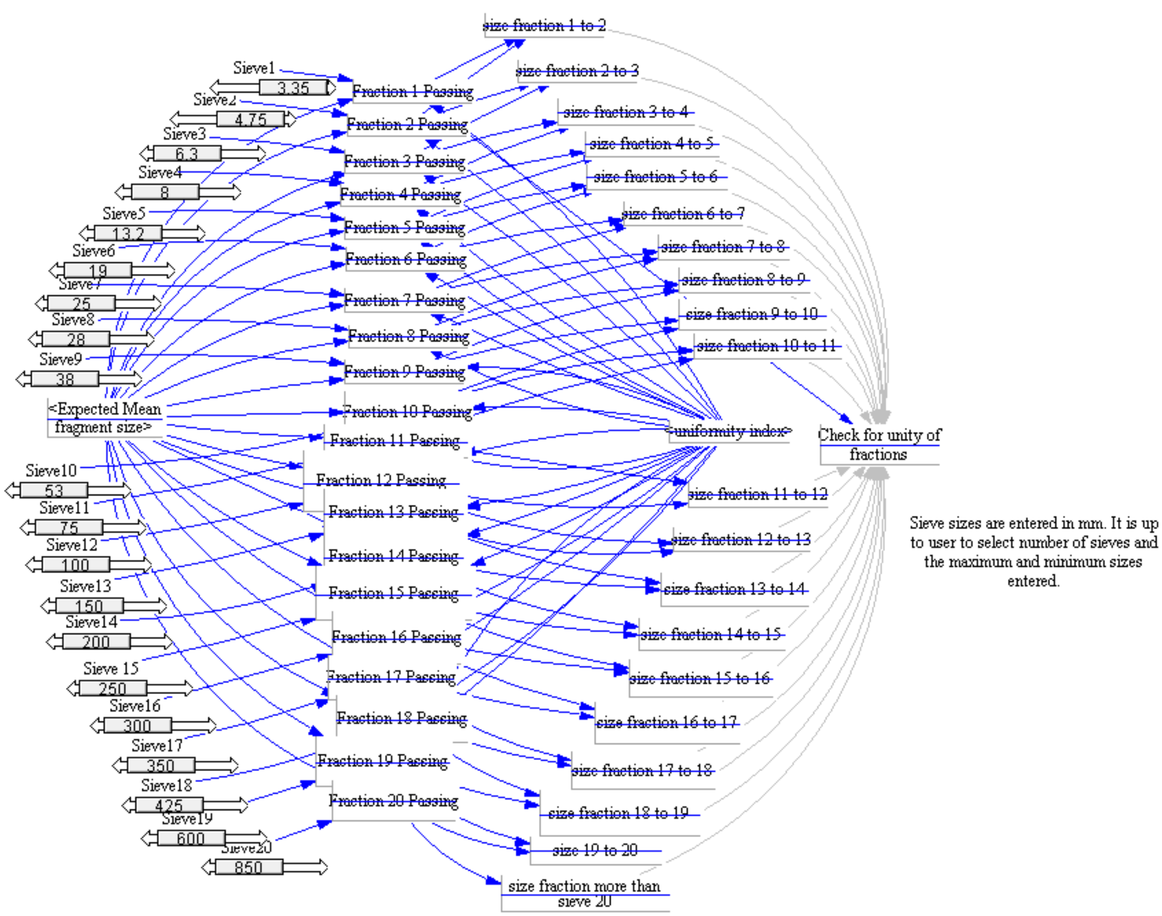


Figure 111 Crushing plant sieve sizes linked to Kuzram fragmentation prediction model.

Any drill plan starts with the detailed short term mine planning. The mine plan schedules certain blocks to be mined at regular intervals to feed the plant. The production plan and how it is done will not be discussed here in detail. However, a basic model showing the required tonnage per month or yearly basis is constructed for this thesis. Basic economic functions such as Cost per ton, revenue, present value and capital cost of production of the equipment is included in the model. The detailed model calculating production planning parameters is shown in Figure 112.

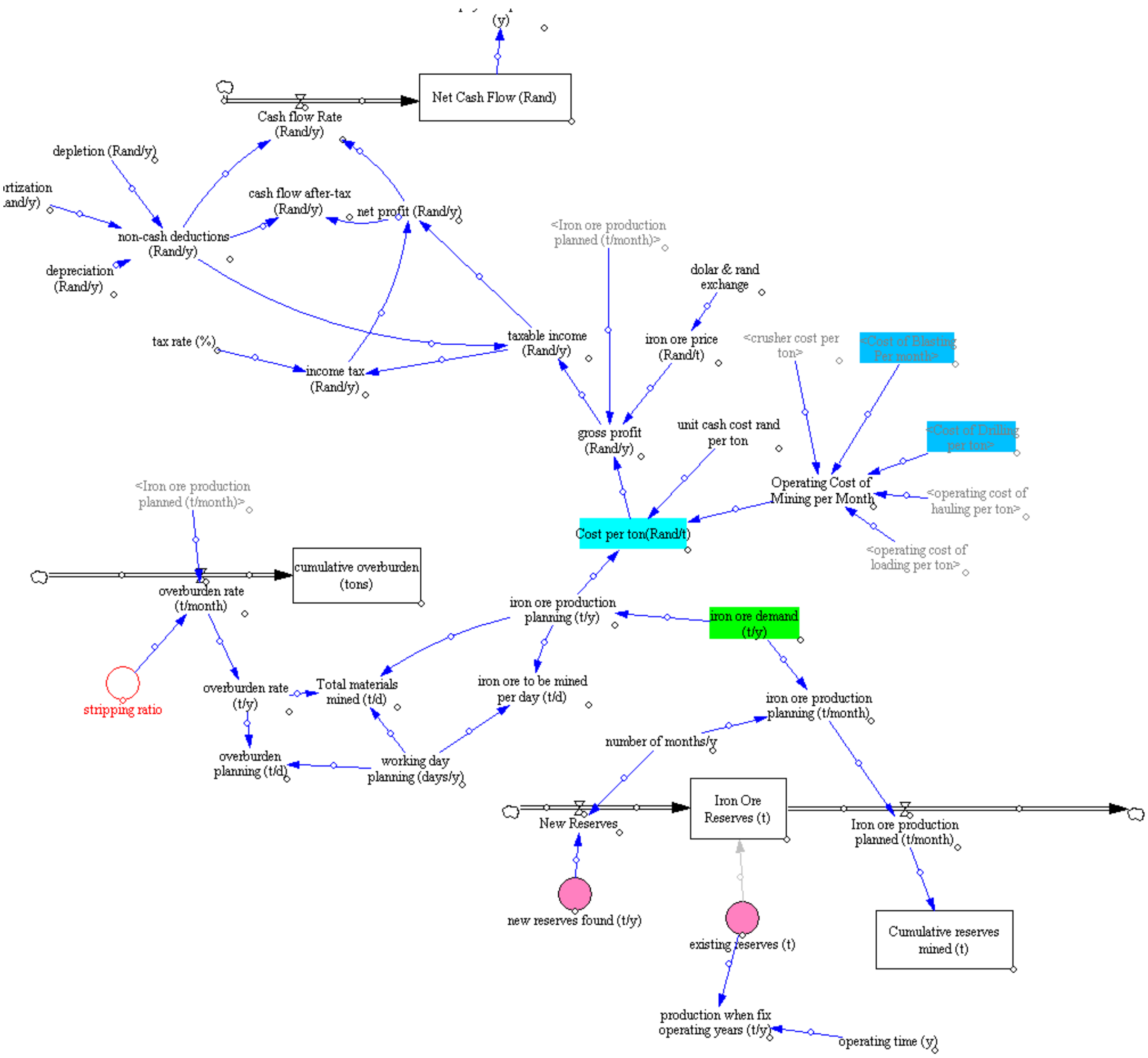


Figure 112 Basic production planning setup

In addition to the basic production setup some extra functionality is added to accommodate big investments, capital costs etc, in this case it is the cost of automation of drill rigs.

The cost items are estimates and does not necessarily reflect the actual costs incurred for the specific mine (Figure 113).

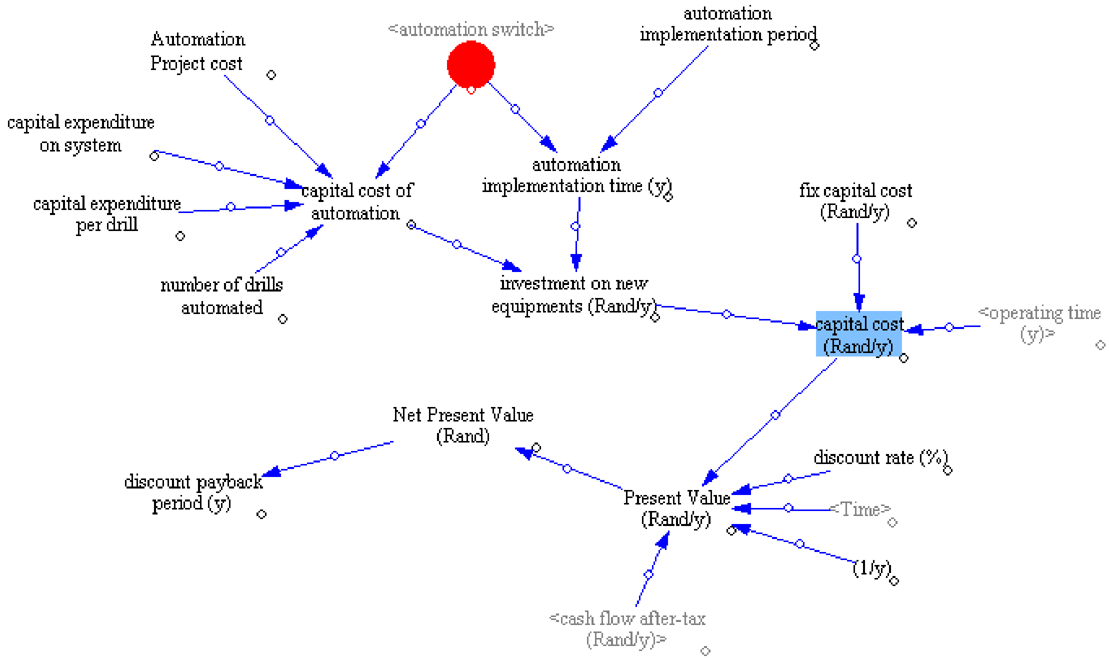


Figure 113 Inclusion of cost of capital in basic profitability calculations

6.1.4 Drilling Productivity

Effective evaluation starts with a complete evaluation of the excavation equipment to be used. There are three basic elements, which must be considered in evaluating a drilling system:

1) Ore production schedules, operating conditions and rock types encountered,
2) Equipment productive capacities: (a) pattern size (b) metric tons of material affected per hole drilled (c) drill production rate (d) drill availability (e) drill utilization (Heinen, 1979).

Firstly, it is necessary to establish the main cost drivers of a drilling process for any realistic cost estimation for a set time interval. Time is the essential component in any dynamic simulation. Planning and scheduling are based on time specific assumptions and predictions over a period of one year. The available time in a year is approximately 350 days. If 3 shifts and 8 hours is scheduled for each drill rig, then 8400 hrs are available per year per drilling rig as was discussed in chapter 5.6 .

During this time-period drill availability will depend on cyclic drilling operation. A drill may not be available due to maintenance, time losses holidays and operator availability, etc.

An SD model as shown in Figure 114 is constructed using utilisation and availability of a drill rig over a calendar year. Some of the items that make up the SD model tie the drill rig productive time to simulation of cyclic unit processes. The time available per drill rig is then incorporated into a bigger model. The calculation of the utilisation formulae that are behind the scenes of the sub model built as shown in Figure 114 is described in the next section.

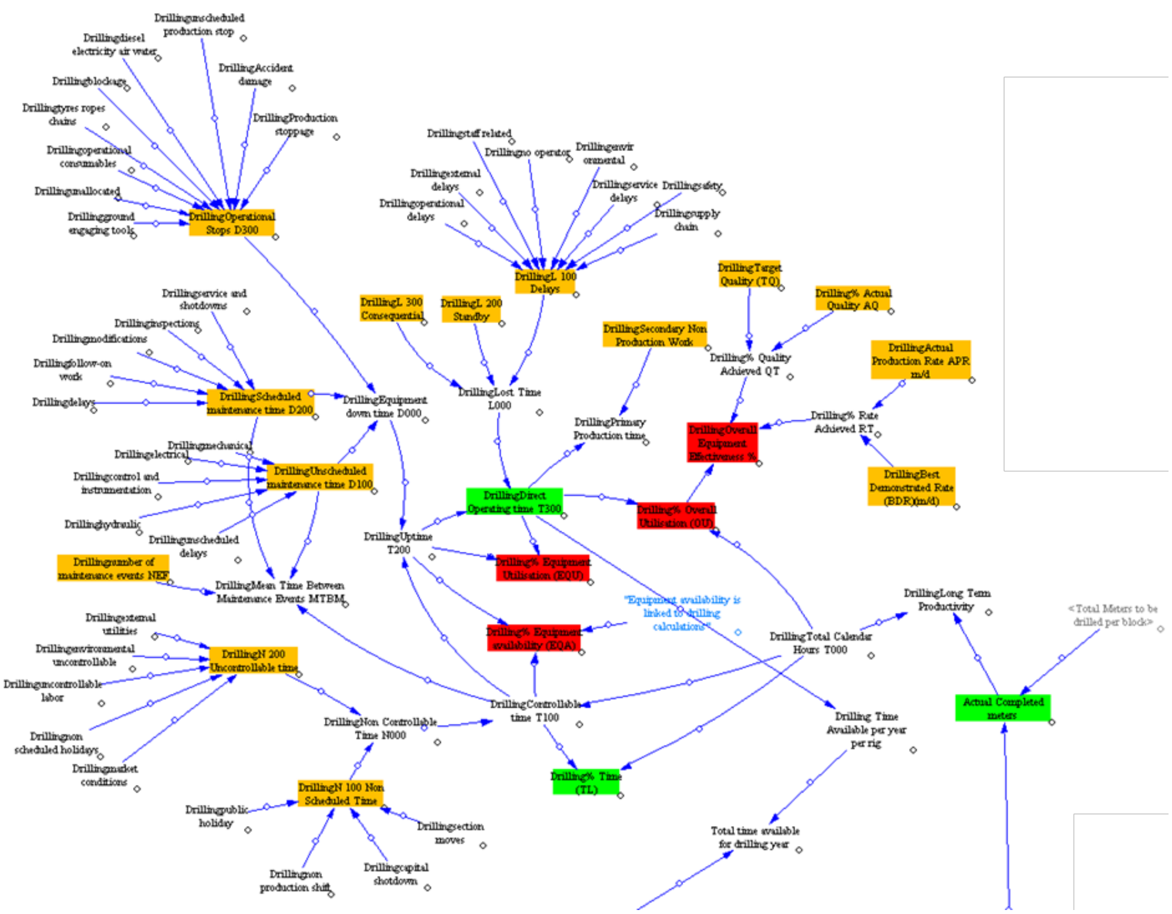


Figure 114 Drilling time losses model applicable to each drill rig

6.1.5 Calculation of utilisation and operational availability of the drilling rigs

Gokhale (2010) defined the term drill availability as follows:

$A= T_c/ (T_c+T_m+T_r)$ **Equation 37**

Where:

A= Blasthole drill availability

B = in hrs. for which drill was available for cyclic drilling operations

T_m= Time in hrs. for which drill was under maintenance

T_r= Time in hrs. for which drill was under repairs

The procedure for determining drill availability and utilization are as follows:

Availability takes "lost time" into account and is calculated as follows:

$$\boldsymbol{Availability = \frac{Net\ Available\ Time - Downtime\ Losses}{Net\ Available\ Time}\ x\ 100}$$ **Equation 38**

Performance takes "speed loss" into account and is calculated as follows:

$$\boldsymbol{Performance = \frac{Operating\ Time - Speed\ Losses}{Operating\ Time}\ x\ 100}$$ **Equation 39**

Quality takes into account "product losses" and is calculated as follows:

$$\boldsymbol{Quality = \frac{Net\ Operating\ Time - Defect\ Losses}{Net\ Operating\ Time}\ x\ 100}$$ **Equation 40**

Drill availability decreases as the drill becomes older. If the availability of drill in the initial period of operations is A_I and in the last year of operation is A_L, then availability reduction factor is

$\mathbf{A_R = (A_I - A_L)/A_I}$ **Equation 41**

Where,

A_R=Availability reduction factor

A_I= Availability of blasthole drill in first year

A_L = Availability of blasthole drill in last year

From these values the time for which a drill is available during its life can be calculated as

$\mathbf{H_C = L_D * M * {}_{AI} * A_R}$ **Equation 42**

Where,

H_C = Cyclic operation hours in drill life

L_D = Drill life in hours

M = Availability multiplier

A_I = Initial availability

A_R = Availability reduction factor

The following table from Gokhale (2010) can be used as a guide in terms of life and availability related factors for different types of rotary blasthole drills.

Table 21 Life and availability related factors for different types of rotary blasthole drills

Size	Size Factor (A)	Power Type	Power Factor (B)	Power Distribution	Power Distribution Factor (C)	Useful Life	Initial Availability A_I=AxBxC	Availability Multiplier if automatic lubrication	Availability Reduction Factor (A_R)
Extra large	0.97	Electric	0.98	Electric	0.97	85000-100000	0.922082	1.06	0.92
Extra large	0.97	Diesel	0.96	Hydraulic	0.95	80000-95000	0.88464	1.06	0.9
Large	0.95	Electric	0.98	Electric	0.97	75000-90000	0.9030700	1.05	0.92
Large	0.95	Electric	0.98	Hydraulic	0.95	70000-90000	0.884450	1.05	0.88
Large	0.95	Diesel	0.96	Hydraulic	0.95	60000-80000	0.866400	1.05	0.85
Medium	0.92	Electric	0.98	Hydraulic	0.95	55000-75000	0.856520	1.04	0.88
Medium	0.92	Diesel	0.96	Hydraulic	0.95	50000-70000	0.839040	1.04	0.85
Small	0.88	Electric	0.98	Hydraulic	0.95	40000-60000	0.816280	1.03	0.85
Small	0.88	Diesel	0.96	Hydraulic	0.95	30000-50000	0.802560	1.03	0.82

Table 22: Six Big Losses that impact OEE (Elevli & Elevli, 2010)

Six Big Loss Category	OEE Loss Category	OEE Factor
Equipment Failure	Downtime Losses	Availability (A)
Setup and Adjustment		
Idling and Minor Stoppages	Speed Losses	Performance (P)
Reduced Speed		
Reduced Yield	Defect Losses	Quality(Q)
Quality Defects		

There are operational restraints such as:

- Shutdown of drills during blasting
- The need for drill fleets to have excess capacity to ensure a steady supply of blasted material for loading
- Scheduling problems
- Unscheduled maintenance failure of the loading equipment

It has been reported that operational restraints can vary from 5 to 40% of the available operating time. Long drill moves, and other major interruptions are defined as occurrences when the drill must be moved far enough on one level to require additional trail cable such as in the case of electric drills. This depends on the block size being drilled, say 200 meters. Then, the drill mast must be lowered, or the drill is moved to a new level. These are all time-consuming activities and needs to be accounted for in a production cycle. The frequency of these events depends on the size and nature of the mine.

For being consistent with a real operation the operational efficiency model is constructed using the mine specific performance reporting tool.

Some examples of drill availability calculations are shown below (Table 23). Note that the numbers are just an estimate and need to be changed based on the operation being modelled.

Table 23 Factors affecting drill availability and cycle time (Heinen, 1979)

Drill Availability Calculation	**Hr.**	**Days**
Total calendar time	8760	365
Less holidays per year	216	9
Possible available time	8544	356
Less M&R outages	1440	60
Available operating time (equipment availability)	7104	296
Less operational restrictions	624	26
Less long drill moves and other major interruptions	216	9
Less personnel time		
Travel time	432	18
Lunch	432	18

Other	72	3
Less other non-drilling time		
Lubrication and inspection	288	12
Short moves	72	3
Running repairs	216	9
Other	216	9
Net drilling time	4536	189

Example calculations:

% Availability = [(356-60)/356] x 100%=83%

% Utilization = [(296-26-9-39-33)/296] x 100%=64%

% Operated total time = 83% x 64%=53%

Operational hr. per shift = [9296-26-9-39-33)/296] 8 hr. /shift = 5.1 hr. /shift

Table 24 is an indication of the required number of drills based on the drill bit diameter. The reason for the drastic differences is because of the drill pattern expansion requirement with larger diameter holes and these decreases drilling density per square meter.

Table 24 Actual number of drills needed. Economic analysis (Heinen, 1979)

Drill bit size	*Min no of drills needed*	*Additional long drill moves time needed/ available op day*	*Total increase in long drill moves, hr.*	*Increase in long drill moves/drill*	*Adjusted net drilling time/drill*	*No of Drills Needed*
229 mm (9 inch)	4	0	0	0	4336	4
270 mm (105/8 inch)	2	4	1184	592	3944	3
311 mm (12¼ inch)	2	4	1184	592	3944	2
381 mm (15 inch)	1	6	1776	1776	2760	2

When a blasthole drill is to be used for 350 days in a year for 3 shifts of 8 hr. each, it is calculated as 350*3*8=8400 hrs. During this time period a blasthole drill can be available for cyclic drilling operations, or not available as it is under maintenance or undergoing repairs. The term availability is often used in this context and defined as

$A=T_c/(T_c+T_m+T_R)$ **Equation 43**

Where,

A= blasthole drill availability

T_c= Time in hrs. for which drill was available for cyclic drilling operations

T_m= Time in hrs. for which drill was under maintenance

T_R= Time in hrs. which drill was under repairs

A rotary blasthole drill rig works in cyclic activities as listed by Bhalchandra & Gokhale (2010) in Table 25

Table 25 Activities in a blasthole drilling cycle (Bhalchandra & Gokhale, 2010)

	Cyclic activity	Typical Time in Sec
1	Moving from one blast-hole location to the other	45
2	Lifting the blasthole drill by hydraulic jacks and level it	30
3	Lowering the bit and starting drilling at low speed	20
4	Drilling the complete depth of the blasthole	Calculation
5	Uncoupling the drill head from the drill pipe	25
6	Moving up the drill head to the top of the mast	30
7	Positioning the pipe changer in drill string alignment	25
8	Coupling the drill head to the new drill pipe	20
9	Removing the pipe changer from the drill string alignment	15
10	Coupling the new drill pipe to the lower drill pipe	25
11	Attaching the desired number of drill pipes	Calculation
12	Moving up the drill head + drill pipes by one drill pipe length	40
13	Moving up the complete drill string	Calculation
14	Uncoupling the drill pipe from lower drill pipe	30
15	Positioning the pipe changer	30
16	Uncoupling the drill head from drill pipe	25
17	Removing the pipe changer	18
18	Lowering the drill head to the bottom of the mast	35
19	Coupling the drill head to the lower drill pipe	25
20	Detaching the desired number of drill pipes	Calculation
21	Lowering the drill on to the crawler base	25
22	Complete cyclic operation	Calculation

Drilling time= Drill depth/Penetration rate **Equation *44***

The time required for drilling from hole to hole depends on burden and spacing, therefore blasting pattern's spacing is recommended for calculation of tramming times and distances for calculation of the hole to hole cycle time.

Time required=Distance/Tramming speed **Equation 45**

Once at the location the alignment of hydraulic jacks needs to be done which can take up to a minute.

Change of a bit is required every so many meters. This number can be obtained from on-site records per specific drill bit and diameter. Drill bit changing time is typically 4 minutes. Mine data can be used for the frequency of bit changes, or it can be calculated from bit consumption per 100 meters according to the manufacturer specifications.

Operating Rate is then calculated based on total drilled meters and total drill time (m/min) then:

Hourly production (m/hr)= Production efficiency X operating rate **Equation 46**

Production efficiency of an operator over a shift work depends on the operator and for the experienced drillers it is estimated at 50 min/hr, otherwise 40 min/hr actively drilling. This depends on site and training effectiveness. Drilling production should match hauling and loading, therefore accurate estimates of tonnage for a drill block should be made based on the available rock types, densities, as well as swell factor of the rock.

These discussions around drill rig planning and cycle times are in support of the constructed model as shown Figure 115.

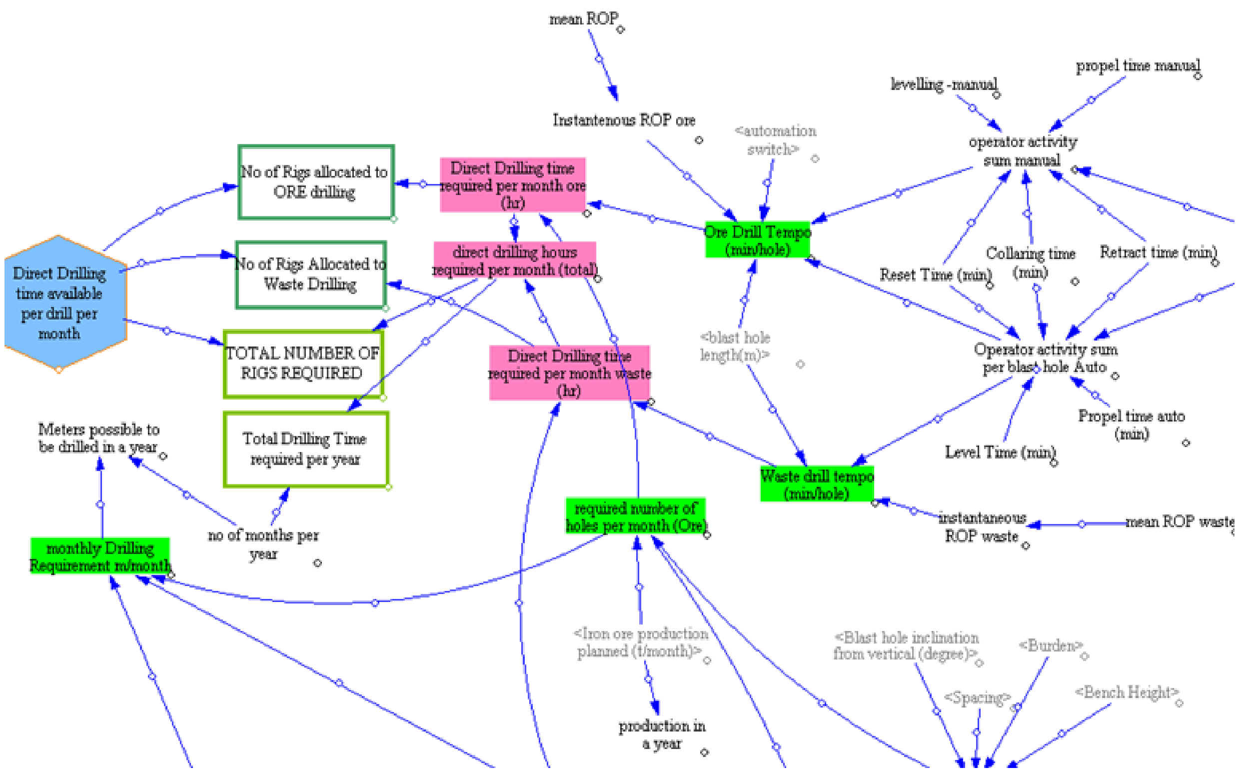


Figure 115 Determination of monthly and yearly drilling achiavable based on cycles

The drill pattern obviously depends on the blasting parameters set and they are rule based design parameters. All the blast design parameters that are used in this model are typical values for a large iron ore mine. The blasting design setup as well as requirements of explosives are constructed with the formulae in the background in the model below.

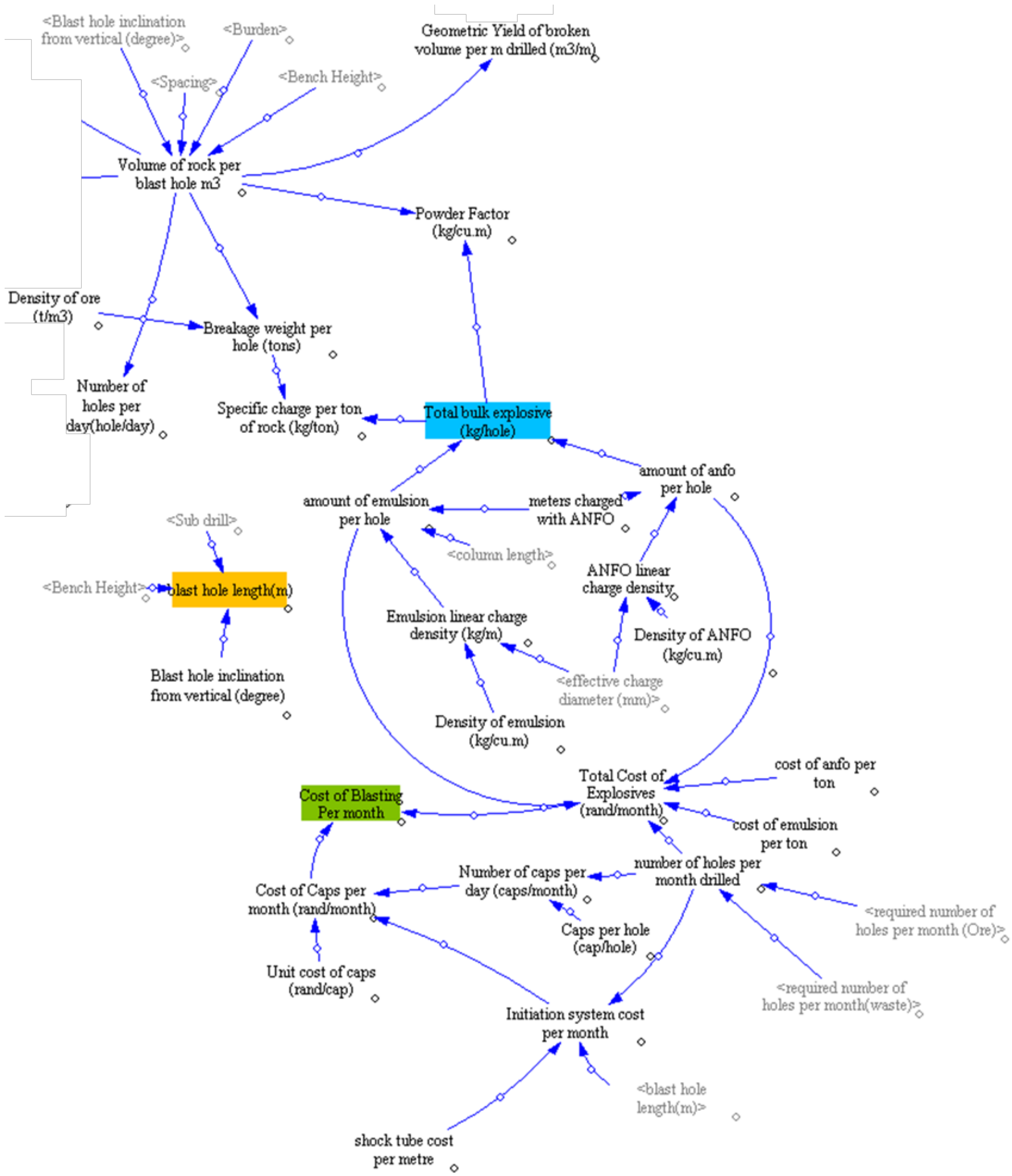


Figure 116 Blast design parameters and explosives amount determination

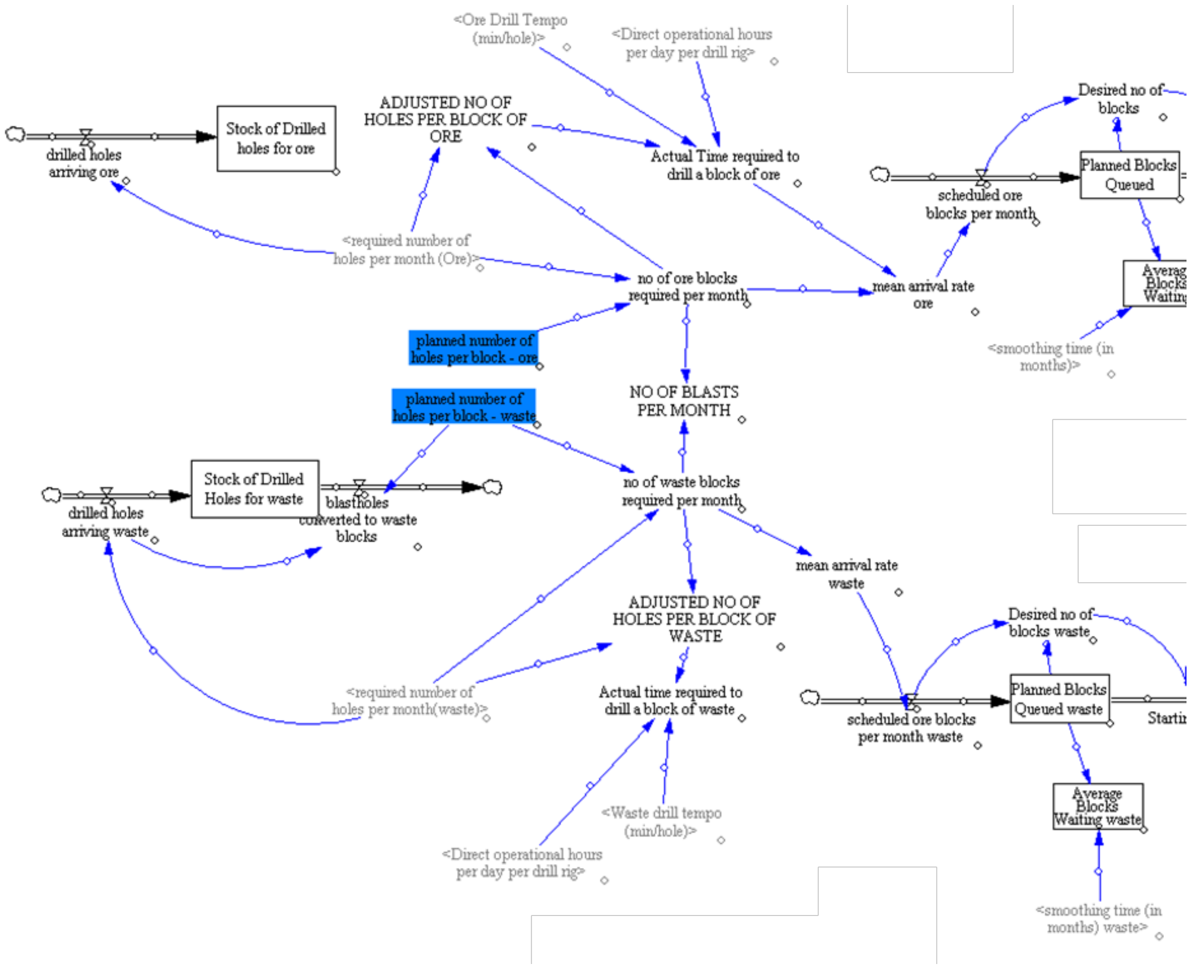


Figure 117 Interface between drill capacity and discrete drill block scheduling

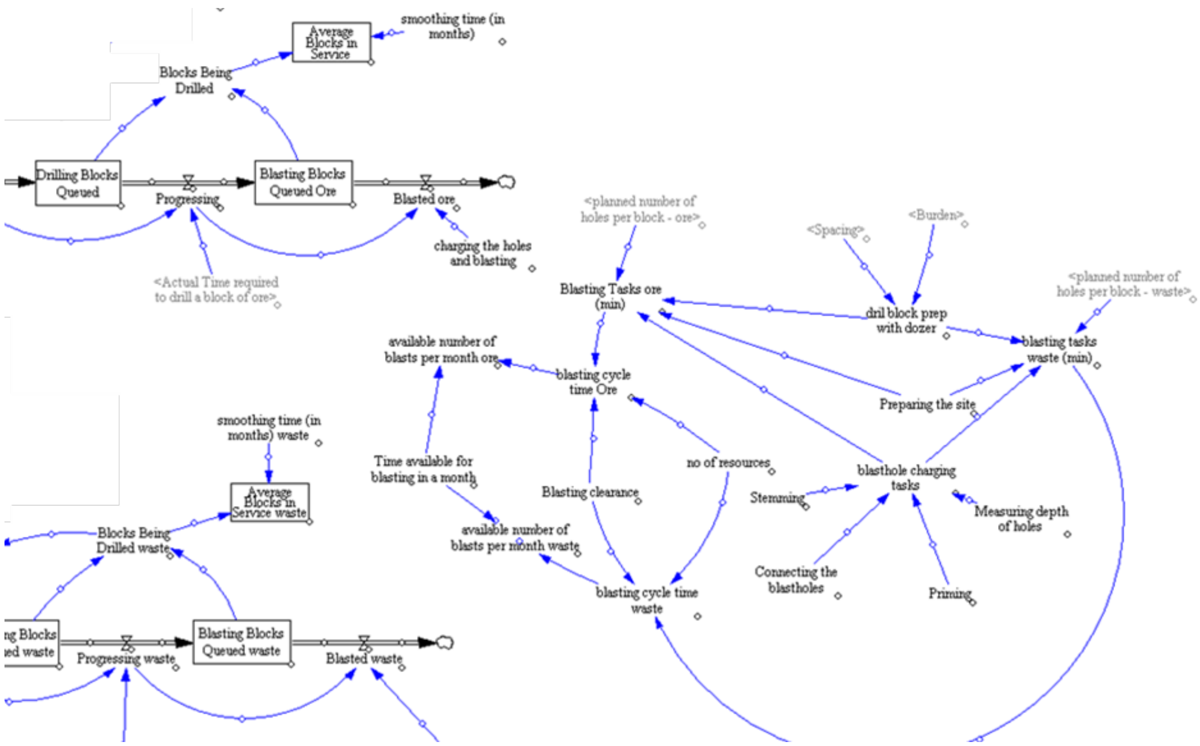


Figure 118 Interface between blasting capacity and cycle times to the scheduled block blasting times

Cost of Power

Most of the rotary blasthole drills are powered by diesel engines. The maximum power output of such engines varies over a wide range from 175 to 1129 kW. 75% of maximum power outage is normally used. If a blasthole drill has 250 Kw it is likely to use 48 l of diesel (Bhalchandra and Gokhale, 2010).

For a diesel powered drill rig a sub model is created that is specific to the drill rig type. The data normally comes from the manufacturers of the type of drill rig for the constructed model.

It is a well-known fact that cycle time increase means increased fuel consumption therefore mining costs. This effect should be realized in fuel consumption values as an outcome of the simulation. Activity based fuel consumption is determined per drill rig is shown in Figure 119.

Each rig has different operating times and various levels of fuel consumption. Typical activities during drill operation include pipe handling, drilling, propelling, idling, levelling. They are indicated in blue. If the total amount spent on each level of activity is known, then the fuel consumption of a certain rig type can be normalized to per hour level.

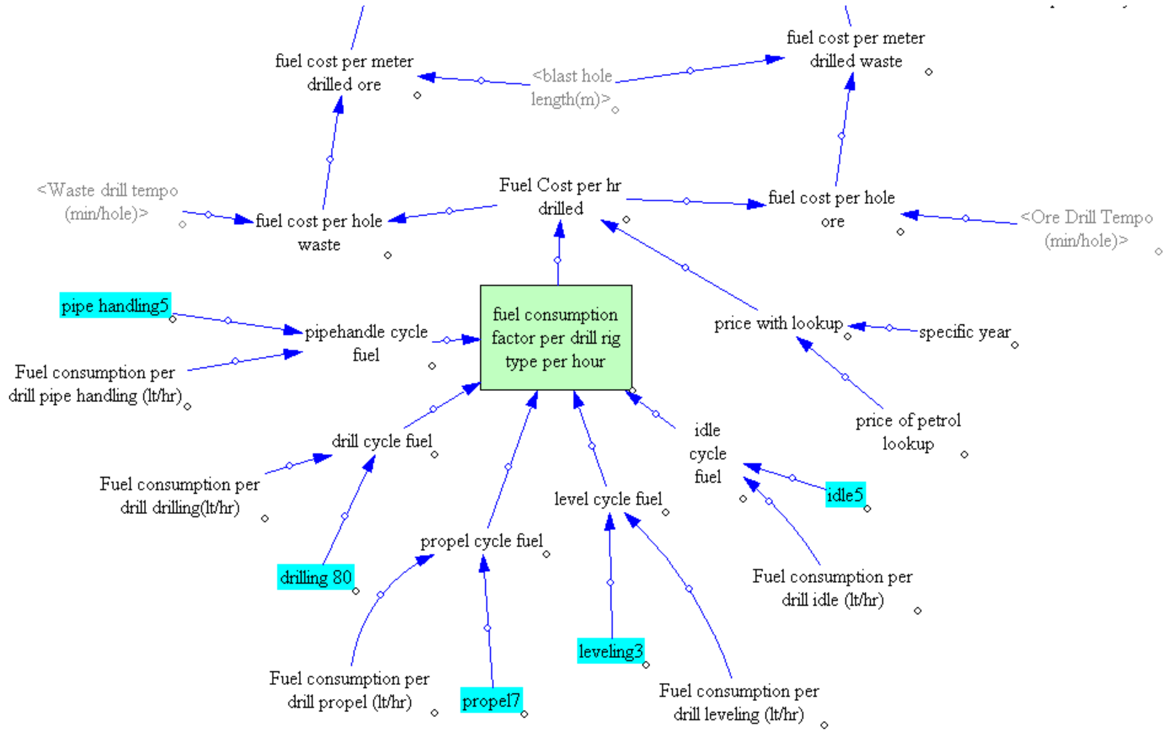


Figure 119 Establishing activity based fuel consumption per rig type.

Drillability

Drillability, drilling rate and drill performance are some of the variables that are part of the input for operational scheduling and planning. Determination of accurate prediction of drilling rate and drill tempo is critical for evaluation of the effect of drill cycle times. This section will attempt to do a review of current theory and practice and define best prediction methods to be incorporated into a drill scheduling programme for a surface mine.

Drill Specific Energy

Drill specific energy can be evaluated by the formula below as accepted comparative measure for rock mass hardness (Schunnesson and Mozaffari, 2009).

SE=(F/A)+(2π/A)+(NT/P) **Equation 47**

The formula in Eq 46 is expanded with the meaning of the terms in eq 47.

Specific Energy= (Feed Force/Drill hole Cross Section)+(2π/Drill Hole Area)

+(Rotational speed x Torque/Penetration rate) **Equation 48**

This formula although not used in this thesis modelling can be used where mine data is not available. The values used in this thesis are real values from an actual mine and therefore prediction was not necessary. The fuel consumption of the specific rig was entered into the model.

6.1.6 Cost of Drilling and Blasting

The cost items for drilling and blasting are:

- Cost of ground engaging tools (G.E.T.) consumables
- Cost of fuel and electricity
- Cos of explosives and initiation systems
- Cost of labour
- Cost of capital

Table 26 Diesel and petrol average price per year (aa.co.za)

YEAR	PETROL	DIESEL
2021	16.15	13
2020	15.50	13.5
2019	15.77	14
2018	15.42	13.72
2017	13.62	12.10
2016	12.59	10.5
2015	12.41	10.5
2014	13.89	12.37
2013	12.92	11.80

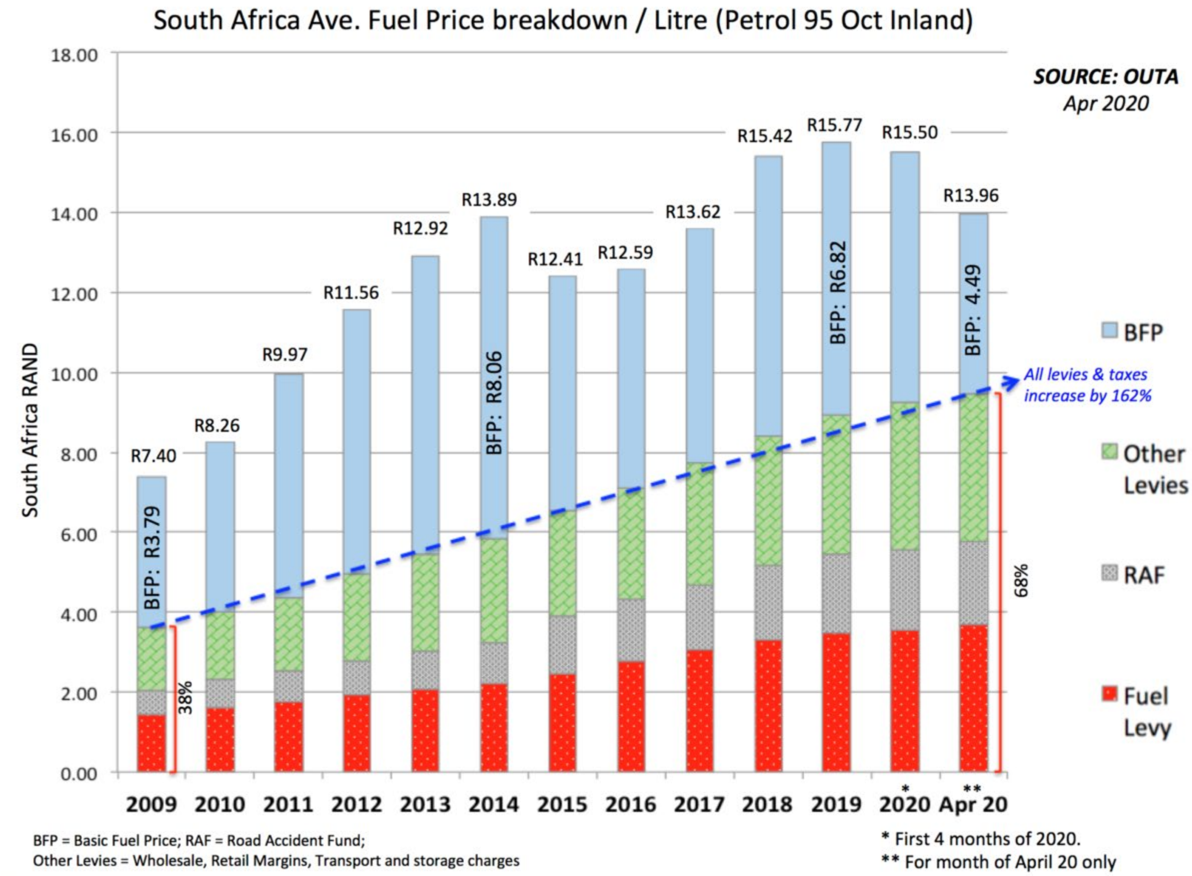


Figure 120 South Africa Average Fuel Price Breakdown

It is easier to calculate the cost of explosives which is based on the number of holes drilled and the distances between blastholes. The calculation of drilling cost requires a little planning to construct in the model. The drill rig has some variability in terms of times of engaging the rock and the rates at which it drills several types of rocks creates variability in the fuel consumption, drill cycle times and the G.E.T consumption. Most of them are incorporated into the sub model constructed for this purpose except the rock types related rates of drilling. For simplicity only two generalized rock types are used, but more can be added if required.

Cost of drilling is calculated via the sub model developed as seen in Figure 121.

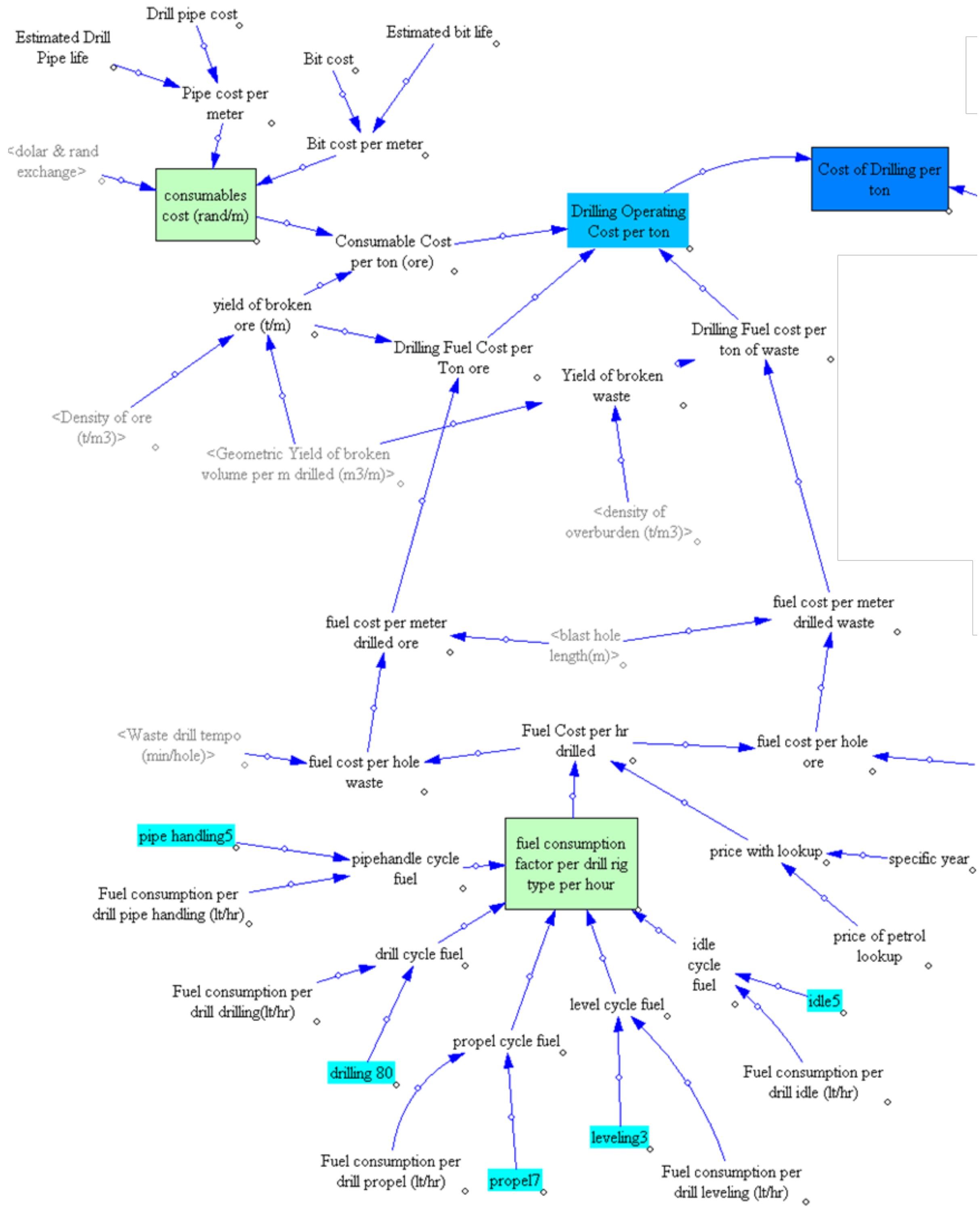


Figure 121 Drill costing calculations based on different levels of activities for each blast hole.

6.2 LOADING AND HAULING

6.2.1 Excavator/Loader Productivity

The impact of connected processes in an open pit mine is undeniably high and loading and hauling is one of them that is affected by the drilling and blasting activities. An overview of load and haul process and relevant data that can be used in the newly developed SD model is discussed in this section.

Loader performance is defined as the hypothetical absolute production of any loader (Hardy, 2007). Hardy's research discusses loader performance and productivity and mentions that mining research should not be done in isolation of other processes. In addition, he recognizes the richness and variability of data collected in mines makes a holistic research more difficult, therefore empirical approaches cannot be avoided.

Production cost is assumed to be a complex function of the Equipment Size Sensitive Variables (ESSV) Production cost is therefore a function of equipment size and each related cost item. It is required to identify the equipment size sensitive variables and collecting data regarding each equipment. The data related to cost are MTBF, MTTR, delays, standbys, production data and costs. Since loader and trucks are affecting each other's productivity, they should not be isolated from each other.

Beyoglu measured that P&H 4100C shovel, with a dipper capacity of 81.7 tons, loads a CAT793 truck with a capacity of 227 tons in three to five cycles depending on the diggability and fragmentation of the muckpile. (2016).

Elevli and Elevli reports a typical OEE (Operating Efficiency Estimates) for a shovel as seen in Table 27. They calculated OEE based on calendar which is 37% and based on loading time which is 47%. They have realized that there is a discrepancy between the two methods and that loading time-based approach but for almost the same amount of tonnage. Their conclusion is that loading time-based approach overestimates. Every second needs to be accounted for calculating correct OEE if time-based loading approach is used.

Table 27 Time lengths for a shovel operation

Item	Description	Time (hours/month)
Total Time	24 hours/dayx30days/month	720
Non-scheduled Time	2 days off	48
Scheduled maintenance	3 days	72
Unscheduled maintenance	breakdowns	97
Setup and Adjustment	0.5 hours/shift	45
Idle time	0.6 hrs/shift	54
Truck waiting time	0.5 hrs/shift	45
Job Conditions	Equipment did not work due to bad weather	25
Speed Loss	0.5 hrs/shift	45
Propel Time	4 moves/month, 2 hrs per move	8
Quantity Loss	Filling factor (87%)	
Shovel Bucket Capacity	15 m3	
Ideal Production	1.5 bucket/min	

The calculations in Table 27and Table 28demonstrated by Elevli regarding OEE show that for the same production amount of 359.4 meter cube of rock, the calendar time approach OEE is 0.37 and loading based OEE is 0.44. Therefore, one needs to be careful when drawing conclusions based on OEE calculated with different methods. What is more important is the principal outcome which is total production per year. Both methods result in the same production with both methods. The calendar time-based approach is being used for this research for being consistent.

Table 28 OEE estimation of Shovel (Elevli and Elevli,2010)

	Calendar time-based approach	Loading time-based approach
Total Time	720 hrs	(720-48-72) =600 hrs
Availability	AAT/TT= (720-(48+72+97+45+54+45))/720 =0.5	AAT/TT= (600-(97+45+54+45))/600 =0.6
Performance	NPT/AAT=(361-(25+45+8)/361=0.85	
Quality	0.87	
OEE	Availability x Performance x Quality	
	0.5x0.85x0.87	0.6x0.85x0.87
OEE	0.3698	0.4437
Total Production	(720x60) x1.5x15x0.37 =359 445 m^3	(600x60) x1.5x15x0.47 =359 397 m^3

Hauling cannot be evaluated isolated from loading. Simulations of real conditions need to classify measurements based on total hauling unit properties, operator efficiency, hauling cycle time, bench conditions, traffic and method used during loading. Hauling is the highest cost component in the mining operation therefore, it is necessary to breakdown each cost component and relationship between these attributes.

Some of the items that impact the fuel consumption of loading and hauling will be linked to inferior drilling and blasting efforts. Therefore, cost items affecting the high fuel consumption is to be modelled based on the breakdown of the listed activities of loading and hauling. For example, trucks queueing at the loader site consumes petrol on idle. Some of that wastage can be directly attributed to the bad fragmentation due to inferior drilling and blasting quality.

Blast fragmentation has two key effects on loading and hauling performance in a mining operation due to digging time and bucket payload which is the ratio of void and fill factor. Improved fragmentation may cut down the time taken to haul by preventing unnecessary queues at the shovel. If the time losses at the shovel location can be prevented loading equipment will not be idle at the production face. Energy losses while trucks waiting is another downside of the fragmentation distribution related hard digging conditions. Presence of boulders in the muckpile will cause further time delays. This may cause loaders as well as haulers to wait idle until the boulder is removed from the production face. There seems to be a direct relationship between smaller fragmentation and increased tonnage and individual dipper and hauler cycle.

The digging time against mean fragment size relationship is mine specific. A relationship obtained at a quarry where a digging time study was conducted by Jethro (2016) shown in equation 48.

$$\mathbf{X_{op} = (0.15 - 0.2)B_c^{\frac{1}{3}}} \quad \textbf{Equation 49}$$

He mentions that decreasing mean fragment size by 10 cm means gaining 2 seconds of loading time at each pass of the excavator.

Excavator and truck matching is an extremely important factor for the relationship to be built in. Therefore, Table 29 has been used to create an automatic loading parameter selection simulation by the author.

This section of the model also has built in lookup tables to select the number of passes required per truck payload selected as well as the loader selected with matching swing times. See Figure 122.

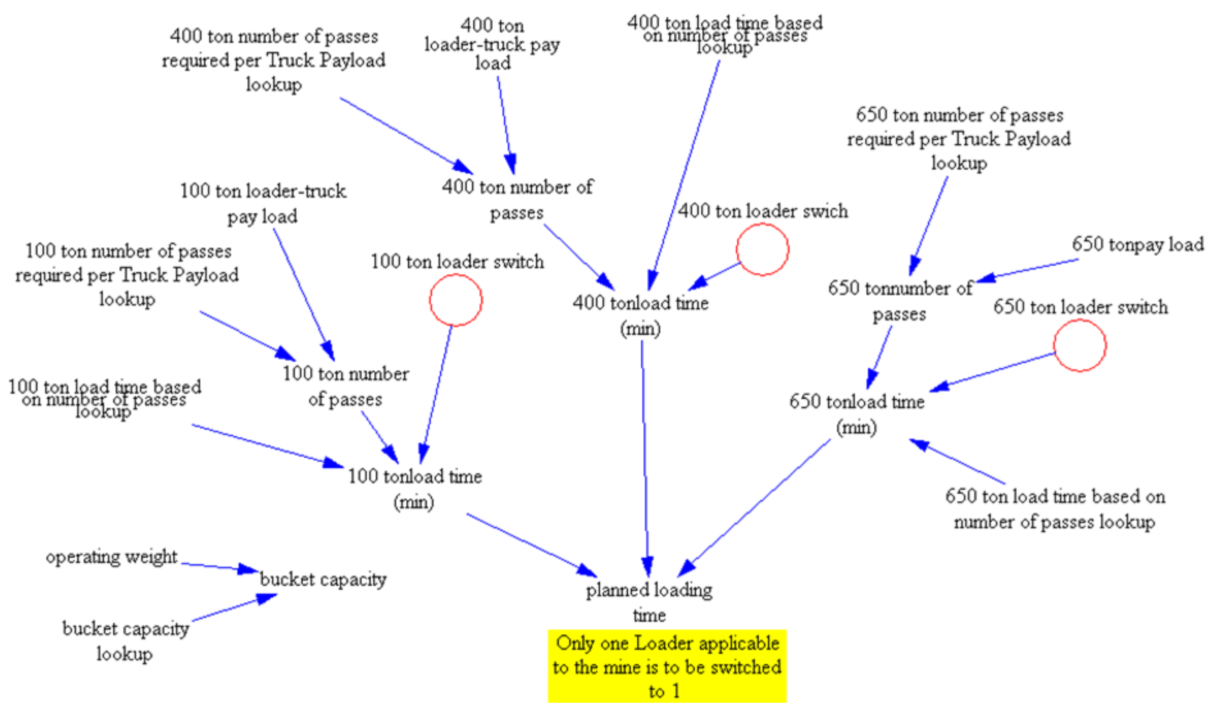


Figure 122 Automatic loader assignment matching the truck payload

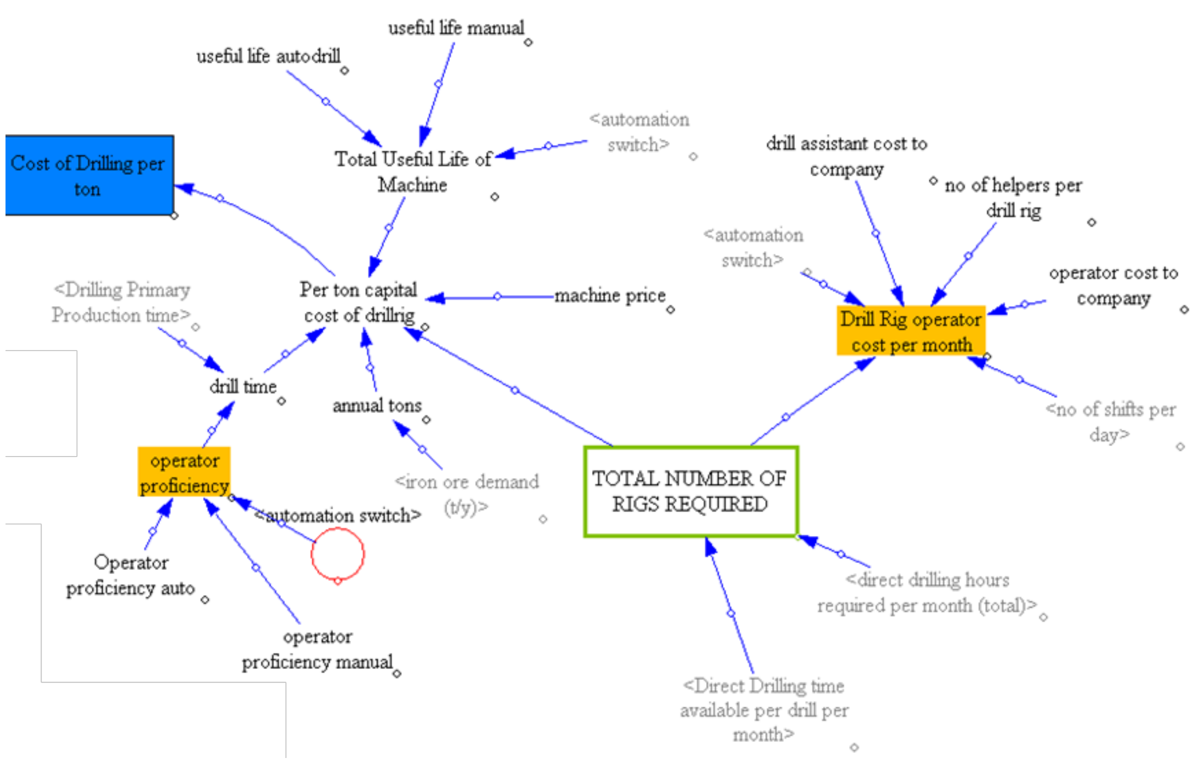


Figure 123 Modelling of operator proficiency and the cost associated with manual drilling

It was previously mentioned that an experienced operator is effective for 50 minutes in an hour long shift, meaning 83% effectively uses the time. For an inexperienced operator this number can be as low as 40 minutes per hour which is effectively 66% of the time. What is of significance is with automated drill rigs, the drill can drill with one touch and will be effectively using almost 100 % of the time available for drilling. An additional switch is included here for the SD user to choose the option of automation when studying the causality (Figure 123).

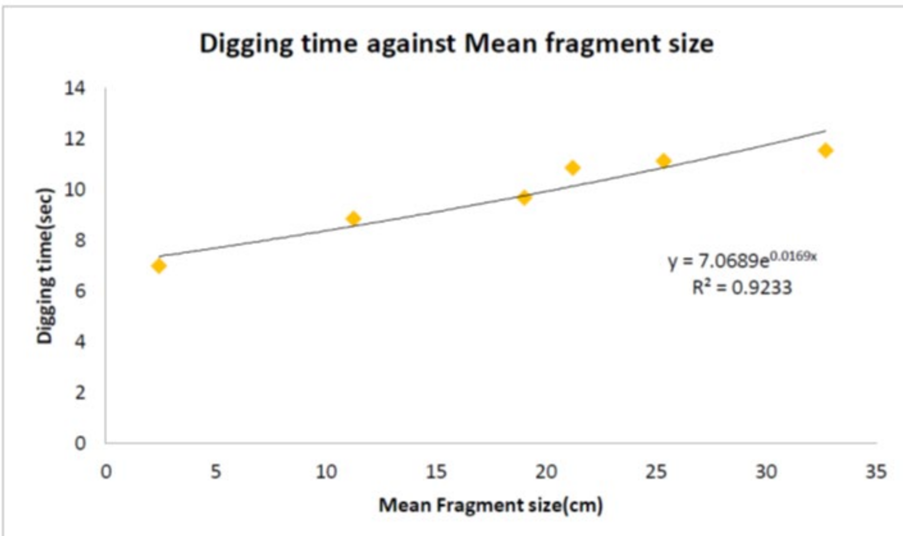


Figure 124 Mean fragment size versus digging time (Jethro, 2016)

Table 29 Effects of Excavator/Truck Matching on Load and Haul Costs (Gregory,2003)

Operating Weight	100			400			650		
Bucket Capacity	6.5			20			34		
Truck Payload (tonnes)	Number of passes	Load Time (mins	Cost Index (%)	Number of passes	Load Time (mins	Cost Index (%)	Number of passes	Load Time (mins	Cost Index (%)
49	4	2.13	200						
91	7	3.48	144	2	1.25	125	2	1.25	135
146	12	5.73	130	3	1.72	114	3	1.72	115
187	15	7.08	126	4	2.18	104	3	1.72	102
230	18	8.43	134	5	2.65	101	4	2.18	100
353				6	4.05	110	6	3.12	106

Age and specifications of the excavating machines operator skills, condition of muckpile, machine breakdowns, excavator marching etc are all site specific. There is one generalized approach found in literature is by Rzhevsky (1995) (cited in Choudhgary, 2019), where, the relationship of the fragmentation to that of excavator bucket size is reported as follows.

$$X_{op} = (0.15 - 0.2)B_c^{\frac{1}{3}}$$ **Equation 50**

Where X_{op} – optimum fragment, m

B_c - nominal bucket size capacity, m^3

Also, the excavator cycle time is affected by muckpile angle and mean fragmentation size. This effect can be built into the model in the form of a lookup table. One tends to use the average fragmentation size obtained per muckpile to say that cycle times will be within range. However, statistics here will be misleading since it does not take into account the oversize within a muckpile. This should be somehow calculated per sieve size range per muckpile. This also means that oversize rocks are to be determined based on the bucket size used at the mine. If a bucket cannot handle oversized rocks, they need to be either blasted or pecked with a hydraulic pecker.

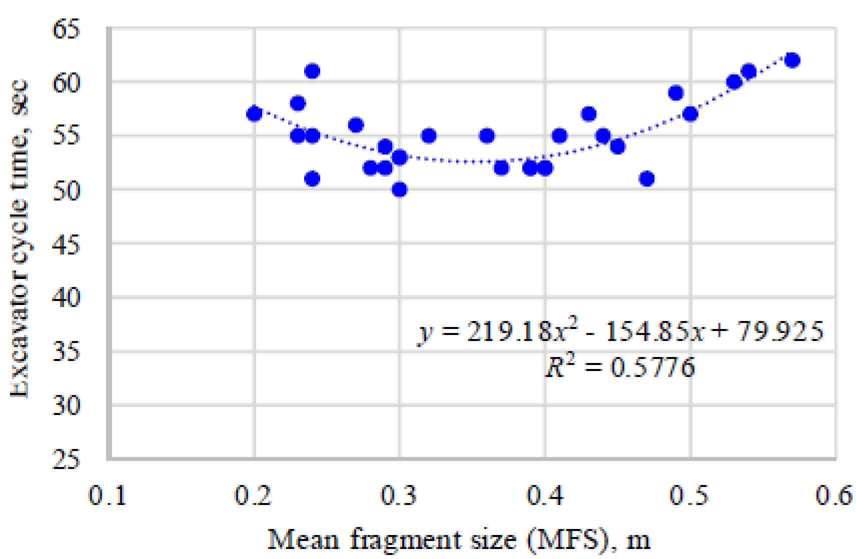


Figure 125 Effect of fragment size on excavator cycle time (Choudhary, 2019)

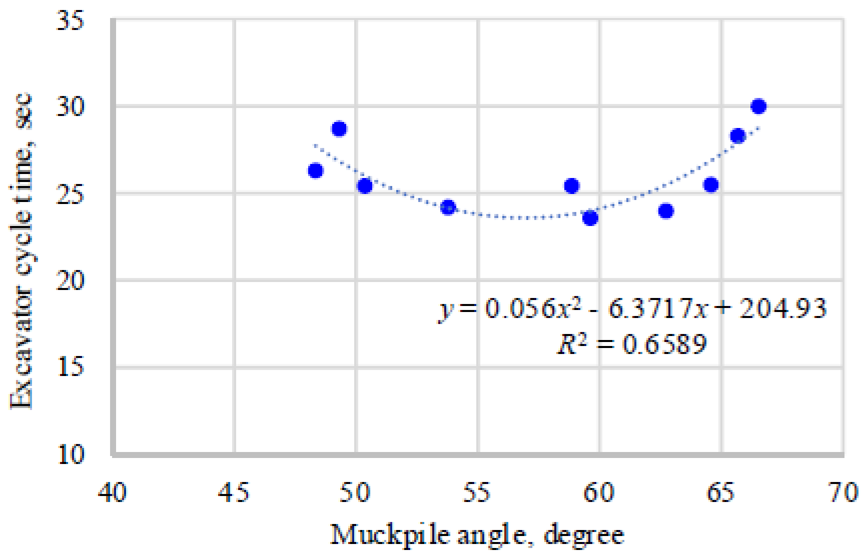


Figure 126 Effect of muckpile angle on excavator cycle time (Choudhary, 2019)

Figure 125 and Figure 126 provide information on the cycle time calculations and it is clear that approximately 5 second is added when the muckpile is 5 degrees off from the optimum angle of 55 and similarly 5 seconds should be added to the cycle time if the optimum fragment size range moves out by about 10 cm up or down.

It is worth noting that hauling cost trend has a relationship with the truck payload. Assuming a unit cost for a small payload, as the truck pay load increases the haul cost index reduces to around 40 % of the original cost item. The relationship of the cost index can be seen in Figure 128.

Matching of the loading equipment to haulers and cycle times are two critical loading and hauling parameters. Loading equipment cost per cubic meter changes based on the size of the truck payload of which the relationship is shown in the graph by Runge (1998). The cost reductions are normally associated with the concentration of drilling operations with fewer drill rig moves and prolonged access for maintenance, hole charging, etc. Fewer shovel moves, easier and better supervision, reduced consumption of explosives per ton of broken ore and better fragmentation are the direct results of the large-scale blasts. This affect can be demonstrated by setting the number of holes per block being drilled.

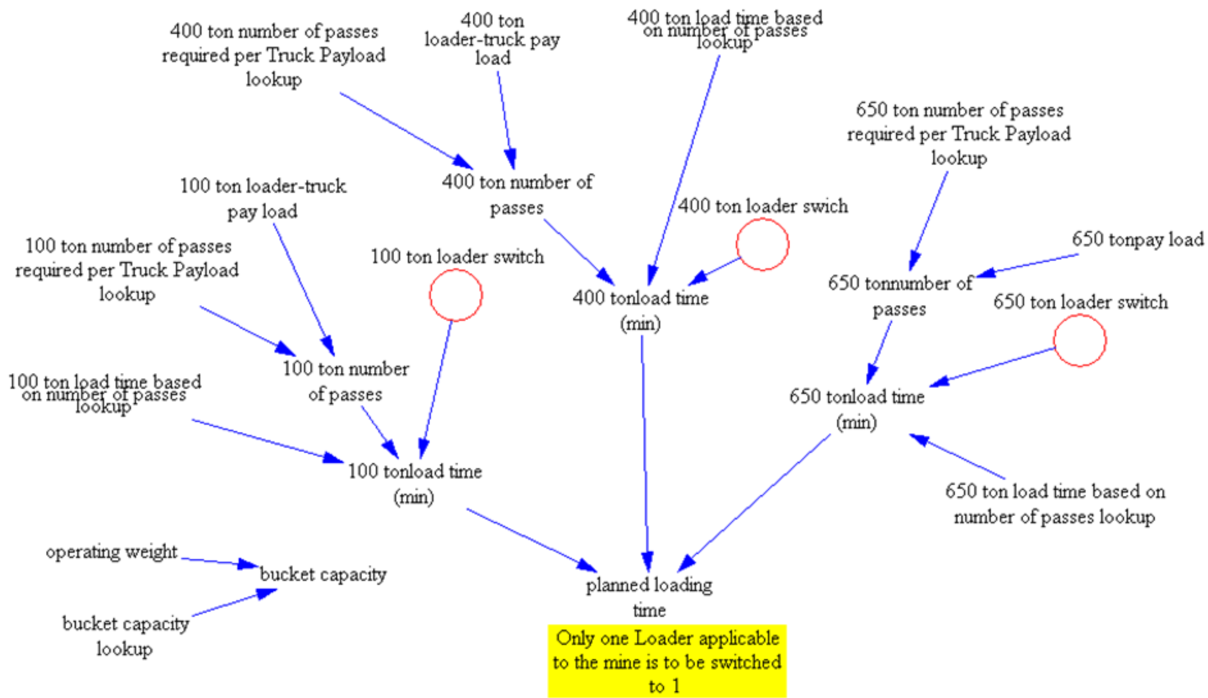


Figure 127 Truck matching model for correct loader size based on Gregory (2003)-see relevant table (Table 29)

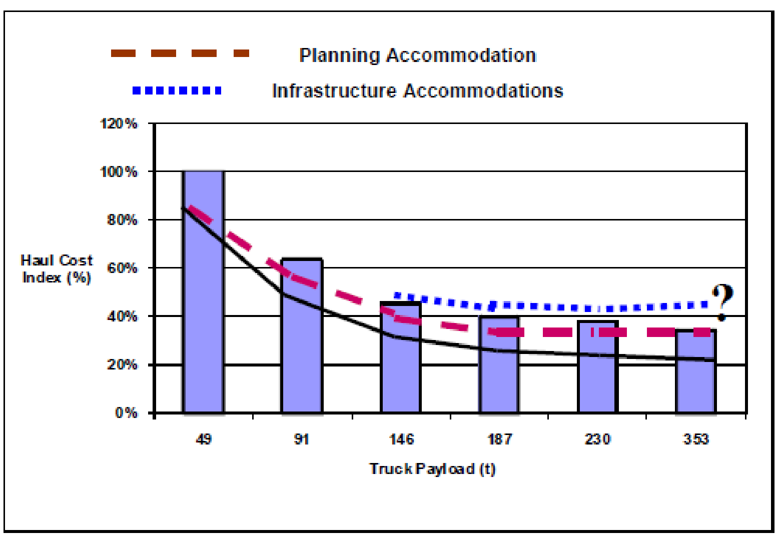


Figure 128 Hauling cost trend with Truck Scale (Gregory, 2003)

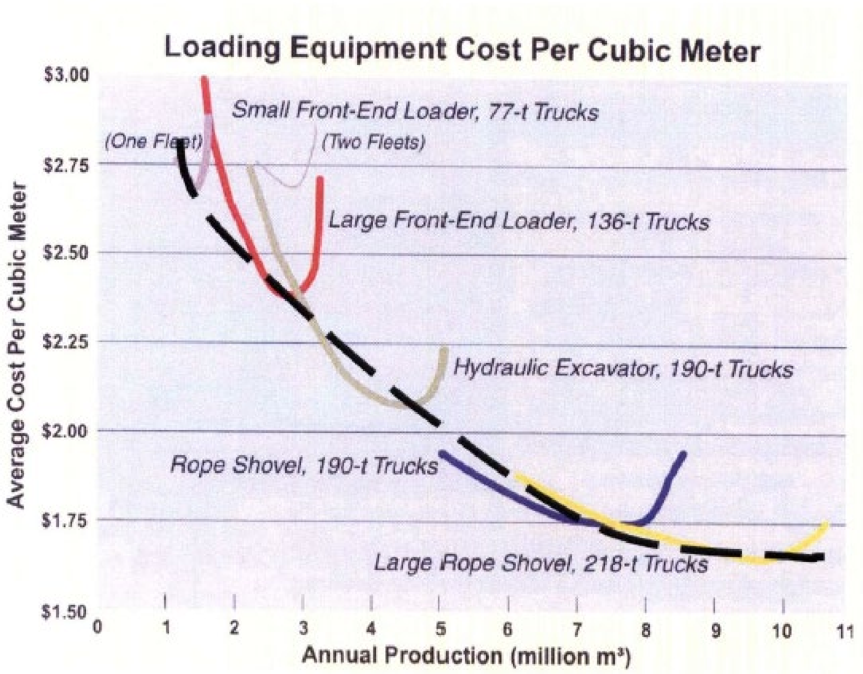


Figure 129 Loading rates-average cost curves for different production systems (Gregory, 2003)

Blast fragmentation size distribution is measured in t/hr and depends on:

- Loader-truck matching
- Truck cycle time
- Truck assignment

Production depends on:

- Physical availability
- Use of availability
- Operator efficiency

Productivity and production estimates of loaders depend on the following factors as listed in Table 30.

Table 30 Factors effecting the productivity and production estimates of loaders (Brunton, 2003)

Swell factor	%	Bucket/dipper rated capacity	m3	Truck production per period
Loose density	t/m3	Bucket/dipper rated capacity	m3	Effective time
Bucket/dipper rated capacity	yd^3	Idle time	%	No of trucks loaded per hr.
Bucket/dipper fill factor – theory	%	Loader productivity		Truck payload - real
Bucket/dipper fill factor – theory	t	Scheduled time per period		Bucket/dipper fill factor – real
Truck capacity	st	Physical availability		Load time 1st pass
Truck capacity	t	Use of availability		Load time subsequent pass
Bowl fill factor	%	Operator efficiency		Truck exchange time
Truck payload – real	t	Effective time		Losses in loading cycle
No passes – theory	No	Loader production per period		Total loading time
No passes – practice	No	Bucket/dipper rated capacity		Other time losses
Bucket/dipper fill factor – real	%	In situ material density		Haul/dump/return/queue
Load time first pass	sec	Swell factor	%	Truck cycle time
Load time subsequent pass	sec	Loose density	t/m3	Truck productivity
Truck exchange time	sec	Bucket/dipper rated capacity		Scheduled time per period
Losses in loading cycle	sec	Bucket/dipper fill factor – theory		Physical availability
Total loading time	sec	Truck capacity		Use of availability
		Bowl fill factor		Operator efficiency

The basic excavator performance indicators are dig, swing, dump and return times. The dig time is the most sensitive component of the loading cycle to variation in muckpile characteristics. Dump time and the time for the operator to spot the next digging location are also affected (Brunton at all, 2003). Excavator productivity is measured in t/hr. and depends on:

- Loader-truck matching
- Loading cycle time
- Truck assignment (loader saturation=maximum loader productivity)

6.2.2 Swell Factor and Loading Efficiency

Swell factor can be determined by dividing post blast volume by preblast volume. If swell factor for each blast is determined individually variations in shape and volume can be eliminated. This data may not be available for the purposes of this study. Therefore, some estimates based on rock types that are available in Beyoglu (2016) are being used as an initial estimate of parameters dependent on swelling factor of rock.

The following needs to be determined for calculating the impact of blast induced fragmentation:

- Mean bucket weight and standard deviation
- Count of load cycles
- Mean buckets per truck and Standard deviation
- Count of load cycles
- Preblast volume
- Post blast volume
- Swell Factor (Calculated from the preblast and post blast volume)

Swell factor formulae are

Swell Factor = Loose Cubic Meters/Bank Cubic Meters = (1 + Swell)

Bucket Factor is B_f (F_f / S_w BCM or ARD. Ff / Sw tonnes)

What should be the target fragmentation for efficient loading and crushing? It was found that ore hardness is the most influential in the mid-range fragmentation and influence of fragmentation is more pronounced in the coarse region for Aitik mine, according to Beyoglu at all (2016). Typical individual loading times based on loader activities are summarized by Brunton et al (2003) in the table below:

Table 31 Summary of individual loading cycle times (Brunton at al., 2003)

	Dig Time (Sec)	Swing time (Sec)	Dump Time (sec)	Return Time (Sec)
Mean	17.1	6.6	4.9	10
Median	15	6	4	9
Standard deviation	7.3	2.2	2.7	4.7
Minimum	7	2	1	2
Maximum	56	18	29	46
Number of samples	578	578	578	501

	Bucket Passes		Total Dig Time (Sec)		Truck Load Time (sec)	
	Cat 777	Cat 785	Cat 787	Cat 785	Cat 777	Cat 785
Mean	5.0	7.6	81.1	127.1	181.8	282.8
Median	5.0	8.0	77.0	127.0	175.0	274.0
Standard deviation	0.5	1.0	19.8	36.0	37.0	62.3
Minimum	4.0	6.0	47.0	67.0	113.0	189.0
Maximum	6.0	10.0	131.0	242.0	264.0	447.0
Number of samples	29	48	29	48	29	48

It was demonstrated by Brunton et al (2003) that at P80 passing size for Liebherr 994 excavator demonstrates the best correlation. In conclusion the following relationship was found to be true for average dig time versus P80 fragmentation:

Average dig time (sec) = 12.36+0.0072 X P_{80} (mm). **Equation 51**

Figure 131 shows how this formula is integrated into the model. Since this is a mine specific metrical formula one needs to remember to measure this for the specific mine concerned. For the sake of establishing the prototype model, the relationship will be used in the model setup (Figure 130).

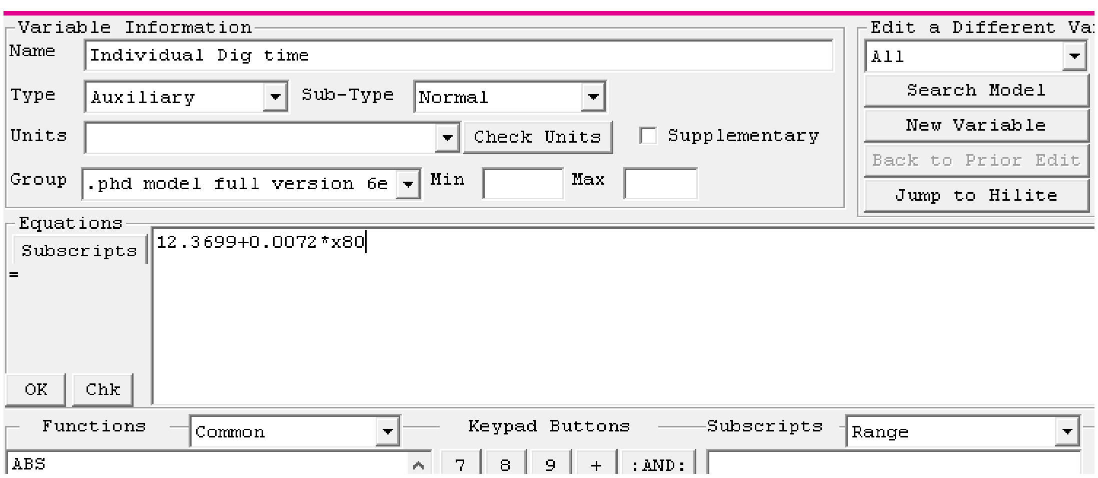


Figure 130 Formulae box used for calculation of individual dig time using Brunton formulae (2003)

Further info that will be needed for modelling is the fuel consumption of the trucks as well as loaders. The model is setup for Cat 777D truck model with a payload of 96 ton (Bajany, 2019). The fuel consumption for this truck as well as two types of excavators are reported in a literature where the case study has similar conditions of the proposed mine simulation for this thesis. The table is extracted from Bajany (2019) is shown in Table 32.

Table 32 Fuel consumptions for loaders and haulers (Bajany, 2019)

Parameters	Specifications
Model	777D
Rated payload	96 t
Gross vehicle weight	161 030 kg
Speed of empty trucks	50 km/h
Speed of loaded trucks h	36 km/
Fuel consumption (idle time)	22.38 L/h
Fuel consumption empty truck	44.76 L/h
Fuel consumption loaded truck	78.33 L/h
Shovel capacity	1600 t/h and 2000 t/h
Fuel consumption of shovel **- during idle time** **- during working time**	 1600 t/h - 6.6 L/h 2000 t/h -9.5 L/h 1600 t/h - 117 L/h 2000 t/h -130 L/h
Mine topography	Downgrade mine
Fuel consumption loaded truck	78.33 L/h
Gradient	1:14 4°

Fragmentation specification of "80% passing" (P80) can be used to calculate cost for each drill diameter as well as help select drill machine suitable to drill with a minimum production cost (Rajpot, 2009). Rajpot's work focuses on sensitivity of drilling and blasting cost to various parameters such as bench height and blast hole diameter. Beyoglu (2017) reports that the effects of fragmentation are first seen in loading no matter what type of machinery is used.

P80 determination is done in another part of the simulation setup using Kuzram Model as mentioned previously. The Kuzram model integrated into the model built for this thesis can be seen in Figure 132.

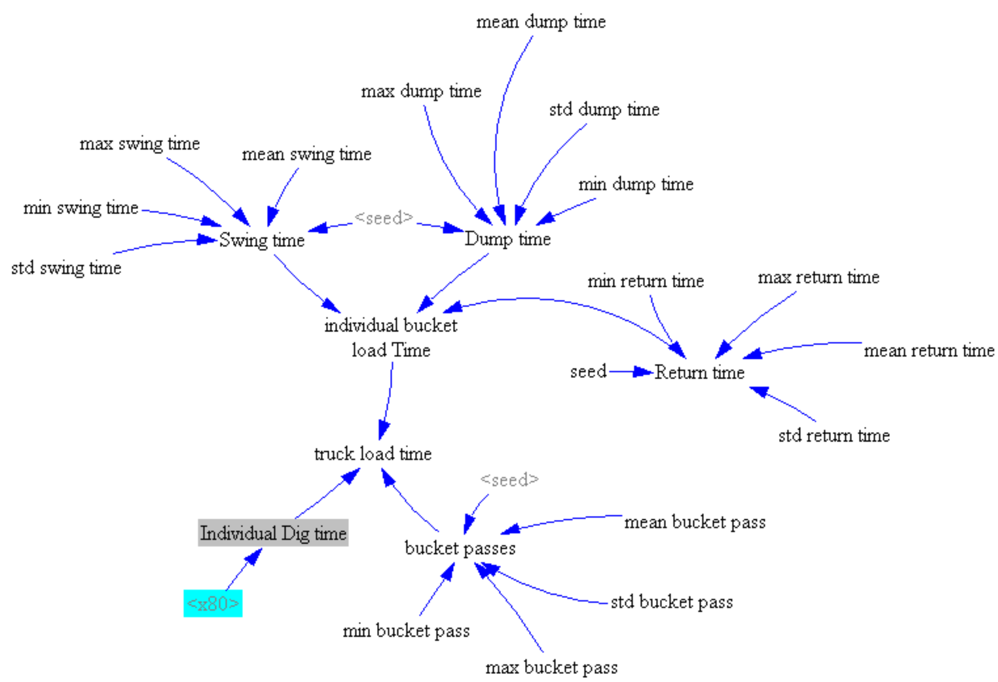


Figure 131 Building the model around the P80 size for digging time determination based on fragmentation.

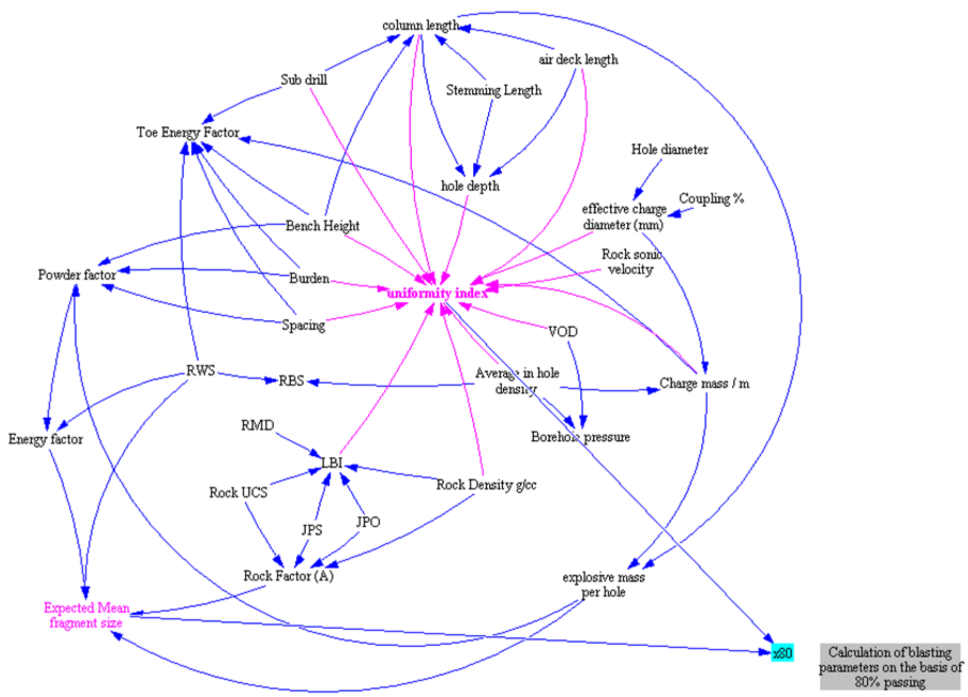


Figure 132 Fragmentation model for determination of P80 and other size fractions

Trucks and loaders are matched simply based on the loader size. The cycle times are simulated based on the historic statistical data for the loaders. The number of haulers is then determined based on the cycle time of the loader and the production demand. For checking whether hauling capacity will be sufficient to match the selected colour another calculation is done by the model. Included here is a typical run of the model showing simulated swing times of the loader, truck cycle time inputs and the resultant number of trucks as an example is included in this case for a 200 ton payload the number of trucks required will be 24 for the specific setup (Figure 133).

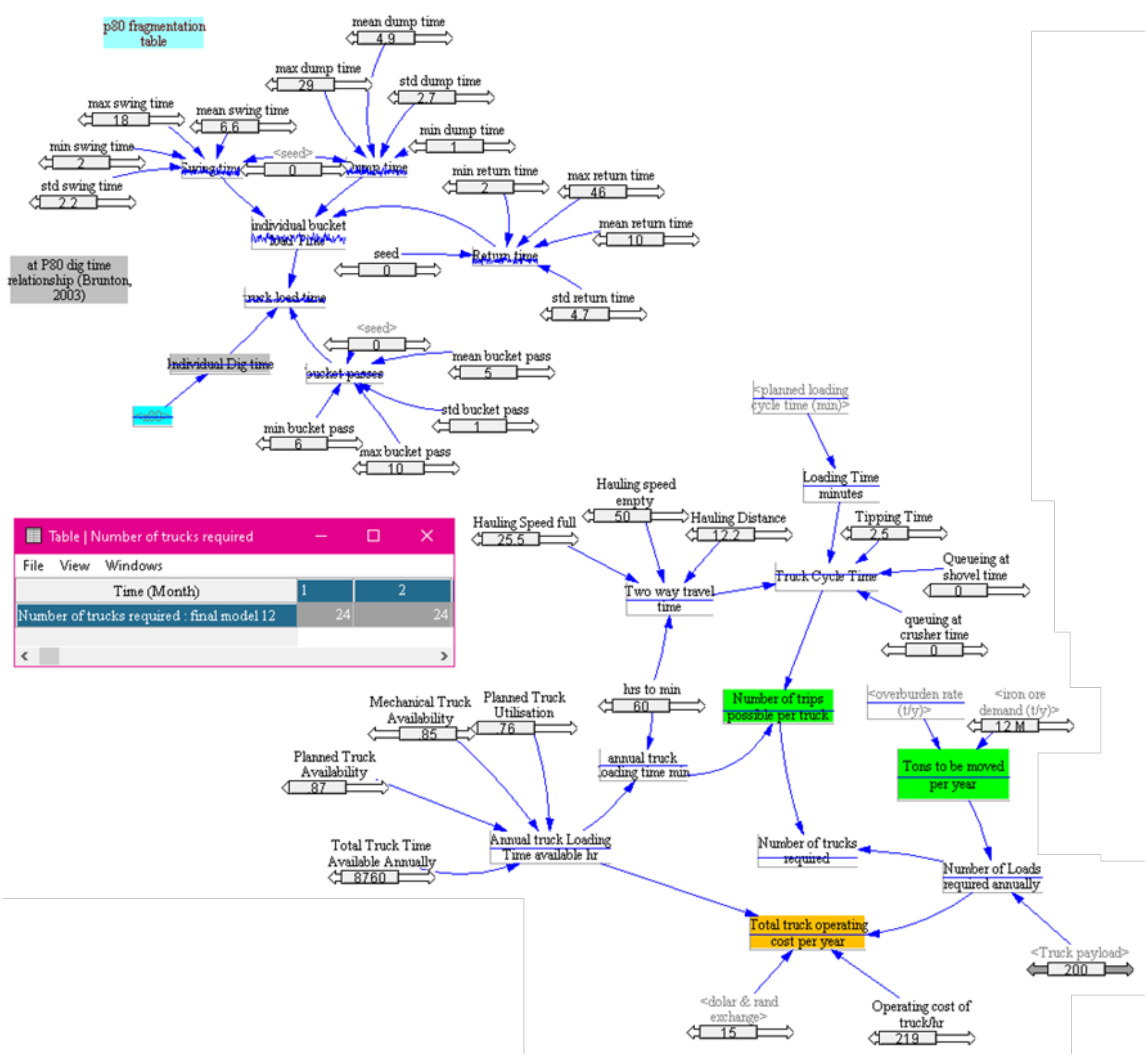


Figure 133 Resultant display from a synthesim mode simulated run of the loaders and haulers

6.2.3 Evaluation of Loading Productivity – Queueing Time

There are qualitative as well as quantitative factors that are associated with loading. Dipper fill factor, dipper payload, dig rate and frequency, dig cycles are all used as performance indicators (Onederra et al 2004). Sanchidrian (2011) considered rock properties and muck pile properties and explosive energy. In the end all efficiencies are linked to available time for loading and effectiveness of the loaders which in turn depends on a conducive environment.

It is most likely that queueing time is costing any load and haul operation considering 50 to 60 % of the operating costs is load and haul. Running the trucks and loaders idle means time losses and leads to delayed production as well as additional costs incurred such as ownership cost of trucks and loaders. If idle times of trucks could be prevented this could contribute to the cost savings in a great deal. These are easily modelled and calculated at that level of detail for an objective specific setup of the model. The focus therefore should be to include all these cost items and include in the dynamic modelling.

Loading productivity model is the same as the drilling productivity model and uses a calendar setup as that of drill rigs except there is no automation option built in for the loaders.

6.3 CRUSHING PROCESS LINKED TO BLASTING OUTPUT

The next step in production after loading is primary crushing. Although it consumes much less energy compared to grinding it has a massive influence on downstream processes. The power drawn by gyratory crushers depends on size distribution, hardness and shape of the feed, as well as liner profile, feeding rate, Closed Side Setting (CSS), eccentric speed and stroke of the mantle (Beyoglu, 2016). Beyoglu defined five classes of fragmentation that represents a muck pile fragmentation distribution. Then he correlated crushing energy as well as crusher throughput for each class of fragmentation. Crushing energy as well as mean crusher throughput graphs are shown in figures Figure 139 and Figure 140. These fragmentation classes were found to be useful to use in the new SD model to define energy consumption per class.

6.3.1 Rock Crushability

Rock Crushability is defined as the capacity of a crusher to produce a certain product fraction. It is also the capability of a crusher to produce desired product gradation and particle shape. Rock crushability is a combination of many elements (Rock Excavation Handbook, Sandvik Tamrock 1999).

Crushability Work indexes are classified into five categories. Where, relatively easy to break rocks' work index is less than 10 and difficult ones are greater than 22. Work index number is an estimate used by the mineral processing engineers.

The following are the general formulae used to estimate power consumption rates at the crusher based on the rock characteristics (Rock Excavation Handbook, Sandvik, 1999)

A= 2 • M • H / C **Equation 52**

Where A is impact strength [kpm/cm], M is the mass of one hammer [kp], H is the drop height [m] of hammer when the sample breaks, and C is thickness of the sample [cm].

Work Index is calculated accordingly:

Wi = 47.6 • A / p **Equation 53**

Where Wi is work index [kWh/t], A is impact strength [kpm/cm] and p is specific rock gravity [t/m3]. The result is the average Wi of all the samples. The maximum value of WI is also recorded. Average Work Index for granite is approximately 16 kWh/t.

6.3.2 Crushing Capacity

Crushing capacity is measured in tons per hour. Crusher throughput depends on the volume of the material, crusher settings. Same setting for same type of rock should give constant output in tons per hour.

Various rock types are given in the table below together with the work index associated with that specific rock. In addition, characteristics of the rock types are given such as density and compressive strength which are useful for comparisons (Table 33)

Table 33 Ch aracteristics of common rock types for prediction of of crushing energy (911metallurgist.com)

Typical Raw Materials		Characteristics of common rocks in alphabetical order				
Name of Rock	Type of Rock	Impact Work Index	Compact Density	Bulk Density	Abrasion Index (AI)	Compressive Strength
Amphibolite	Metamorphic	16±4	2.7-3.3	1.8	0.2-0.6	
Andesite	Igneous	17±3	2.4-3.0	1.6	0.1-0.6	170-300
Basalt	Igneous	20±4	2.8-3.2	1.8	0.1-0.3	300-400
Diabase	Igneous	18±4	2.7-3.0	1.7	0.1-0.4	250-300
Diorite	Igneous	19±4	2.7-2.9	1.7	0.1-0.4	170-300

Dolomite	Sedimentary	13±3	2.4-2.9	1.6	0.01-0.04	50-200
Gabbro	Igneous	22±3	2.8-3.1	1.8	0.4-0.6	170-450
Gneiss	Metamorphic	16±4	2.6-2.9	1.7	0.3-0.6	200-300
Granite	Igneous	16±6	2.6-2.8	1.6	0.3-0.7	200-300
Greywacke	Sedimentary	17±2	2.6-2.8	1.6	0.1-0.4	150-300
Gritstone	Sedimentary	16±3	2.6-2.8	1.8	0.1-0.3	
Hornfels	Metamorphic	18±3	2.7-3.1	1.8	0.2-0.6	150-300
Limestone massive	Sedimentary	13±2	2.3-2.8	1.5	0.001-0.2	80-180
Limestone unconsolidated	Sedimentary	7±3	2.2-2.7	1.5	0.001-0.2	80-180
Marble	Metamorphic	12±3	2.3-2.8	1.5	0.001-0.2	80-180
Porphyry	Igneous	18±2	2.7-3.0	1.7	0.2-0.9	180-300
Quartzite	Metamorphic	15±4	2.6-2.7	1.6	0.7—0.9	150-300
Sandstone	Sedimentary	11±3	2.5-3.1	1.7	0.1-0.9	30-180
Magnetite	Iron ore	8±4	4.0-5.2	2.4-3.1	0.2-0.6	
Hematite	Iron ore	11±4	4.0-5.2	2.4-3.1	0.3-1.0	

This source gives an average value for these rock types. Crushing and milling requires work indexes are determined differently and not the same. Jack Eloranto has a table of values regarding crushing and grinding especially for iron ore mines. Therefore, a value of 3 will be used for Crushing Energy consumption estimate for this thesis. Note that none of the values entered to the model are cast in stone but closer to the real values and should be adjusted as needed for specific studies and sensitivity analysis.

Table 34 Energy consumption for blasting, crushing and grinding (Eloranto, 1997)

PROCESS	FEED SIZE	PRODUCT SIZE	W(CALC)	W(Actual)	Apparent Efficiency
			kw-hr/t	kw-hr/t	
Blast	4 m	.5 m	.15	0.43	36%
Crush	.5 m	2 cm	.95	3.24	29%
Grind	2 cm	60 microns	20.39	17.82	114%
TOTAL	4 m	60 microns	21.49	21.49	

6.3.3 Effect of Fragmentation on Crushing

Crushing is the first mechanical stage in the process of comminution in which a principal objective is the liberation of the valuable minerals from the gangue. It is a dry process and lumps of run-of-mine ore as large as 1.5 m diameter are reduced to 10 t0 20 cm in size.

The generally accepted and prediction of the specific energy consumption is based on Bond's law as follows (Kecojevic and Komjenovic, 2007):

$$W = 10 * W_i * \left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right)$$ **kWh/short-ton, or** **Equation 54**

$$W = 11 * W_i * \left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right)$$ **kWh/metric-ton** **Equation 55**

W-predicted mill energy consumption in kWh/st

Wi= Work index, in kWh/st.

The Bond equation is used to scale the energy consumption, but the energy required to crush a ton of material differs with crusher type. There is a wide range of values obtained for the same P80 or F80 due to the differences in the type and nature of the rock, the design of the blast, the pattern of the drilling and the explosives amount used per ton of rock.

A representative power draw based on Work Index (WI) of the iron Ore based on the P80 product size is being calculated with the following setup in the model (Figure 134).

Some researchers suggested size specific energy to be used for estimation of power consumption instead of using Bond Work Index. (Foggiata, 2017)

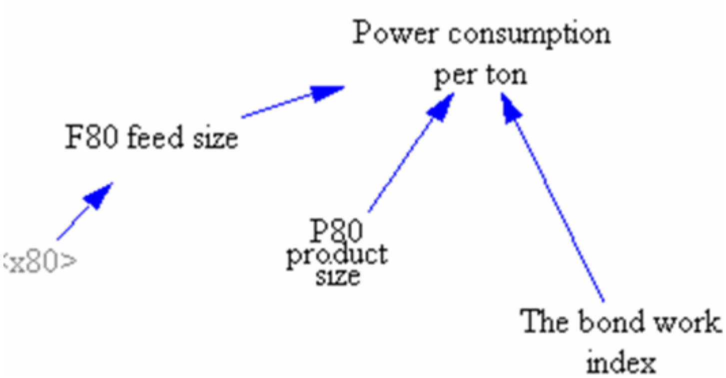


Figure 134 Calculation of power consumption using Bond's Work Index developed in VENSIM

The cost of crushing is to be calculated for ore production only as waste does not go through the crusher.

The cost of electricity is the major factor which keeps increasing over the past few years in South Africa. The historical electricity tariff is summarised in Figure 135. This makes it interesting to the researcher to search the effect of ever increasing cost of power on the overall economy of the mining.

Cost of electricity is entered as a lookup table in Vensim (Figure 136).

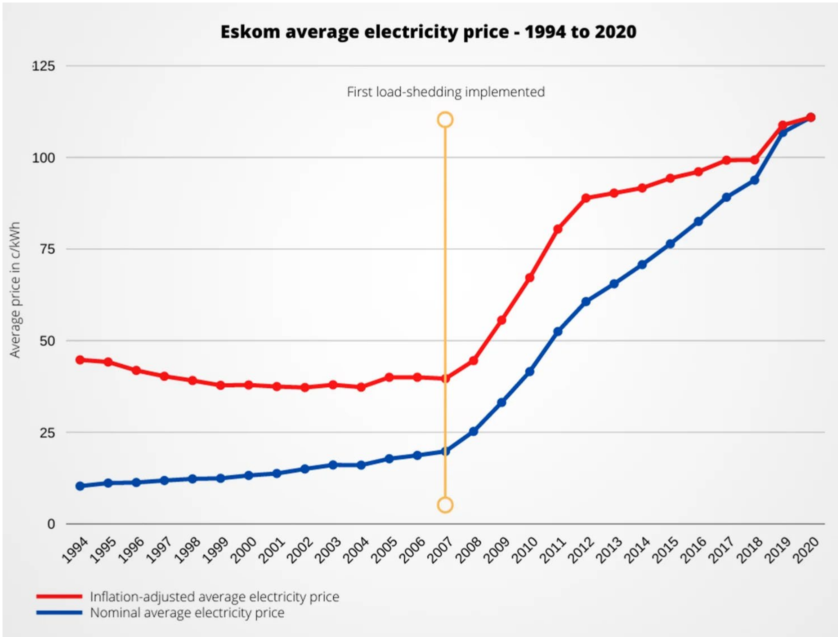


Figure 135 Cost of electricity from year 1994 till 2020

The cost of electricity is built into the model as a graph as seen in Figure 136. The numbers are extracted from the Eskom average electricity price graph as in Figure 135.

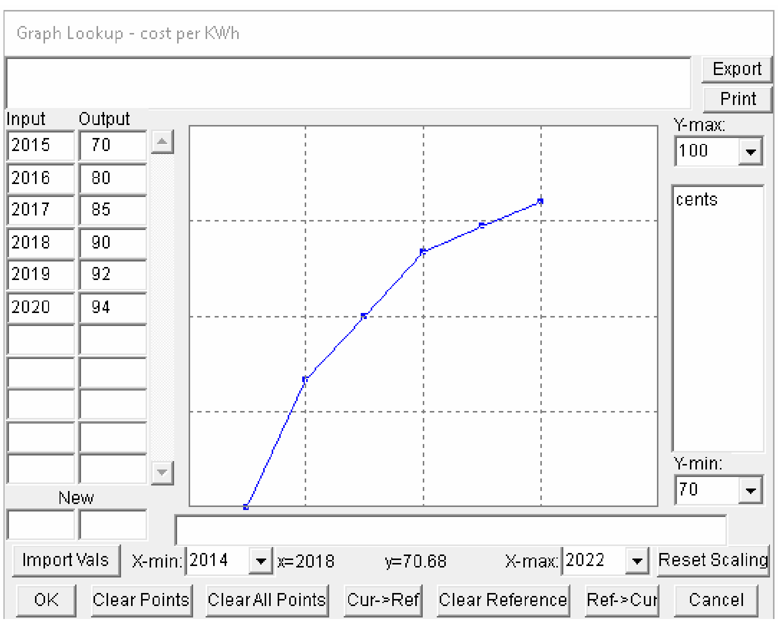


Figure 136 Cost of electricity in South Africa since 2015

Bond's third law of comminution depends on 80% passing particle feed size to convert it into 80% passing particle product size using a constant called the Work Index (WI). Bonds Work Index is defined KWh per short ton required to reduce the material from theoretical

80% feed size which passes a size of 100 microns. This law is still the best estimate of the power requirement determination.

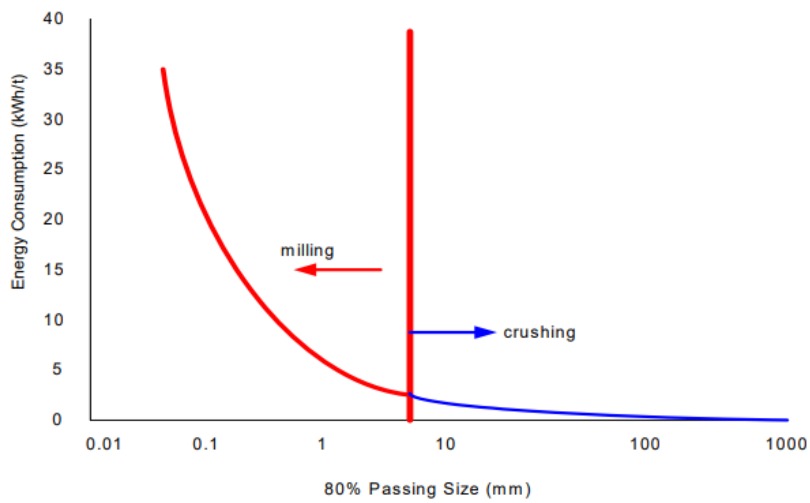


Figure 137 Communition energy consumption as function of product size (Valery & Jankovie, 2002)

Typical values for crushing work index are listed on a manufacturing site on mineral processing are tabulated in Table 35

Table 35 Crushing work index for common rocks (max-plant.com)

Softer Rocks	Work Index (kWh/t)	Hard Rocks	Work Index (kWh/t)
Glass	3	Quartz	14
Clay	8	Gold Ores	16
Coal	13	Iron Ore	17
Limestone	13	Basalt	22

Crusher type and capacity affect the power draw. For a reasonable comparison of power draw before and after automation of drill may give a better indication of the effect of fragmentation on power demand. When there is a lack of data a prediction model may be used to estimate the power draw. The prediction models depend on Bonds work index which in turn uses 80% passing size of the muckpile. 80% passing size can be estimated with a reliable fragmentation prediction model, such as Kuzram.

Processing ore requires a significant amount of energy of which the largest portion is communition. A pie chart by US department of energy in 2007 shows the distribution of energy requirement per type of process as shown in Figure 138.

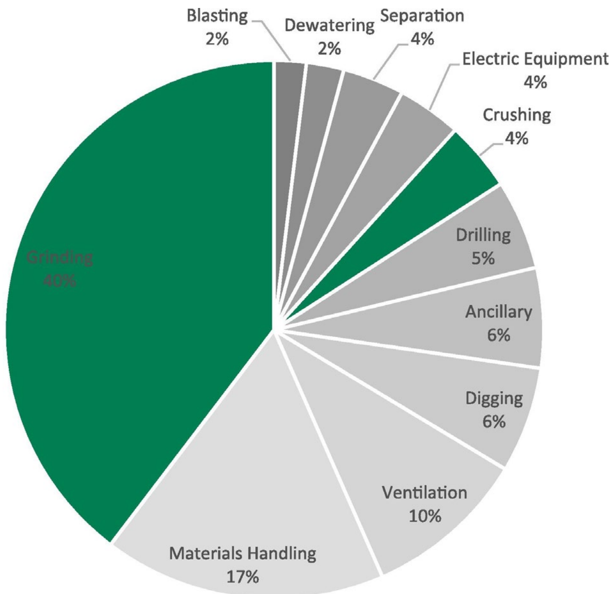


Figure 138 Energy consumption by mineral processing activity

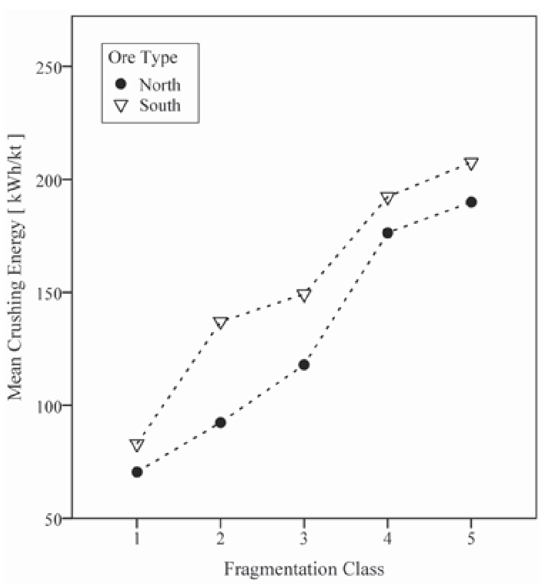


Figure 139 Mean Crushing Energy versus fragmentation class

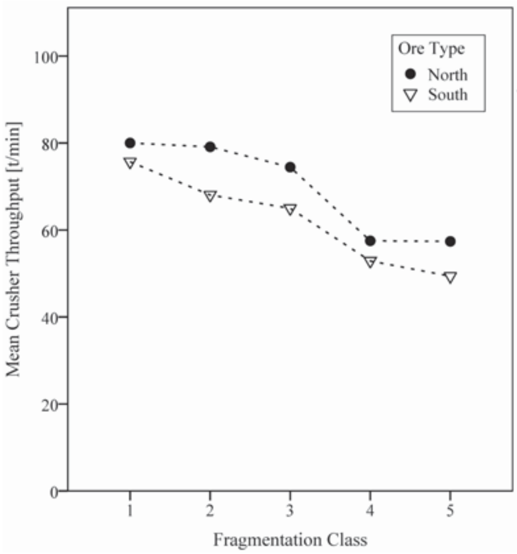


Figure 140 Mean crusher throughput versus fragmentation class

The effect of mean fragmentation against crushing energy, crusher throughput and shovel fill factors were quantified by Beyoglu (Pg¸43). These values are only valid for relatively medium strength rocks of up to 120 MPa. Harder ore figures are shown in Table 36. Fragmentation classes being are classified from very fine (1) to very hard (5). The fragmentation classes are reported as in Table 36.

Table 36 Feed fragmentation and their indicators for relatively harder ore (Beyoglu, 2016)

		Crushing Energy (kWh/kt)		Crusher Throughput (t/min)		Shovel Fill Factor (%)	
Fragmentation Class	Ore Type	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Std dev
1	North	70.46	19.59	80.01	18.32	87.88	10.34
	South	82.91	13.71	75.70	11.21	78.67	9.96
2	North	92.36	21.49	79.12	20.55	90.89	7.46
	South	137.04	34.66	68.05	16.16	93.32	8.44
3	North	117.95	28.80	74.46	20.34	84.30	9.84
	South	149.21	28.77	64.99	17.34	80.58	14.33
4	North	176.38	31.71	57.52	10.16	74.12	11.95
	South	192.30	36.16	52.89	14.92	65.32	10.08
5	North	190.03	19.18	57.39	9.27	63.49	13.11
	South	207.57	45.08	49.39	10.58	60.62	10.35

It can be seen in Beyoglu’s (2016) research that power draw varies based on the crusher throughput rates and which in turn influences the shovel fill factor. As the crusher throughput is decreasing the shovel fill factor is also decreasing. This relationship although not directly usable in this thesis, it gives the author the idea that the model can be built with a shovel fill factor or influence of the diggability to the crusher being active. Most times crusher would be waiting idle, and this would draw at least 50% of the power required compared to a choked crusher operation in a choked mode which is considered as more efficient in terms of effective crushing. Production delays may mean crusher is working idle for longer periods.

Table 37 Pre-defined approximation of P50 and P80 sizes for fragmentation classes (Beyoglu, 2017)

Class	P50 (mm)	P80 (mm)
1	209±39	464±51
2	336±54	615±73
3	510±59	785±74
4	653±77	994±105
5	820±91	1174±156

The crushing energy draw depends on the CSS opening size and the feed choke situations as well. Choking the crusher leads to more efficient crushing but also increases the power draw during the choked crushing operations. The relationship between ton per hour production, CSS opening, and the energy consumption is best represented with a graph as seen in the example of (Liu, 2018). This type of graph if exists for iron ore could be useful for modelling purposes and could be built in as a lookup table within the model. As a quick input the model assumes a simple value based on the work index of the crusher. The paper by Beyoglu (2016) also presents typical crusher loads according to the size ranges for each class definition which can be seen in the Figure 141.

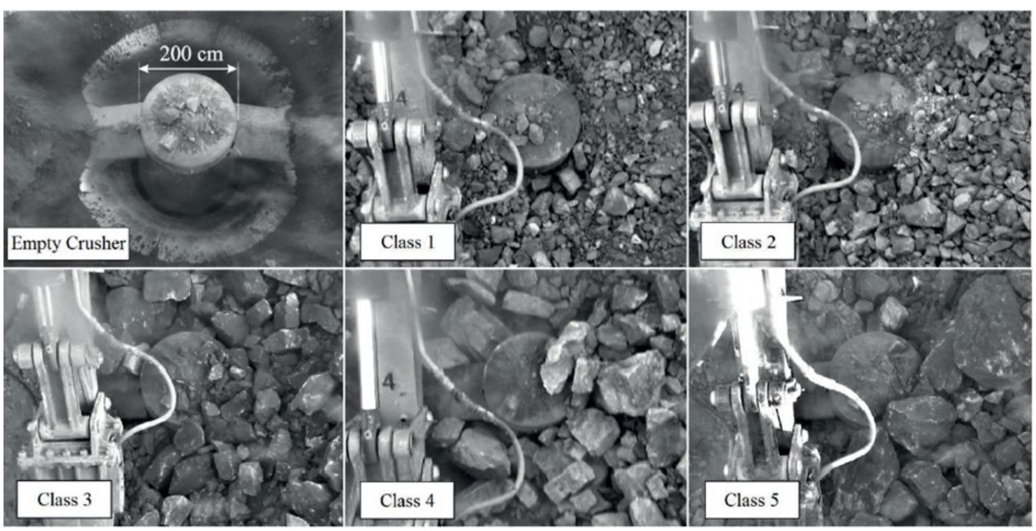


Figure 141 Beyoglu's (2016)defined class definitions pictures

The classes defined by Beyoglu (2016) was used in this research to incorporate mean fragmentation size based power draw and throughput for estimation of the energy demand and therefore costs. The SD model is modified with the following small sub model inclusion.

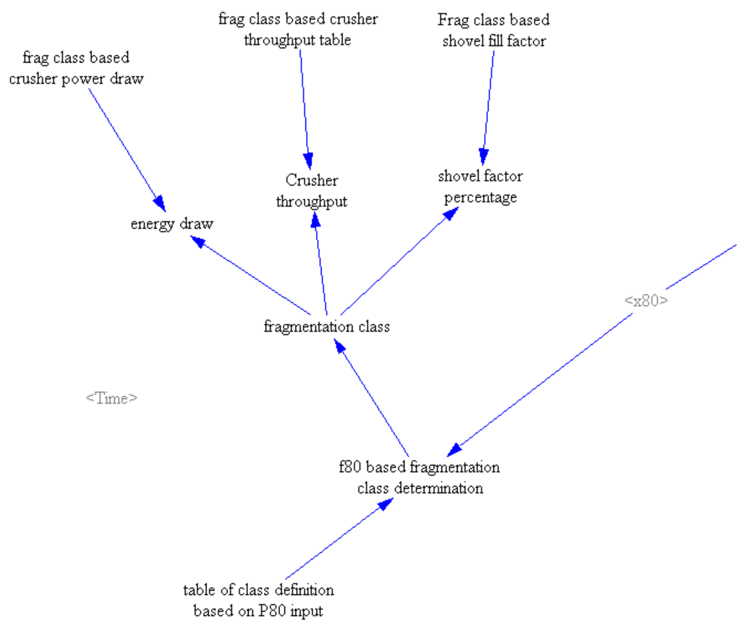


Figure 142 Determination of power draw and bucket fill factor based on fragmentation input to the crusher

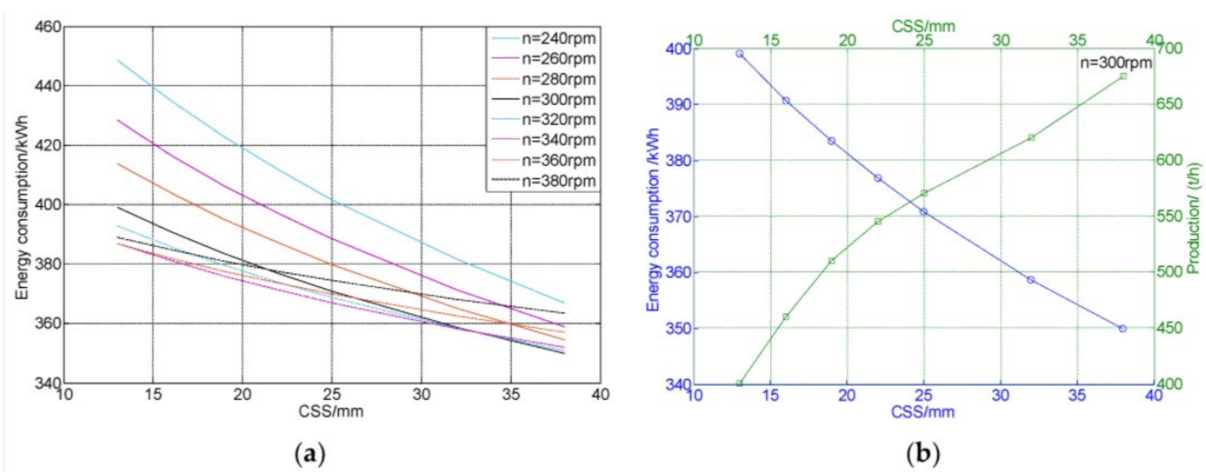


Figure 143 Relation of CSS and energy consumption: (a) CSS vs energy consumption; (b) CSS, production and energy consumption. (Liu, 2018)

Crusher energy draw is also dependent on the amount of oversize which is largely impacted by drilling and blasting practices. Modelling of the correct energy demand and fluctuations due to changes in drill and blast parameters then makes it possible to see the immediate effect.

The simulation of the crushing is therefore a critical aspect of the total value chain in terms of costs related to energy demand.

There are difficulties regarding building an estimated power draw due to inferior drilling and blasting output. But it is undeniably highly influential based on the evidence of the literature reviewed.

There are complex crusher power draw estimate models available, but the complexity of the draw also requiring further measurements from the related crusher plant it was too complicated for this author to also built that process into the existing process. This could be the topic of another researcher who is specialized in mineral processing. Not the complicate the requirement of fragmentation related cost calculation, a simple approach was used for this thesis using the crushability index formulae based on the Bond Index. In addition, Figure 137 was a guide to say at least the model built in is within the ranges of the literature.

6.4 Chapter Summary

The research around modelling the mining environment towards quantification of a technological change using system dynamics required an abstraction of larger sub systems in a mining environment. This included modelling the behaviour of each unit process implemented fundamentally using Vensim. The subsystems modelled are drilling, blasting, loading, hauling, and crushing. Understanding the value chain helps in exploring the contributing factors to a certain output or behaviour. The potential intended and unintended consequences can be assessed only when all the correct contributing factors are captured. The problem is then redefined by having a closer look at the contributing factors.

This chapter highlighted all contributing factors towards a specific problem in a systematic way following the value chain described earlier. The next chapter will put the SD model developed to the test with a case study.

Chapter 7

7 DISCUSSIONS OF THE MODEL WITH A CASE STUDY

In Chapter 6 the unit processes were systemically defined with all the contributing factors towards building a bigger model. In this chapter the model is further refined and tested using with a case study identified at the beginning of the research. In this chapter the case study identified as well as personal experience have been used to set the boundaries of a realistic simulation environment to test if the model behaviour is within expectations.

7.1 Case Study Background

Kolomela mine is located in Northern Cape which primarily produces export grade iron ore. The total mining area is roughly 5kmx6.5 km.

Kolomela Mine produced about 13 to 14 million tons of iron ore and moved 50 to 55 million tons of waste in 2017. Unit costs reported per ton of ore is 240 to 250 Rands per ton. Stripping ratio has been indicated as 2.8 for the life of mine. (Themba Mkwanazi Speech, 2017). Kumba Iron Ore production summaries reported end of 2018 states that expected unit costs were between R315/tonne and 325/ton for Sishen and R265/ton to R275/ton for Kolomela

This statement is dated as 31 December 2018. The same report states the following production figures in 2017 and 2018 (Table 38). The cost figures for both years include probably the transport cost to the port.

Kolomela mined tonnes were 71.8 Mt for the year of 2017 and 72 Mt for 2018 of which 13.9 Mt was ore production and 56 Mt waste moved. Waste Shovel tempo increased by 36% and the mine achieved 80% mine to plan compliance. Iron ore export price was $72/ton in 2018 and $71/ton in 2017. Reserve life of Kolomela is 14 years since December 2018. More information on Kolomela mine can be found in Table 39. The values entered into the model are as reported in these figures.

Table 38 Kumba ore production

000' tons	December 2018	December 2017	% Change
Total	43106	44983	-4
Lump	29172	29812	-2
Fines	13934	15171	-8
Mine Production	43106	44983	-4
Sishen	29246	31119	-6
Kolomela	13860	13864	-

Table 39 Kolomela mine information

KEY DETAILS OF THE MINE	
Commodity	Iron ore
Country	Republic of South Africa
Mining Method	Open pit – conventional
Reserve Life	14 years
Lump to Fine Ratio	60:40
Saleable product design capacity	15 Mtpa
Estimated Saleable Product in 2018	14.2 Mt pa
Estimated Waste Production in 2018	56.5 Mt pa
Overall planned stripping ratio (Life of Mine)	4.1:1
Estimated product sold in 2018	13.7 Mt

Recent technology activities focus on value add, capex, site alignment, business skills levels and competency as well as industry maturity. It has been also reported that Kolomela blast hole drilling fleet is fully autonomous by August 2017 where improved safety, increased drill hole quality is reported as major benefits. To be more specific, increased direct operating hours from 17.3 to 18.6 hrs per day per drill machine due to reduced levelling time from 1.3 minutes to 0.4 minutes and improved tramming time from 2.4 minutes to 0.9 minutes.

Production outlook for Kolomela is summarized in two graphs as before 2018 (mined, Figure 144) and after 2018 (planned, Figure 145) as reported in Ore Reserves and Mineral Resources document published by Kumba in December 2018 (pg.43)

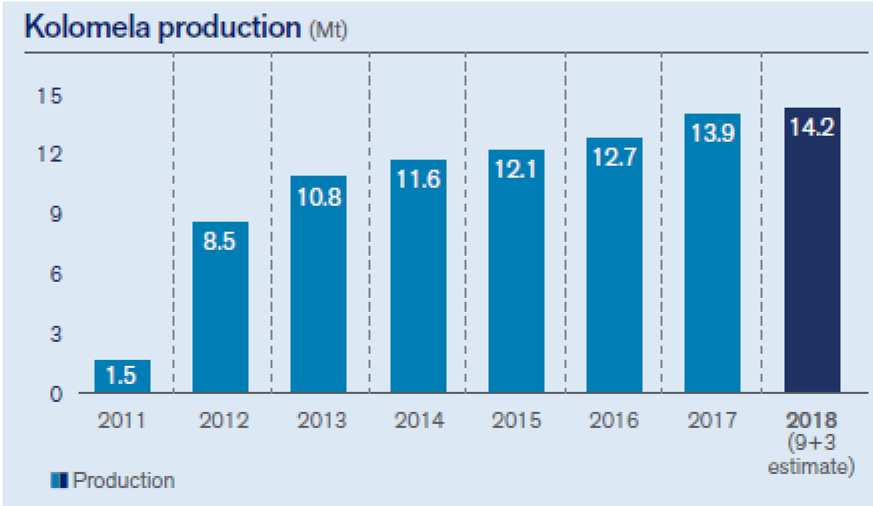


Figure 144 Kolomela production in Mt until 2018 (Kumba Annual Report, 2019)

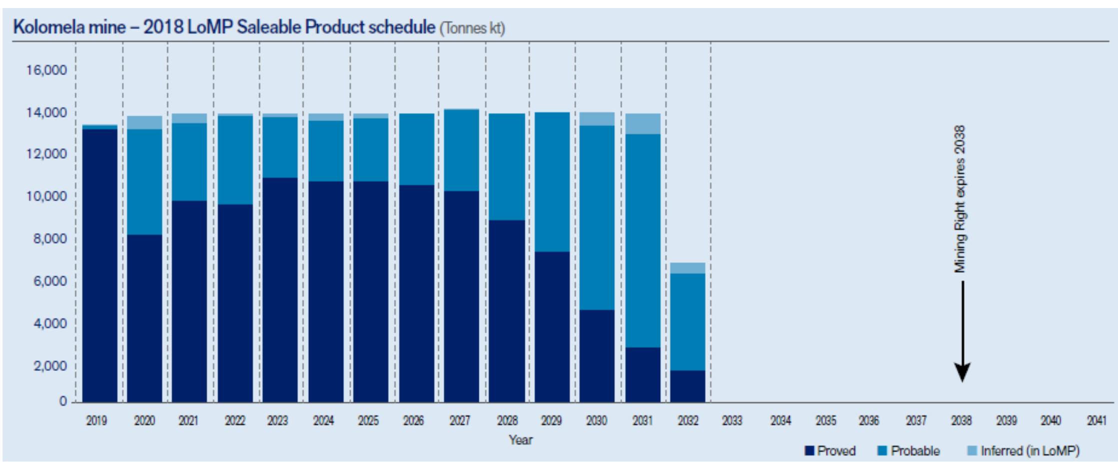


Figure 145 Kolomela saleable product schedule based on modified beneficiated inferred mineral resources (Kumba Annual Report, 2019)

The plant can handle around annually 15 Mt. The ore is not washed and processed, only a small portion (6%) is contributed by DMS plant. The rest of the ore is only crushed and screened and considered as DSO (Direct Shipping Ore). On average 60% lump and 40 % fine ore is being railed to Saldanha export harbour.

7.2 Case Study Operational Parameters

Kolomela is using 730E Komatsu as well as CAT 777 trucks and they are reported to have queuing times reduced from 5.7 to 4.9 minutes due to fleet management system optimization to maximize productivity and efficiencies. The average speed of trucks was increased from 21.2 to 29 km/h. A high Precision GPS shovel guidance system was also part of the technology projects. (Modern Mining, 23 July 2018).

The tyre life of trucks is reported to be much shorter than manufacturer recommended tyre life in hours in an iron ore mining environment. (Lindeque, 2016). In this study the target tyre life for Komatsu 730E trucks is 7200 and Cat 777 trucks is 5600 hrs. Tyre life is dependent on many factors including bench surface area tidiness and haul road conditions and some other factors such as operator efficiency, tyre pressure maintenance. Some of these factors could be attributed to the drilling accuracy related downstream affects. Manufacturing quality also could be an issue as it seems to be a sudden decline in the tyre life in both trucks.

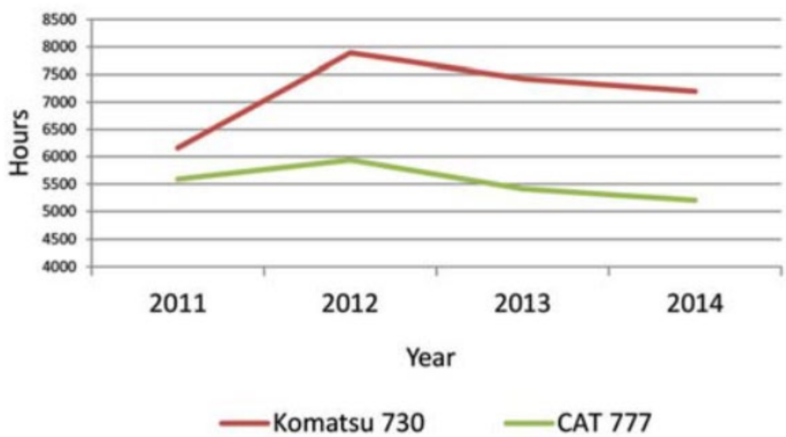


Figure 146 Tyre life for two truck types (Lindeque, 2016)

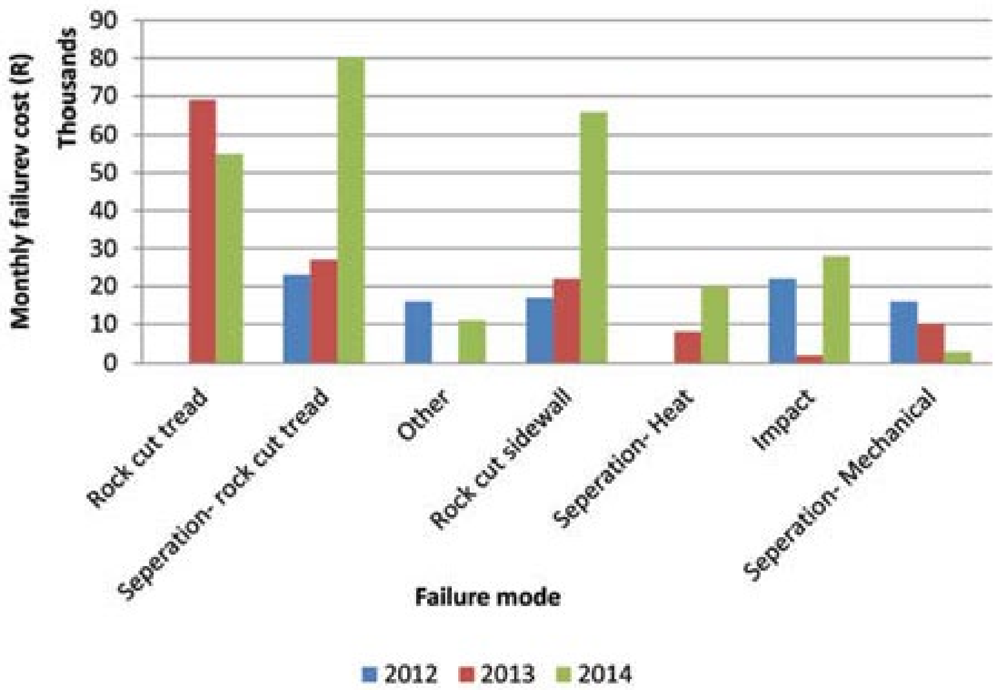


Figure 147. Reasons for tyre failures (Lindeque, 2016)

It has been reported that automated and autonomous drill fleets at Kolomela improved safety and drill quality and increased direct operating hours from 17.3 to 18.6 hours per day on average. The levelling time for each drill reduced from 1.3 to 0.4 minutes and tramming time improved from 2.4 to 0.9 minutes. (Modern Mining, 23 July 2018). The stated numbers above are entered into the created model and will be discussed later in this section.

7.3 Simulation Setup for the Case Study

The model had to be divided into smaller sections for easy navigation as if single page is used for visualisation, then the SD user would not be able to see the immediate picture per sub model due to the extent of the model. Therefore, new SD model created in VENSIM consists of various pages and all are linked to the index page, the entry page where explanations and general layout is shown in Figure 148. Each item is hyperlinked to the respective model page within the same document for easy navigation.

This model has 819 symbols, 11 lookups, 52 levels, 375 auxiliaries, 319 constants in total. Each of the auxiliaries and levels are calculated values and normally contain equations linking all the variables and constants based on the causal relationships.

The simulation model captures almost all main processes in the mine to the level of detail as practicable as possible. Some areas where certain numbers are used as constants and inputs are not mine specific, these however are to be realistic enough for a reasonably comparable output. If the author believed there needs to be a parameter that needs to be measured it was also entered into the model. In the future, this will pave the way to a comparable study of those parameters in more detail with field measurements conducted in a mine.

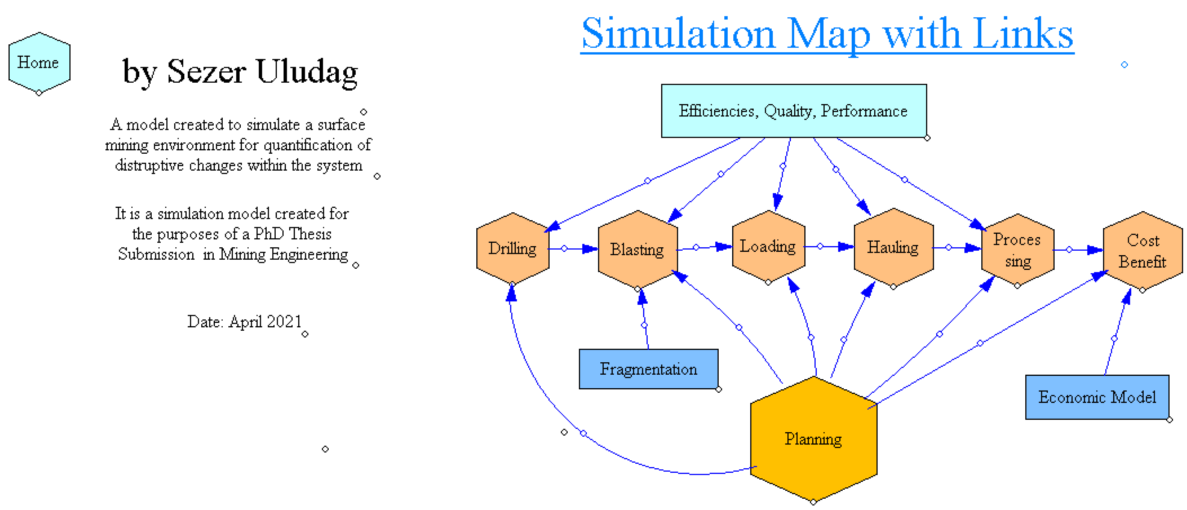


Figure 148 Welcome page of the simulation file

The pages are also accessible via a list at the bottom of the VENSIM interface. The detailed formulae used in the simulation are listed in table in Appendix A and are based on discussions on chapters five and six.

The new SD model starts with a planning process related page with mine specific production profile. The production profile includes waste and ore mining setup on a monthly basis.

The planned production as reported at company website is 14.2 Mtpa (million ton per annum) with a design capacity of 15 Mtpa. The model can be setup based on the production profile for the complete life of mine with lookup tables or with a function modelled as well as kept constant throughout the consecutive years. For the sake of this study the base case of 2018 is used and only one year period is simulated for ease of reference and practical reality checks.

The waste produced is calculated based on the stripping ratio reported. Stripping ratio can also be modelled in as a lookup table in Vensim. The automation process of drill rigs at the mine started in 2015 and in 2017 most of the drill rigs were installed with the GPS navigation systems, cameras and a control room where the drill rigs are monitored and setup.

7.4 Mine Planning Sub Model Setup

The mining process starts with planning. The demand for the commodity being mined defines the mine design, planning and scheduling process. As a first step in setting up the simulation, a basic production plan has been constructed using the information presented in Table 39.

7.5 Setting the TIME, TIME STEP, FINAL TIME in the SD Model

Selecting the time factor involved for this research required some study after a few failures of the intended mode. Since TIME usage makes the model dynamic it needs to be carefully selected since it also effects the output files size, therefore run time becomes longer. It is essential to select the correct TIME STEP. These variables: TIME, TIME STEP and FINAL TIME, are pre-defined variables already existing in the model and cannot be easily manipulated or changed. If the model is not built flexibly to allow the time step changes. Some formulae built in take into consideration that every time step is a month therefore monthly production is directly entered. When these variables are included in the formulation one must be careful since the logic behind the formulae is bound to TIME STEP. IF the time step is changed the results could change to unrealistic numbers unless modified in the formulae area to change as the time changes. A monthly TIME STEP is selected for this model for the sake of simplicity and be able to grasp changes in variables due to dynamic nature. Another reason is that most literature report monthly figures and this has been found to be a practical approach.

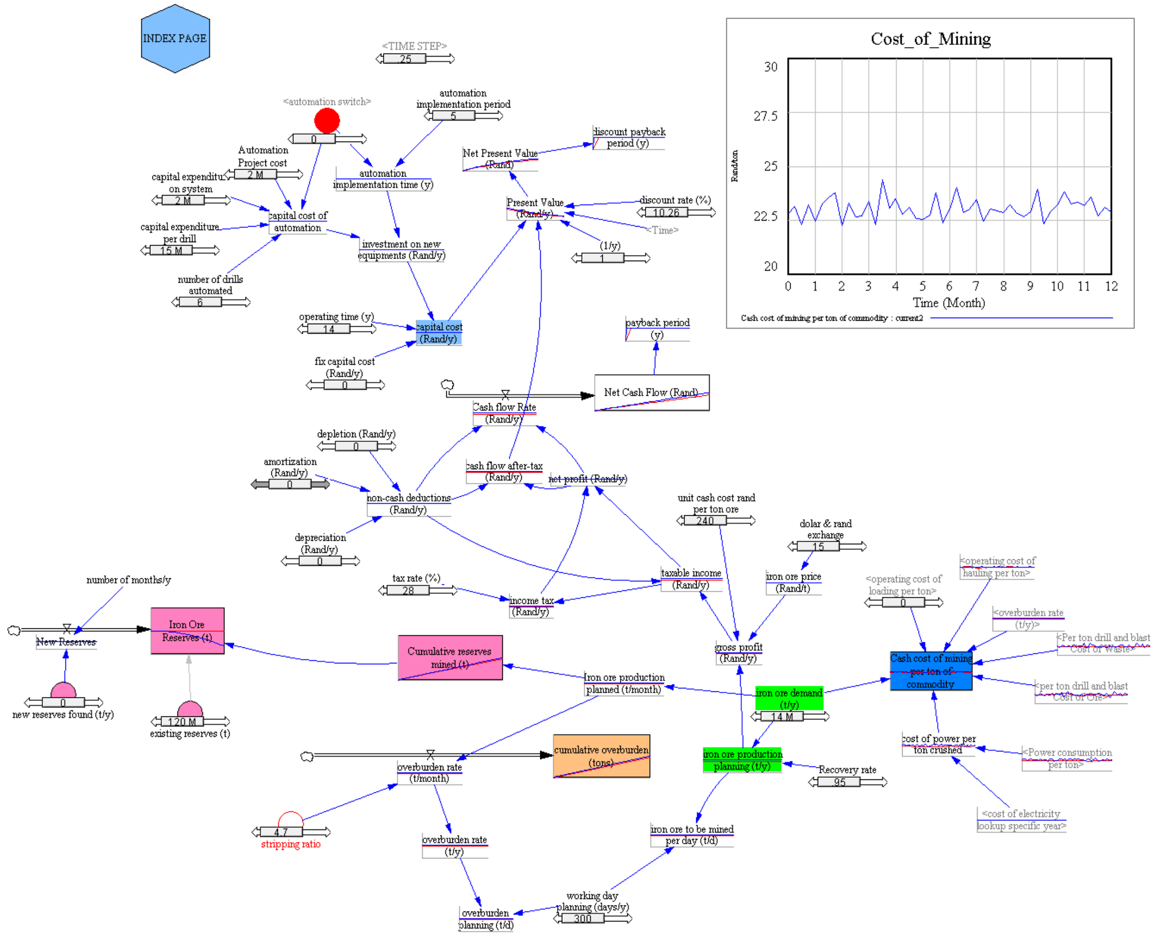


Figure 149 Production planning setup with costs and revenuesCustomers and Servers Analogy

The analogy of customers and servers (see section 5.4) is being used to simulate the mining processes in terms of efficiencies and costs. In this case the customer is a mining block which will be served with various servers or processes. The mining block arrival rate depends on the production requirements. Arrival will be adjusted per time step. TIMESTEP is a function used in VENSIM to describe the moment at which something happens. The mean arrival rate of blocks to be serviced will be calculated based on the mining schedule planned on a weekly or monthly basis. There are two options for modelling as discussed in section 5.4: Discrete or continuous. Continuous modelling is suitable when tons moved is continuous which is true for conveyors. Discrete modelling is however more appropriate as mining is advanced block by block due to discrete drilling, blasting, loading and hauling processes. Each block is treated with distinct levels of service according to capacity and cycle time of those processes or production events.

7.6 Constants, Auxiliaries and Equations in Drilling and Blasting Process

The drill and blast geometrical relationships are setup and linked to production demand for both ore and waste drilling. From this also stems the drilling parameters. Drilling and blasting parameters used in the simulation is approximately similar to the real mine values.

The initial blasting parameters that are entered into the model are listed in Table 40 which are based on this researcher's experience, and most are typical in such mining environment.

Table 40 Blasting parameters used in simulation

Drill and Blast Parameters	Waste Blasting	Ore Blasting
Bench Height	12	12
Subdrill	2	2
Hole diameter	229	229
Burden	6	6
Spacing	5	5
Stemming Length	Burden*(0.7 to 1.1	Burden*(0.7 to 1.1)
Powder Factor	1.2	1.2
Detonator	Shock tube	Shock tube
Rock Density	2.45	3.52
Blasting Block Size	300	150

The blasting parameters will be manipulated to reflect incorrect drilling practices to simulate and capture the resultant fragmentation due to artificially created spatial errors in the XYZ coordinates of the blastholes. A normal distribution is assumed for all these errors, but this can be changed if a mine's error distribution curve is known.

It has been this author's experience that even with GPS navigation there will be at least 5% of the blastholes being deviated from the required location and sometimes re-drilled sometimes not completely noticed. Therefore, it can be easily assumed that XYZ position of the blastholes will have variation in drill position quality. This variation can be simulated with a statistical description for the type of input entered into the SD model to calculate the resultant cumulative fragmentation distribution of the muckpile.

Table 41 Some of the equations and relationships used during the model construction

Performance Measurement

Direct Drilling time available per drill per month	Drilling Primary Production time per year/12
Drilling Primary Production time per year	Drilling Direct Operating Time-Drilling Secondary Non Production Work
Drilling Direct Operating time	Drilling Uptime-Drilling Lost Time
Drilling Secondary Non Production Work	10 hr
Direct operational hours per day per drill rig	Drilling Primary Production time per year/Direct operational days per year
Drilling Lost Time	Drilling Consequential lost time (hr)"+"Direct Drilling Delays (hrs)+"Drilling Lost Time Standby (hr)
Drilling Consequential lost time (hr)	30 hrs

7.7 Dynamic Flow of the Mining Sub processes

The first level is preparation of a blasting block (after loading is completed). Preparation involves levelling off the bench for the next drilling-blasting-loading-hauling cycle. During preparation, a dozer is brought to the side and the bench is levelled. Also, any toes, loose material, clutter from previous cycle is removed. There after the block is surveyed and staked with next blasthole pattern.

The drill block discrete completion time model is shown below.

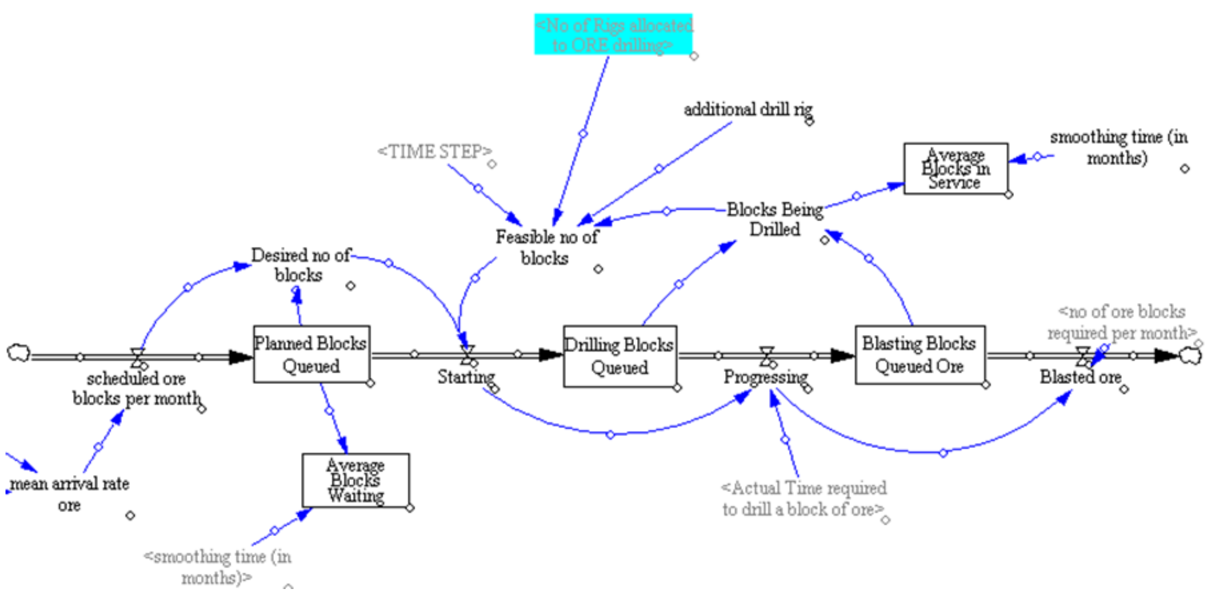


Figure 150 Discrete modelling of drill block completion time.

This part of the model's intention is to have the production flow which is linked to the performance parameters in order to see the effect of cycle times on the output. The run is based on constant inputs and relationships for block sizes and bench configuration. However, this could be much more complex, meaning the mining blocks being scheduled for ore and waste vary in size considerably due to being driven by the demand of the plant and in pit blending efforts of the mine itself.

The model was useful to see the effect of having smaller or larger blocks on the cycle times of blasting. The scheduling of blocks and the real effect is not considered a direct consequence of drilling efficiencies therefore it is left at the simple level for this thesis, can be further expanded for another research. The blasting block sizes in a mine may be selected in the initial mine modelling stage during feasibility studies of the mine. This block size often determines the number of equipment required. One the mine starts operating the block sizes stay the same. But this may not be practicable in certain times of the mine's operational life. The weekly or monthly planning may adjust the blasting blocks, this has an effect on the resource allocation. The same is valid for reducing the size of the blocks in dire times.

The issue of designing smaller block sizes at a mine may in turn increase the requirement of the number of drill rigs on site to be able to drill multiple small blocks to keep up with the demand of the plant. It is possible to add drill rigs within model (in addition to the existing fleet) to balance the production demand, i.e., having a levelled stock for demand and meaning no accumulation of the work inflowing or queued, the additional resource is named as "*additional drill rig*" in the model. The additional drill rig function can be related to the additional workforce from drilling contractors due to mine simply does not have the capacity to allocate additional drill rigs.

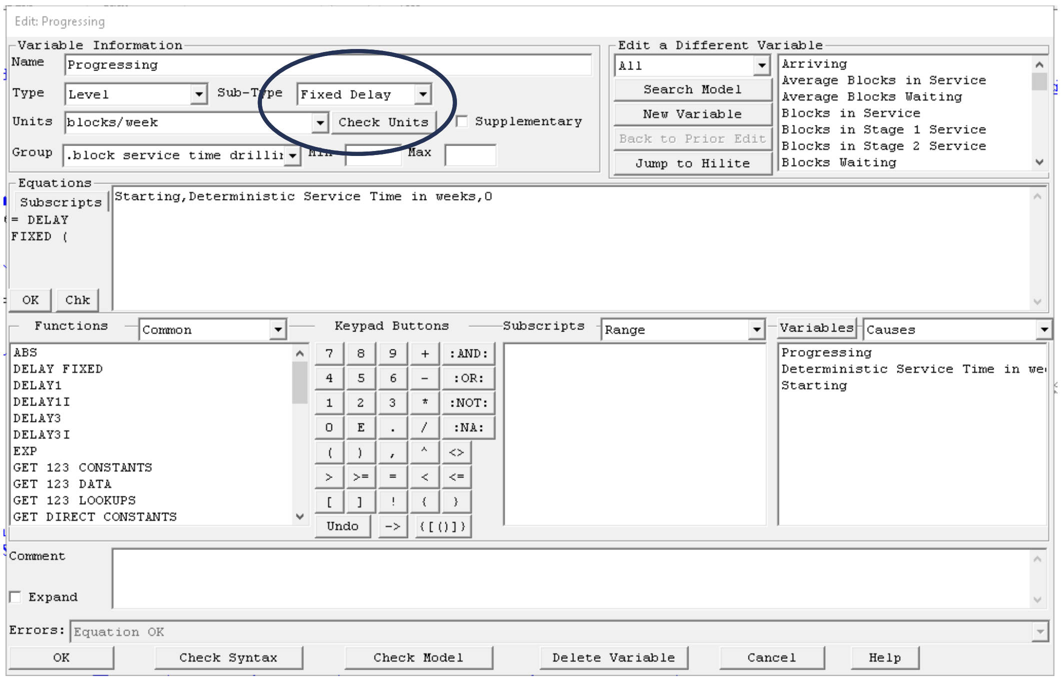


Figure 151 Fixed Delay function used for drill block completion time determination in ideal conditions

The SD model created was used to see the impact of changing the blocks sizes and how it created the variability in the required number of drill rigs.

7.8 Dynamic Simulation of Fragmentation

Fragmentation simulation method used during the first modelling stages considered to have results on the entirety of the muck pile in terms of fragmentation profile. In principle this seems to be right and gives an overall estimate of the muck pile as an indication of the fragmentation profile. However, individual quality of the blastholes due to inaccuracies cannot be easily incorporated into the model to capture the impact of the fragmentation variation per hole due to shorter/longer blastholes drilled. The same is true for the variation of Burden and Spacing inputs due to errors created on the XY location during manual drilling of the blastholes. Therefore, the original fragmentation profile that is dynamic in nature is modified with a mock-up statistical formula to introduce variation to the blasting parameters, burden spacing, subdrill and stemming. This means each and every drilled hole will be calculated individually based on a statistical representation of the errors realized over time as previously discussed in section 4.2. The dynamic drill hole creation is then applied with the dynamic XYZ error variation.

The output of the run for this sub model should be more or less same volume of blocks designed and scheduled. Due to this variation, however, the blasted block will have different size fraction profiles for the cumulative of the blastholes drilled. This means the fragmentation curve will be slightly flattened with increases in the amount of fines as well as boulders created within the muckpile. A normal distribution curve was assumed for the errors to be generated within the simulation model which mimics the errors during drilling location spotting as well as incorrect depths of blastholes. In Figure 152, the sub model causalities can be seen with introduced variations of the “stemming length”, “subdrill” and therefore “blasthole length”. In addition, variation of “Burden” and “Spacing” is achieved with the same hypothetical error introduction using statistical error profile.

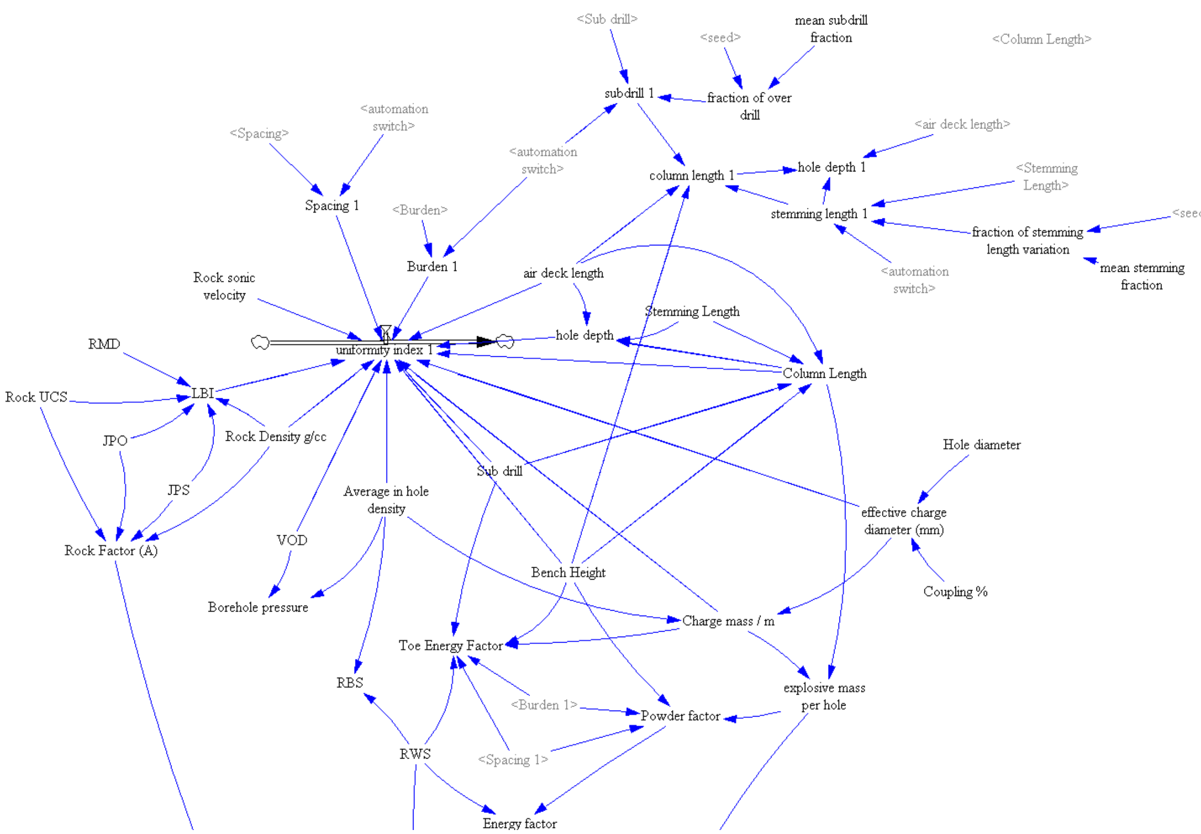


Figure 152 Variation of burden and spacing and blasthole length per blasthole calculations of uniformity index

Both the uniformity index and expected mean fragment size per blast hole would be then captured and used dynamically in the simulation. The resultant volumes are calculated per blasthole overtime with the simulated error is shown in Figure 153.

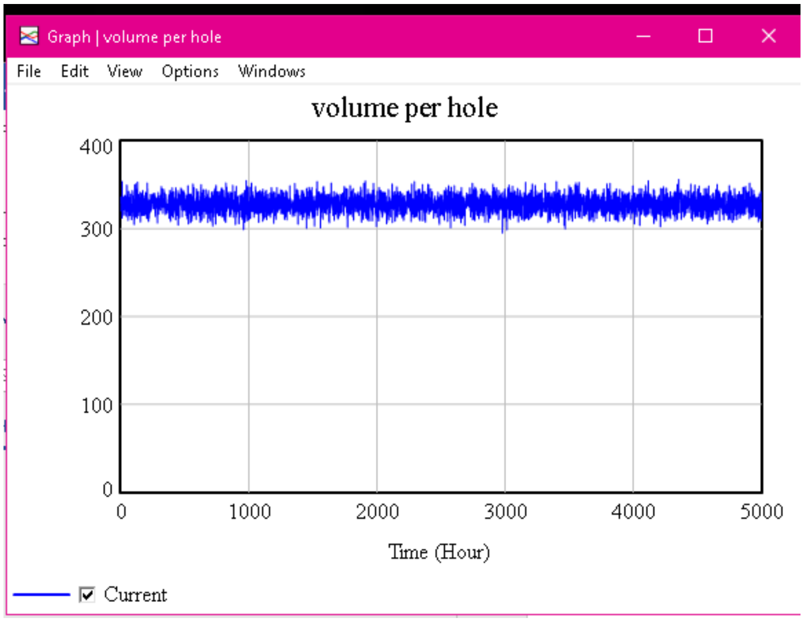


Figure 153 Blasthole volume variation due to hypothetical errors introduced to XYZ dimension

The fractions of fragment sizes per size fraction is then calculated again dynamically in a series of calculation steps as seen in Figure 154.

The output of the volume per hole over time is then obtained using a series of calculations of fraction of muckpile per size range and adding them up for the total drilled blasthole volume checking against the geometrical calculation of the full muckpile volume (in situ)

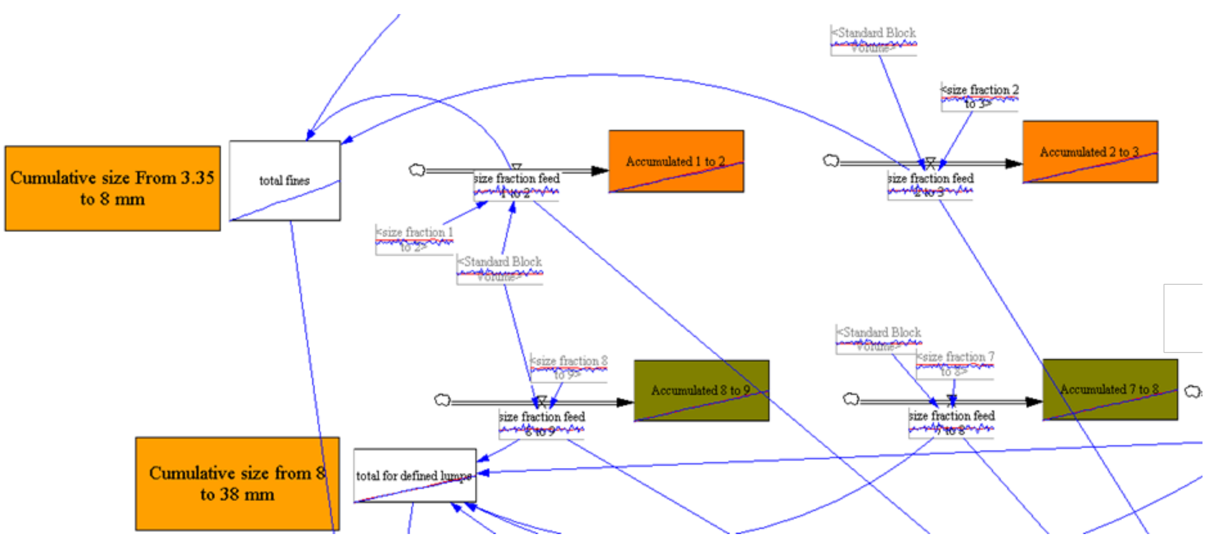


Figure 154 Screen capture of dynamic calculation of blasted volume over a set time

As discussed in chapter five the variability of the auxiliaries is created due to errors in drilling and this effect become increasingly pronounced. The planned blast design and the actual achieved changes considerably due to drilling accuracy. The deliberate error is introduced to the designed burden and spacing as well as stemming length and subdrill areas by having a normal distribution function where mean value of the error is zero and the variability range is plus minus one meter for X and Y coordinates. Similarly stemming area was designed as shorter or longer, the result also effects explosives consumption. Subdrill could disappear completely due to shorter drilling error or can be exaggerated due to drill depth variation. This also leads to multitude of problems especially charging of the blasthole not being uniform at the same horizon leading to subdrill as well as stemming areas either over charged or undercharged. In this simulation, the variability of the drilled holes' length and position effects blast volume which was fairly easy to include as an additional formulae setup. If required, the model can be modified to use tabulated values instead of statistical representation of the final drilled blasthole quality.

Due to variability, the shorter drilled holes may mean additional drilling requirement, leading to hiring contractual drill rig requirements. Assuming that the mine keeps only the designed number of drills, then the additional work created due to variability needs to be completed with contractor workers to keep up with the demand. The graphical output is obtained for the required number of holes per month for ore drilling in Figure 155 after the inclusion of hypothetical errors to the blast geometry.

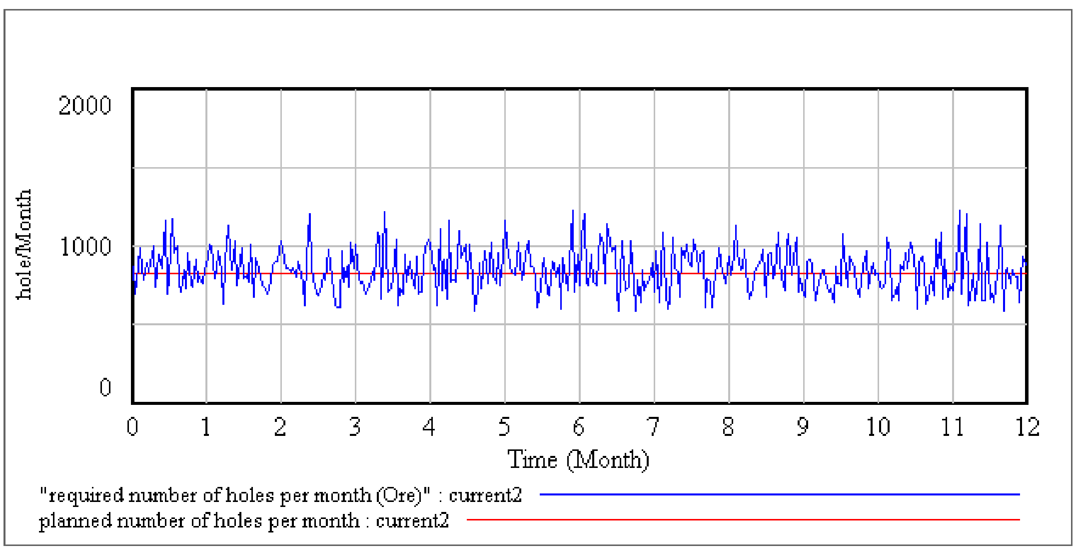


Figure 155 Variability of the required number of hole s per month due to drilling errors

This setup also compares the calculated number of blasthole requirement based on production demand with the proposed blast geometry. On average, required versus planned are the same, but what variability does is that it changes the resource requirements such as number of drill rigs. Assuming the mine has 10 drill rigs but only required nine if all is well. Due to errors introduced the number of drill rig requirements also varied considerably as seen in Figure 156.

When the auto mode is selected in the simulation assuming the correct XYZ positions are achieved in this case the number of drill rigs will be just adequate with little production stress created in the production environment (

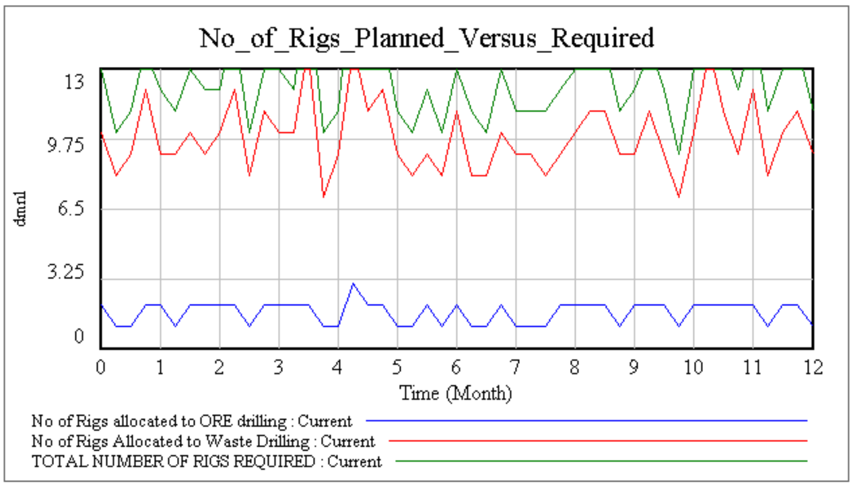


Figure 157).

Figure 156 No of drill rigs requirement simulated with deliberate XYZ errors

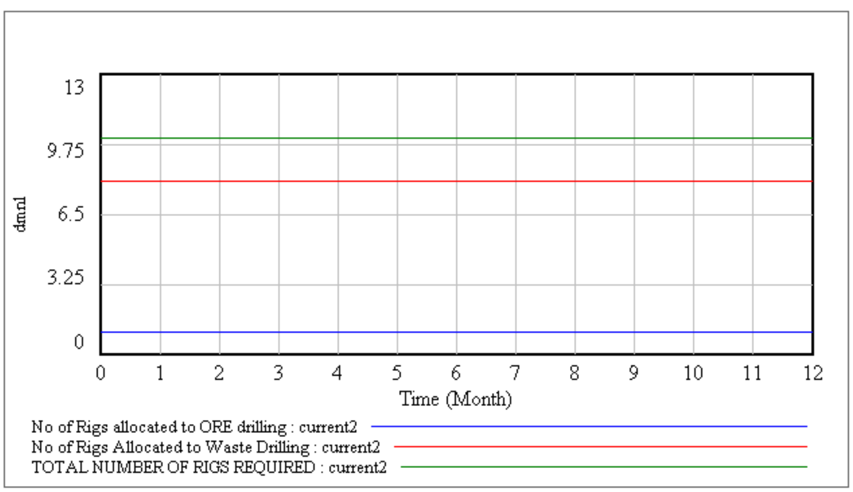


Figure 157 Number of drill rigs required when automated drill mode is on

Scenario 1: This scenario is to establish the baseline where everything is as should be based on the original design of the blast, expected fragmentation levels as well as performance for this scenario is kept constant, assuming automation will bring the variability down.

Scenario 2: In this scenario all parameters that are affected by the quality due to variations in the required parameters is simulated. To create variation of the expected/planned output some statistical variation is randomly created. For example, it is an observed fact in many mines that subdrill control is often difficult and the variations are unavoidable due to spatial elevation of the initial drill setup is difficult to estimate especially for high volume large mines. The variation is artificially created where stemming is sometimes too short and sometimes too long leading to inconsistent explosives distribution in the column as well as powder factor (explosives consumption per ton of rock blasted) per blasthole. Stemming length variations lead to either excessive boulder production or excessive fly rock and sometimes stemming ejection problems due to overcharged blastholes. Shorter drilled holes lead to miscalculation of the required explosives amount leading to overfill, therefore either no stemming (explosives filled to the brim) or little stemming. Smaller stemming length than the designed amount is also contributing to ineffective confinement. As is well known fact that unconfined blasting does not result in effective breakage and explosive energy is wasted into the air, together with ejected fly rock.

The following setup (Figure 158) is devised to create variation in the stemming as well as subdrill zones. Leading to a combination of too long or too short holes randomly for the simulation input.

The variation of the two zones in the blasting column are created using the following formulae. Also please see the output of the simulated variation graphs in the proceeding figure. On average there is no variation but the impact due to variation is doubled in terms of damage it creates in terms of the muckpile quality.

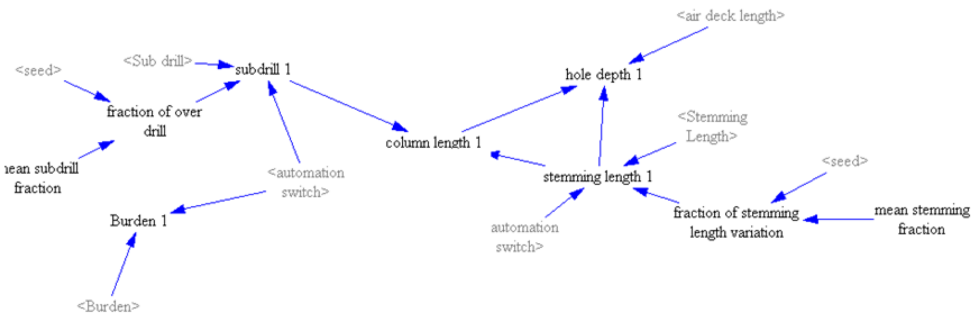


Figure 158 Input variation setup for shorter/longer drilled blastholes

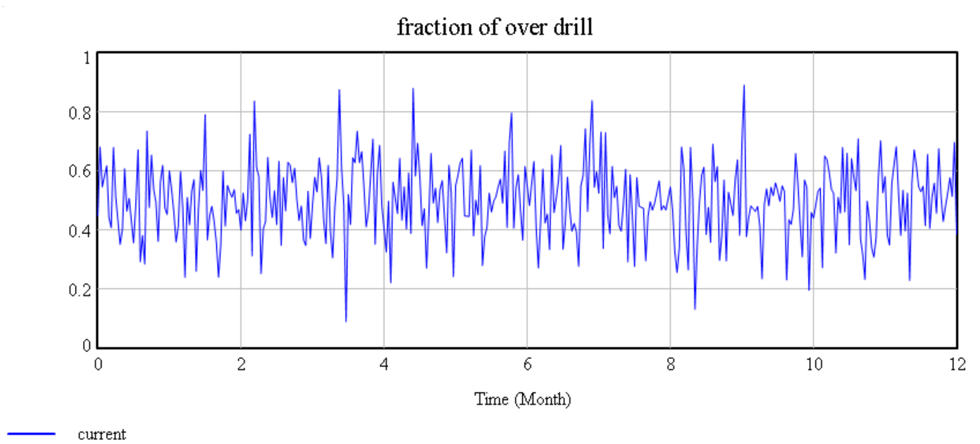


Figure 159 Simulation variation of over drill

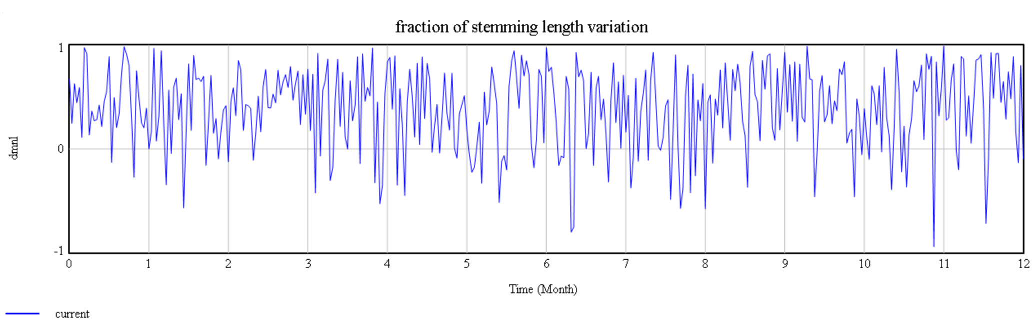


Figure 160 Simulated variation of blasthole length by manipulatinng stemming and subdrill change fraction

Once the simulation mode is entered the results for the fragmentation profile will change depending on the two scenarios: Automated or manual.

Each size fraction range of the muckpile is individually calculated and then added up to check for unity of the original blasting block volume, a portion of that setup is captured in Figure 161.

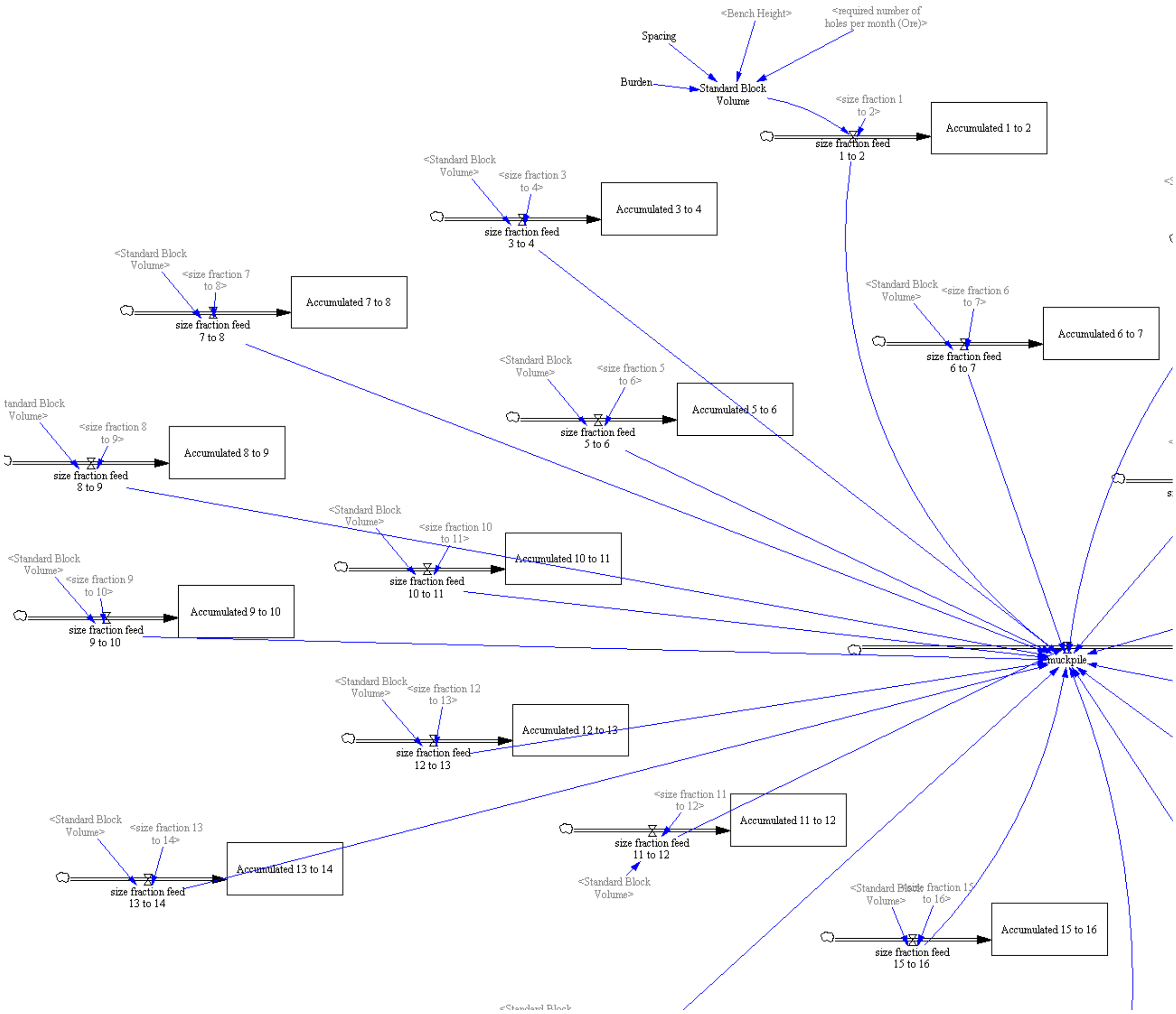


Figure 161 Volumetric calculations of the various size factions

Size fractions are selected based on the sieve sets used by the mine and setup as such for this simulation. The practical use of this portion of the model is that the calculation of the boulders as well as fines are possible to link it to the calculation of overworked and delayed blocks. It also is used to calculate the impact on the revenue due to fines generated. Finely crushed ore it means less premium on that for iron ore. Typically, fines are defined as particles smaller than 8 mm and lumpy ore range is to be within 8 to 30 mm range. This can be setup within the model depending on the mine’s definition of fine and lumpy. The fraction of that can be easily changed in the size fraction setup by dragging the slider or directly entering the number into the input box during simulation. All sieve sizes are entered in mm and the range is indicated in blue in Figure 162.

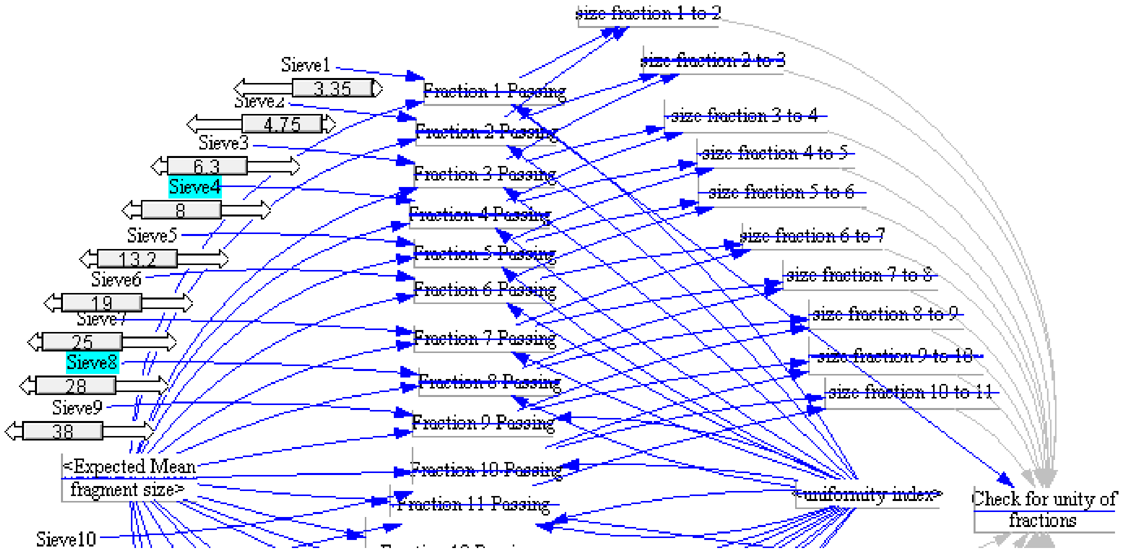


Figure 162 Sieve size ranges for ore lumpiness definition captured while in synthesim mode in Vensim

7.9 Discussion on the Results of Simulations

`The results of simulations are geared towards answering the following questions as was raised in chapter 1 previously just to demonstrate that the model is able to calculate consequences of changes in the processes. These changes can be at drilling and blasting, loading and hauling or mineral processing. Questions initially listed in chapter one is now being put in test within the model to check if the model is working within reason.

Drilling and Blasting Questions:

How does drilling cycle times affect productivity and difference of automated drilling?

The reported numbers regarding the effectiveness of GPS guided auto drilling are entered into the simulation model and certain variables are captured to show the direct impact on productivity.

The output of the simulation of the above setup is captured in the tables form within the simulation indicated as auto and manual as the monthly-time-steps. The part where the ore drill tempo and waste drill tempo are calculated is shown in Figure 163.

The most important parameter in the drilling environment is the available time per drill per time. Previously some numbers were reported for drilling related time losses, and cycle times. These are adjusted so that available drill time versus required drill time is balanced initially as per initial mine planning. The inputs used to calculate drilling time is where the author had to assume the corresponding values found in the literature as discussed in Chapter 4. They are numerous and will not be discussed individually. At Anglo American annual reports (2017), it was mentioned that daily drilling times available increased from 17 to 18. The SD model in this research gave very close figures with slight differences with auto and manual as seen in the screen capture in Figure 164. This is a reality test for the built SD model in terms of accuracy. This setup is then saved as the base setup for sensitivity runs for various inputs.

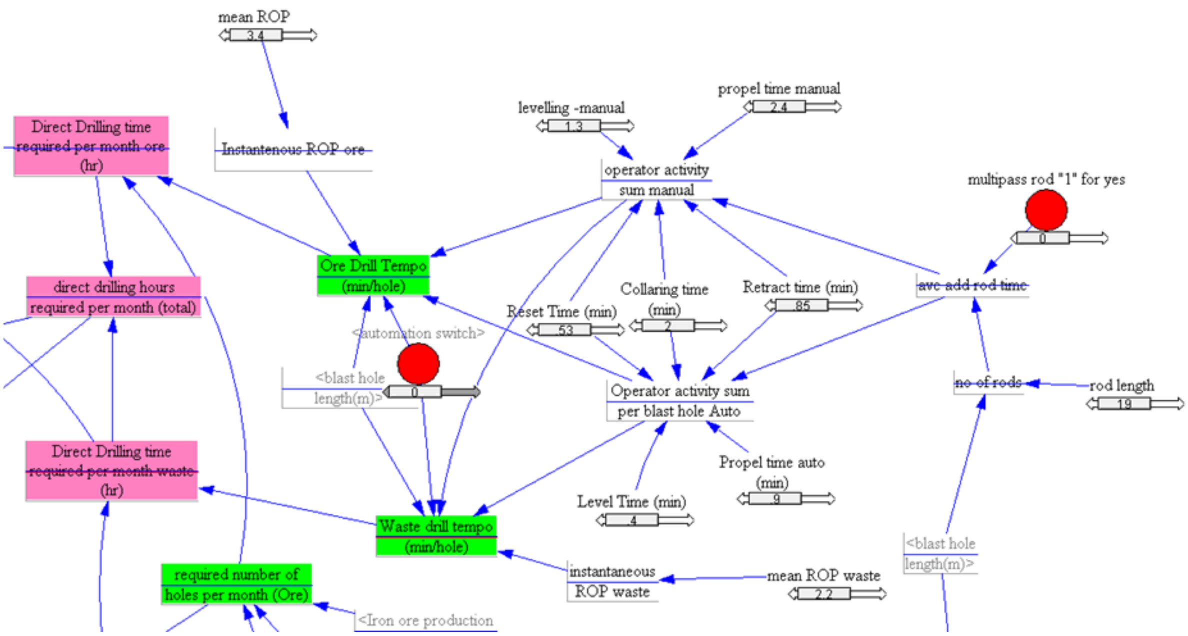


Figure 163 Ore drill tempo and waste drill tempo

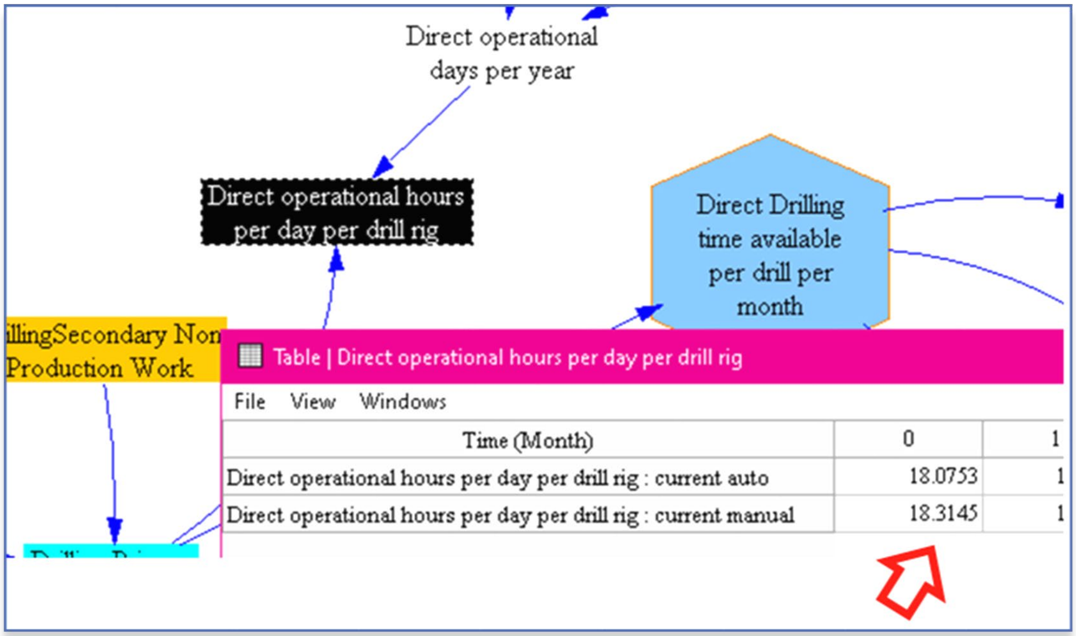


Figure 164 Simulated daily operational time per drill rig in manual and auto mode.

The second realisation of the impact of auto drilling was on the number of drill rigs allocated to drilling tasks required per year. The same simulated values are used in the setup that gives out a total of 9 drill rig requirement per year in manual drilling setup and 8 drill rigs for auto drill mode. Screen captures of these results are shown in the figures below.

Time (Month)	0	1	2	3
instantaneous ROP waste	2.2	2.2	2.2	2.2
operator activity sum manual	7.08	7.08	7.08	7.0
Operator activity sum per blast hole Auto	4.68	4.68	4.68	4.6
"Ore Drill Tempo (min/hole)"	58.08	58.08	58.08	58.
"Waste drill tempo (min/hole)"	40.08	40.08	40.08	40.
TOTAL NUMBER OF RIGS REQUIRED	9	9	9	9
"monthly Drilling Requirement m/month"	82,540	82,540	82,540	82,
Meters possible to be drilled in a year	990,500	990,500	990,500	99
Total Drilling Time required per year	46,590	46,590	46,590	46,
direct drilling time available per year	48,970	48,970	48,970	48,
"direct drilling hours required per month (total)"	3882	3882	3882	38
Direct Drilling time available per drill per month	453.4	453.4	453.4	45

<automation switch>

0

Figure 165 Screen capture of manual mode with 9 drill rig requirements

Time (Month)	0	1	2	3
instantaneous ROP waste	2.2	2.2	2.2	2.2
operator activity sum manual	7.08	7.08	7.08	7.0
Operator activity sum per blast hole Auto	4.68	4.68	4.68	4.6
"Ore Drill Tempo (min/hole)"	55.68	55.68	55.68	55.
"Waste drill tempo (min/hole)"	37.68	37.68	37.68	37.
TOTAL NUMBER OF RIGS REQUIRED	8	8	8	8
"monthly Drilling Requirement m/month"	82,540	82,540	82,540	82,
Meters possible to be drilled in a year	990,500	990,500	990,500	99
Total Drilling Time required per year	43,940	43,940	43,940	43,
direct drilling time available per year	46,110	46,110	46,110	46,
"direct drilling hours required per month (total)"	3662	3662	3662	36
Direct Drilling time available per drill per month	480.3	480.3	480.3	48

<automation switch>

1

Figure 166 Screen capture of auto mode with 8 drill rig requirements

The following research questions (Chapter 1) can also be directly answered via the simulated mine setup.

- By how much does drilling accuracy has an impact on oversize and?
- By how much drilling accuracy affect the fragmentation?
- In what ways drilling delays affect the flow of other processes?
- By how much drilling and blasting costs are affected by speed and drilling accuracy?

Loading and Hauling

Is cycle time of loading and hauling reduced due to effective drilling?

This question has two types of outputs that impact the productivity, such as the cycle time changes due to digging conditions and also the changes due to the bucket fill factor changes linked to drilling and blasting quality. The changes introduced to the model is a result of purely ineffective drill location related outcome. When automation switch is on the simulation does not include variations created into the subdrill and stemming areas, also it assumes, everything else is perfectly working as planned. These are the extreme assumptions to see the range of possibilities in the outcomes. It is almost impossible for this researcher to have all the correct parameters of the mine to have a realistic output. The model runs captured as shown below has extreme assumptions such as effect of fragmentation on the bucket filling factor as 1 when in auto mode and 0.8 when in manual mode. This range could be much smaller than is indicated in the simulated environment in the Figure 167 and Figure 168 below.

Performance of loading is linked to an empirical fragmentation factor to modify the loading cycle times and bucket loading capacities. The causalities in the mine value chain are a train of thought stemmed from the drill accuracy that is the result of stiff muckpile that leads to increased loading cycles therefore, leading to increased queues and idle times.

How does delay in loading rates affect revenues?

All consequential relationships leading to costs are built in the model to check before and after scenarios with a switch. This way the user of the model should be able to immediately see the financial implications of drill accuracy related downstream effects. In this model the only cost effects that are modelled in detailed are fuel consumption and tyres for loading and hauling processes.

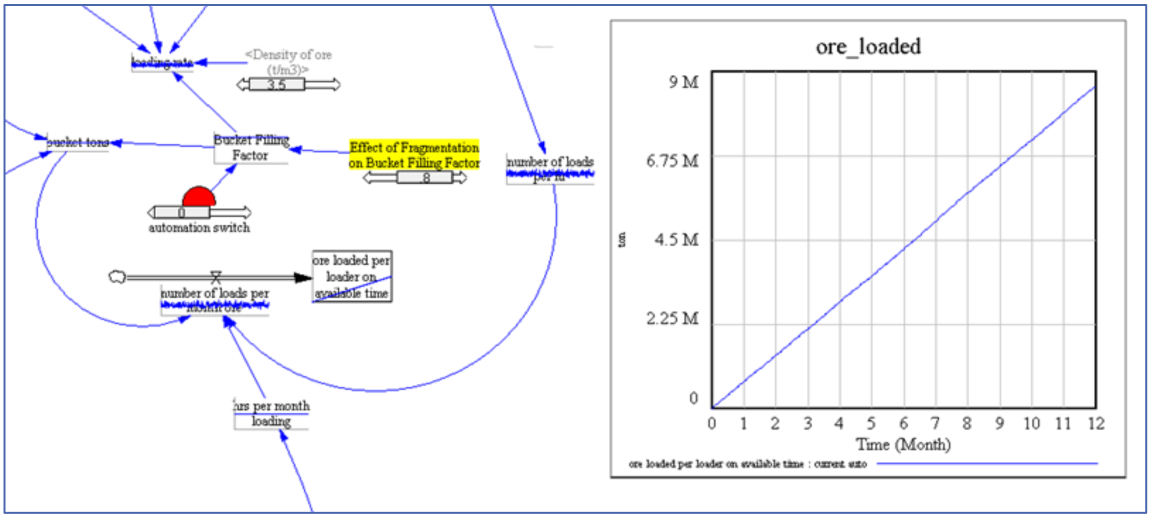


Figure 167 Amount of ore loaded when in manual drilling mode

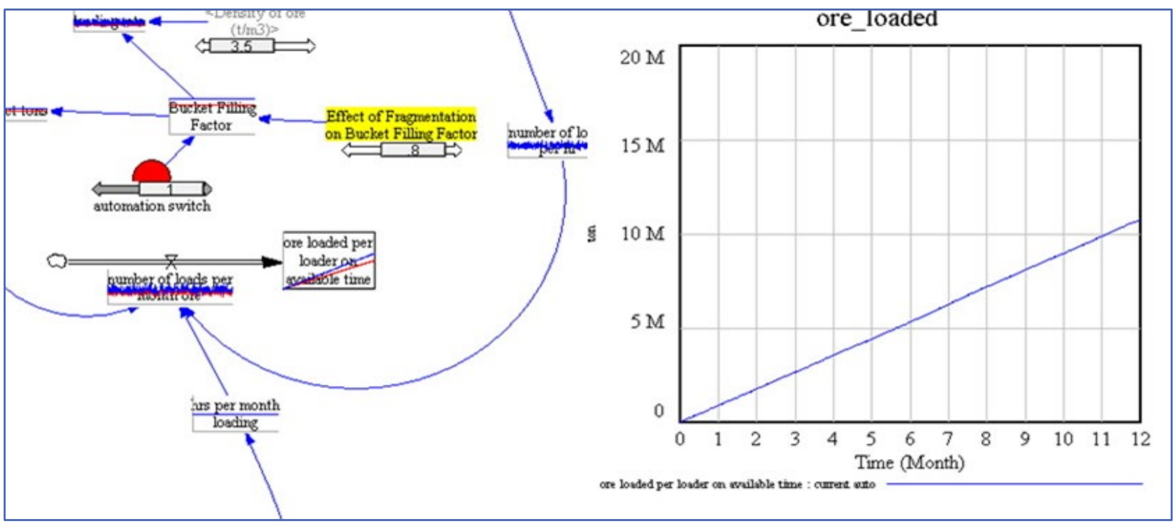


Figure 168 Possible amount of production when auto drill mode is on

How much fuel is saved by reduced cycle time?

This question has a direct answer as the costs are linked to detailed cycle time calculations. Some benefits due to less waiting times at the crusher or by the shovel are modelled. The output of the cycle times is translated into less waiting times at the loader due to reduction of cycle time.

Some attempts of setting the correct configuration of the costing structure are done for loading and hauling based on the obvious costs of consumables such as fuels and tyres. The model is not exhaustive in terms of variability of the results considering the loader capacities already accounted for, number of trucks and fuel consumption based on production achieved also change based on performance of the loaders. Therefore, the costs associated with improved cycle times are automatically determined in the simulated model and sensitive to the changes in the parameters affecting the cycle times.

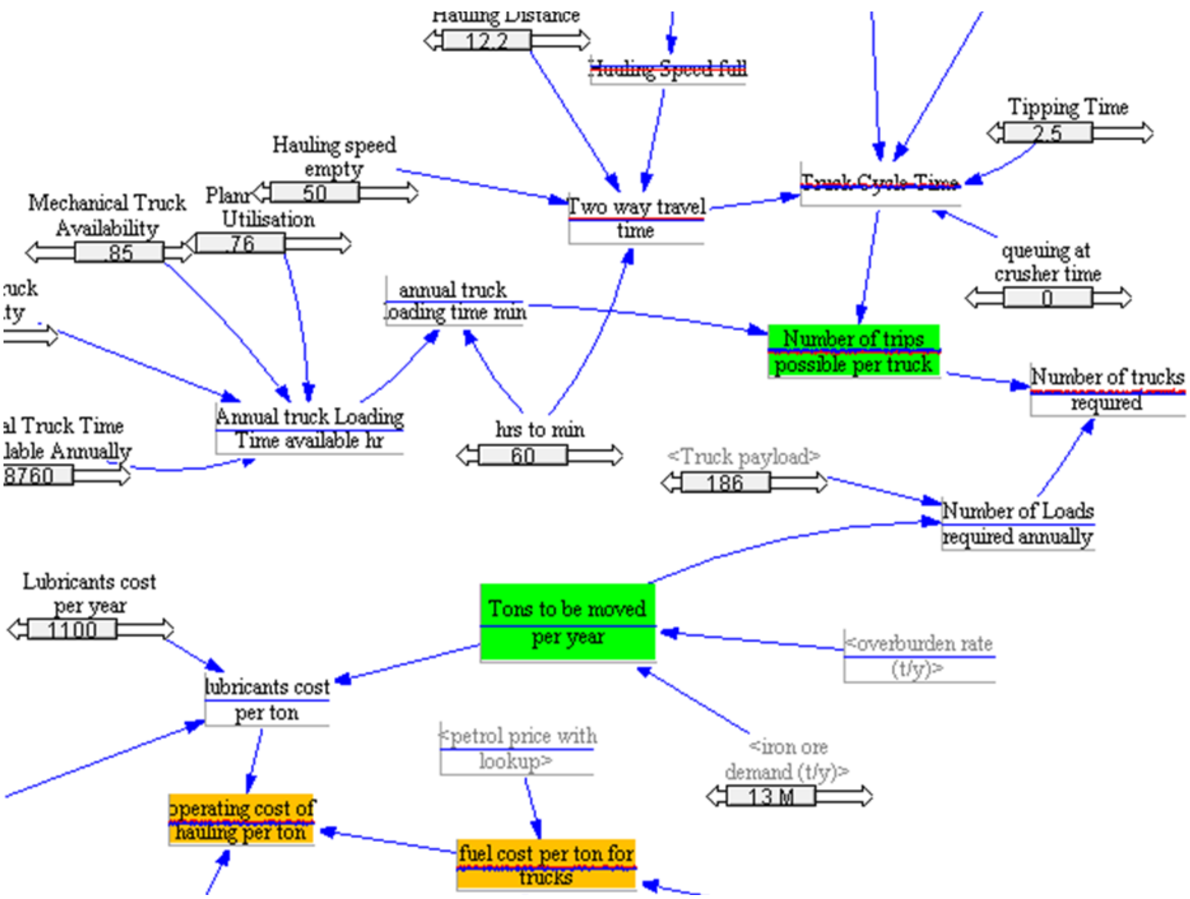


Figure 169 Loading and hauling partial simulation showing the changes in fuel and operating cost of both manual and automated options

What is the effect of fine/coarse fragmentation on revenues?

This option is relatively simple to calculate provided that correct prices are attached to lump or fine ore. The accumulations in the fines are calculated with a separate sales price to calculate the revenue than the lumpy ore. The mine has an approximate 60:40 ratio for lump to ores. Any improvements in this area would be direct difference of premium obtained from lumps to fines. The correct fragmentation prediction or measurements is however difficult. Although Kuzram is incorporated into the model to calculate fragmentation distribution per size range, it is underestimating the fines end. Therefore, this causality is not built for the time being. The confidence exists for the upper size ranges of the fragmentation curve such as P80 and it was used in calculation of energy demand for loading and hauling as well as crushing. The assumptions in Kuzram formula are multi variable and all calculations are based on one type fits all type of ore and waste geotechnical characteristics.

A more accurate fragmentation model should be linked to fines calculation to link it to revenue calculations. Ochterlony's (2009) fragmentation prediction model could be included into the model at a later stage to determine a more accurate fine fragmentation range prediction.

7.10 Drilling and Blasting Performance Observations

The model developed has been carefully put together considering the most important relationships and auxiliaries necessary to reflect the real field conditions and setup. The outcome of drilling and blasting performance is directly attributable to the drilling accuracy, correct blast design for the rock type and composition and the quality of the explosive loaded at the required quantity and correct timing. The design parameters, quantities, rock characteristics all are usable constants and auxiliaries (calculated variables) that make the drill and blast planning and optimization a little complicated as discussed previously. Once the rock blasted causalities are no longer traceable as the evidence is destroyed by blasting. Therefore, a good capture of the history leading to blasting events is necessary to attach the results to the causes, such as powder factor. Trial and error in blasting is therefore a costly exercise. It can be said that empirical formulae, simulating the blasting within some confidence, is therefore essential in the model. Estimation of the quantities and costs can be based on empirical or real values and is a matter of user's preference.

Causalities of the input parameters for blasting can be traceable by clicking on the parameters and querying the causes or uses trees of the parameters entered into the model (Figure 170).

The effect of number of blastholes per block for the same geometrical blast design pattern is simulated as seen in Figure 171 at the selected number of holes the allocated drill rigs are causing production blocks being queued (green free hand circle) and Figure 172 when 3 additional drills are added the graphs showing "planned blocks queued" is flattened. This shows the model is setup correctly at the casual relationship level.

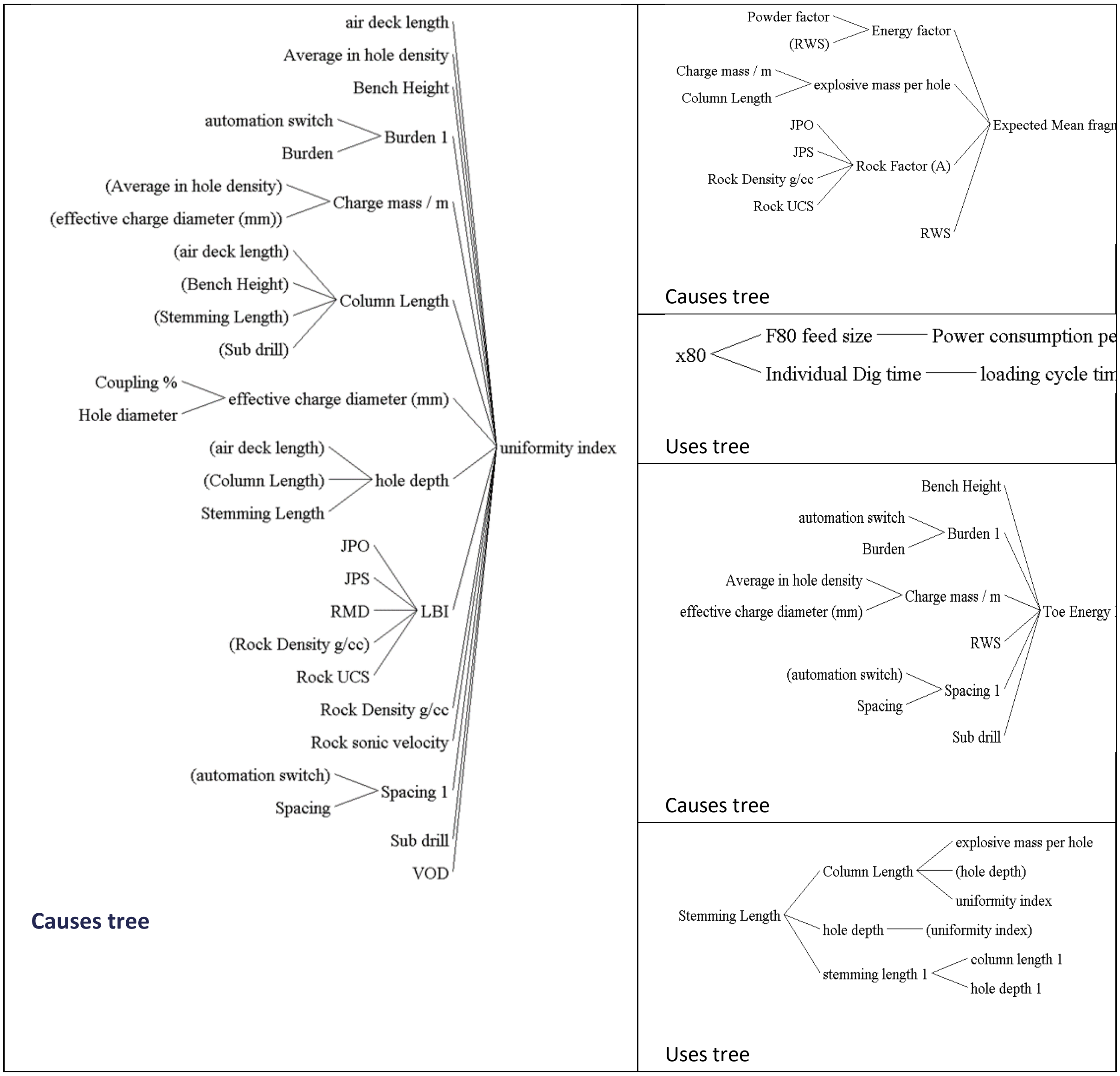


Figure 170 Causes and uses tree examples for blasting related variables

These simulation figures demonstrate how to interact with the model to test and experiment the output.

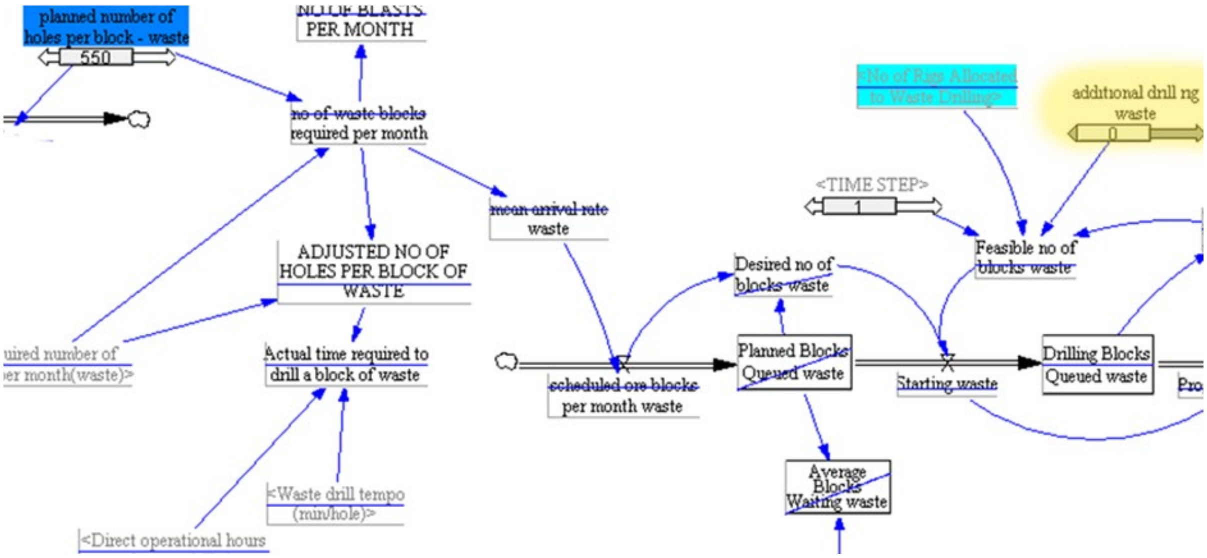


Figure 171 Number of blocks with allocated drill rigs as per schedule

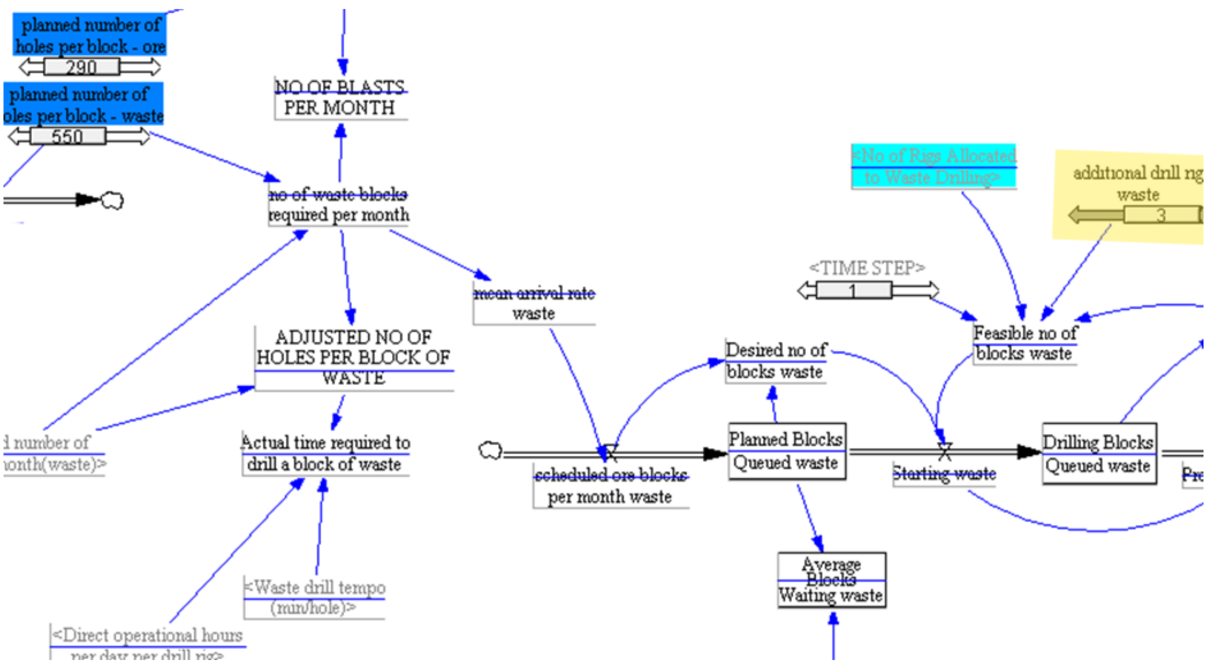


Figure 172 Number of blocks balanced with 3 additional drill rigs

A drilling block's journey will depend on the drill rates and some other delays from start to progress and to the state of blasted muck pile. A drill block is delayed for a fixed time for each process that it goes through. The delay time amount is determined by the cycle times of that process. The function used for this purpose is Fixed Delay (Figure 151). The results of the simulation would only match a specific mine provided the precise cycle times captured during field measurement used in the SD model.

The mining complexity reaches a steady state production despite the variabilities in the processes as they are adjusting resources and rates to changing conditions and requirements of the demand. Which is really to maintain the production at whatever cost. Sometimes at the cost of hiring additional resources or contractors. This is called a state maintaining condition. Therefore, mine management will be using additional resources to balance the production requirements. An additional resource option per process is added to level the rates to a flat line. This can be considered as an additional intervention option to the user. This is demonstrated in Figure 172 by increasing the number of holes per blasting block to be drilled. This obviously will cause stress on the system and results in queues graph showing an increase, since the available resource is limited, additional drill rig needs to be used to flatten the curve at planned blocks queued waste. By adding 3 drill rigs to the waste block the curve will be flattened, therefore balancing the system. The stress created at the drilling process is due to the demand of loading process. And this has been successfully demonstrated in this simulation exercise.

The number of drill rigs required for ore and waste is calculated separately. When variability at the processes increase the number of required resources to keep up with the demand also increase.

Based on the setup of the time losses and drill inaccuracies the output changes considerably. The simulated results on manual and auto are captured for a weekly schedule and can be seen in the figures Figure 173 and Figure 174. Due to variable quality of drilling the number of drill rigs required also is variable, on average the same production output is achieved. But in reality, resources allocated are in limited numbers. This causes stress in the production environment. The auto mode assumes that the variability will be reduced due to accurate and on time drilling as planned.

Time (Month)	0	0.25	0.5	0.75	1	1.25	1.5
instantaneous ROP waste	2.2	2.2	2.2	2.2	2.2	2.2	2.2
operator activity sum manual	7.08	7.08	7.08	7.08	7.08	7.08	7.08
Operator activity sum per blast hole Auto	4.68	4.68	4.68	4.68	4.68	4.68	4.68
"Ore Drill Tempo (min/hole)"	60.97	61.02	61.26	60.41	58.17	59.55	60.15
"Waste drill tempo (min/hole)"	41.95	41.98	42.14	41.59	40.14	41.03	41.42
TOTAL NUMBER OF RIGS REQUIRED	13	10	11	14	12	11	13
"monthly Drilling Requirement m/month"	119,400	95,800	105,900	133,900	110,900	104,300	120,200
Meters possible to be drilled in a year	1.433 M	1.15 M	1.271 M	1.607 M	1.331 M	1.251 M	1.442 M
Total Drilling Time required per year	66,790	53,570	59,190	75,010	62,580	58,570	67,400
direct drilling time available per year	68,540	53,930	58,790	72,470	63,270	58,530	68,540
"direct drilling hours required per month (total)"	5566	4465	4932	6251	5215	4881	5616
Direct Drilling time available per drill per month	439.4	449.4	445.4	431.4	439.4	443.4	439.4

Figure 173 Production output and drilling productivity figures when in manual mode

Time (Month)	0	0.25	0.5	0.75	1	1.25	1.5
instantaneous ROP waste	2.2	2.2	2.2	2.2	2.2	2.2	2.2
operator activity sum manual	7.08	7.08	7.08	7.08	7.08	7.08	7.08
Operator activity sum per blast hole Auto	4.68	4.68	4.68	4.68	4.68	4.68	4.68
"Ore Drill Tempo (min/hole)"	59.08	59.08	59.08	59.08	59.08	59.08	59.08
"Waste drill tempo (min/hole)"	39.88	39.88	39.88	39.88	39.88	39.88	39.88
TOTAL NUMBER OF RIGS REQUIRED	11	11	11	11	11	11	11
"monthly Drilling Requirement m/month"	113,800	113,800	113,800	113,800	113,800	113,800	113,800
Meters possible to be drilled in a year	1.365 M	1.365 M	1.365 M	1.365 M	1.365 M	1.365 M	1.365 M
Total Drilling Time required per year	60,130	60,130	60,130	60,130	60,130	60,130	60,130
direct drilling time available per year	61,820	61,820	61,820	61,820	61,820	61,820	61,820
"direct drilling hours required per month (total)"	5011	5011	5011	5011	5011	5011	5011
Direct Drilling time available per drill per month	468.3	468.3	468.3	468.3	468.3	468.3	468.3

Figure 174 Production output and drilling productivity figures when in auto mode

7.11 Loading and Hauling Performance

When initial mine design performed an operation would have invested in a loading machine matching the production demand of the mine annually. Haulers are then matched to the size of the loading machine in terms of payload per hauler trucks. There is a fine balance between the bench height, blasted muck pile shape and height to that of loader therefore haulers.

The model demonstrates firstly selection of the loader matching the production profile. Based on the loader selected there is an optimum truck payload matching that loader bucket size and cycle times. For a generic model, this option has been built in for calculation of the number of trucks and cycle times to perform on time movement of ore and waste to the crusher and waste dumps, respectively. For the simplicity, the distances are fixed in the model but can be modified with statistical values obtained from a mine to simulate variable distances performed per truck per time if the purpose of the study calls for it.

The model is modified with some additional influences to realize the influence of drilling and blasting quality on the loading and hauling performance. The most obvious ones are:

a) Queuing times at the trucks at the shovel
b) Bucket capacity utilisation due to fragmentation
c) Cycle time of loading due to digging conditions

There is a reported empirical relationship between the fragmentation size and loading cycles and this has been included in the model. Choudhary 2019 reported an optimum mean fragment size required per bucket capacity. There are also those field measurements that represents actual situation of a specific mine.

The case study mine reported (Modern Mining, 23 July 2018) that queuing time at the mine is improved as it was reported that queuing times reduced from 5.7 to 4.9 at the loader assuming this is due to shovel loading efficiencies linked to improved fragmentation of the muckpile. These numbers are incorporated into the model linked to automation switch function.

Bucket capacity is also modified which is linked to level of influence per optimum fragmentation Figure 175. The optimum fragmentation size per bucket type is already a fixed number. The user can define the bucket capacity performance based on the actual versus should be mean fragment size. Since the mean fragment size is automatically determined with empirical formula it can be linked to the loading performance.

There are no real measurements found cycle time of loading due to digging conditions. As a precaution the user is given the option to modify this input as a percentage value. For initial setup this is set at 80 percent efficiency due to increased effort required when in manual mode and 100% efficiency when all is according to plan. The accuracy of the loading conditions setup is currently not perfected but it has the flexibility to match the case study related field measurements.

Based on this initial setup the production capacity of loader is demonstrated with a graph that shows amount of tons loaded per time. The profile changes automatically when the bucket filling factor is changed.

The loading section of the simulation is not directly linked to the actual but to the designed parameters. This is a section that needs to be further developed in the future to automatically respond to what is happening in reality in simulated terms. At this stage it is left to the user’s discretion of input values as per the field measurement.

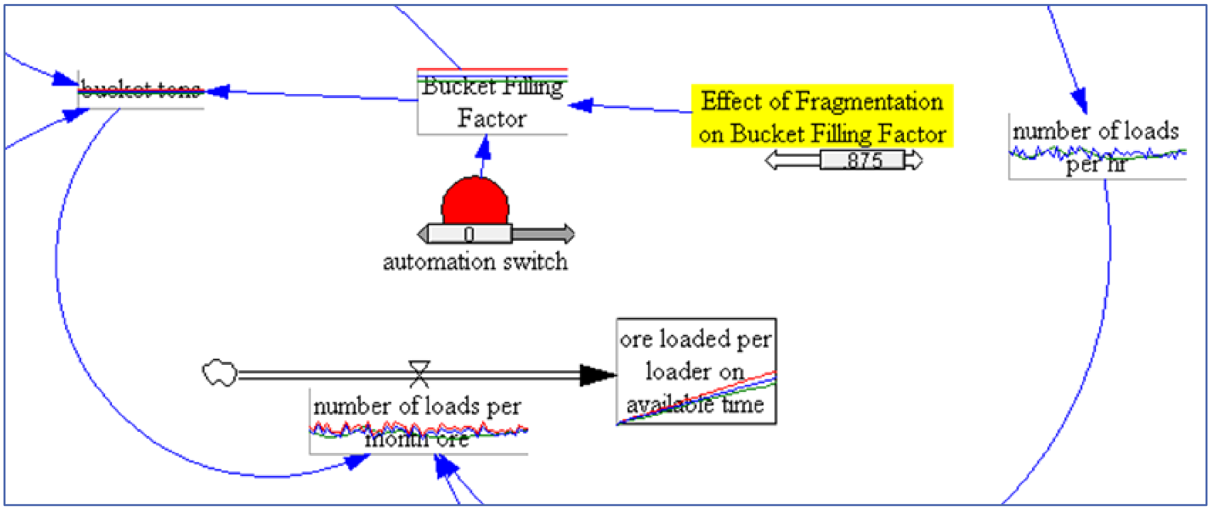


Figure 175 Effect of fragmentation on bucket fill factor.

The loading machine can also have the direct loading time available calculated specifically for that machine similar to the drill rig calculations. The values attached to each parameter for loading is hypothetical and not linked to any literature based study since the focus of the case study is on the drilling performance.

The user has the option of either using actual field measurements for loading cycles or using planned cycle times for loader based on matching loading cycle. The actual loading cycle time is paired with dig time determined based on X*) fragmentation value which is discussed in the fragmentation model section of this thesis. The choice is managed by a built in switch to be set by the user (Figure 176)

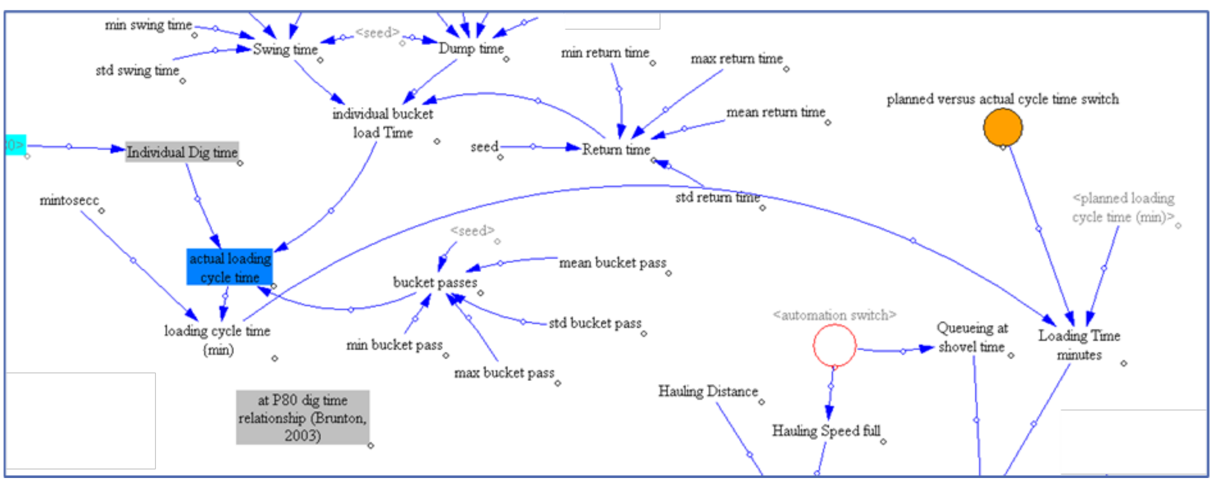


Figure 176 Planned versus actual loading cycle time paired with fragmentation linked dig time.

7.12 X80 Run of Mine Size Based Calculations

X80 can be defined as the 80% passing size when a muckpile is sieved. The 80% passing size here in this thesis is referred to as X80 and sometimes it can also be referred to as F80, where F stands for Feed. F80 feed size is determined either empirically or from photographs of the muckpile or at the crusher during dumping of the ore into the crusher as seen in the picture Figure 141.

F80 fragmentation is being used for power consumption calculations as discussed in section 6.3.3. The numbers used here are the ones determined for the AITIK mine. The inputs for the case study do not exist at this time. This requires a field study which is intended for future work. However, for the sake of demonstration, to include in the simulation model, the existing model is using the values for the Aitik mine as described in section 6.3 of this thesis. This in turn will be used to answer the following research questions:

1. How much of the crusher energy consumption can be attributed to fragmentation?
2. Crusher liner costs due to fragmentation related wear and tear
3. Bucket fill factor based on fragmentation class input
4. Queues in the crusher and effect on overall fuel consumption and cycle times due to idling of haulers

The following lookup tables; Figure 177, Figure 178, Figure 179, Figure 180, Figure 181, Figure 182 are included for determination of power draw, crusher throughput and shovel factors fragmentation class based on F80 feed size.

Figure 177 Crusher Power draw detemination- lookup table setup based on F80 class- formula view

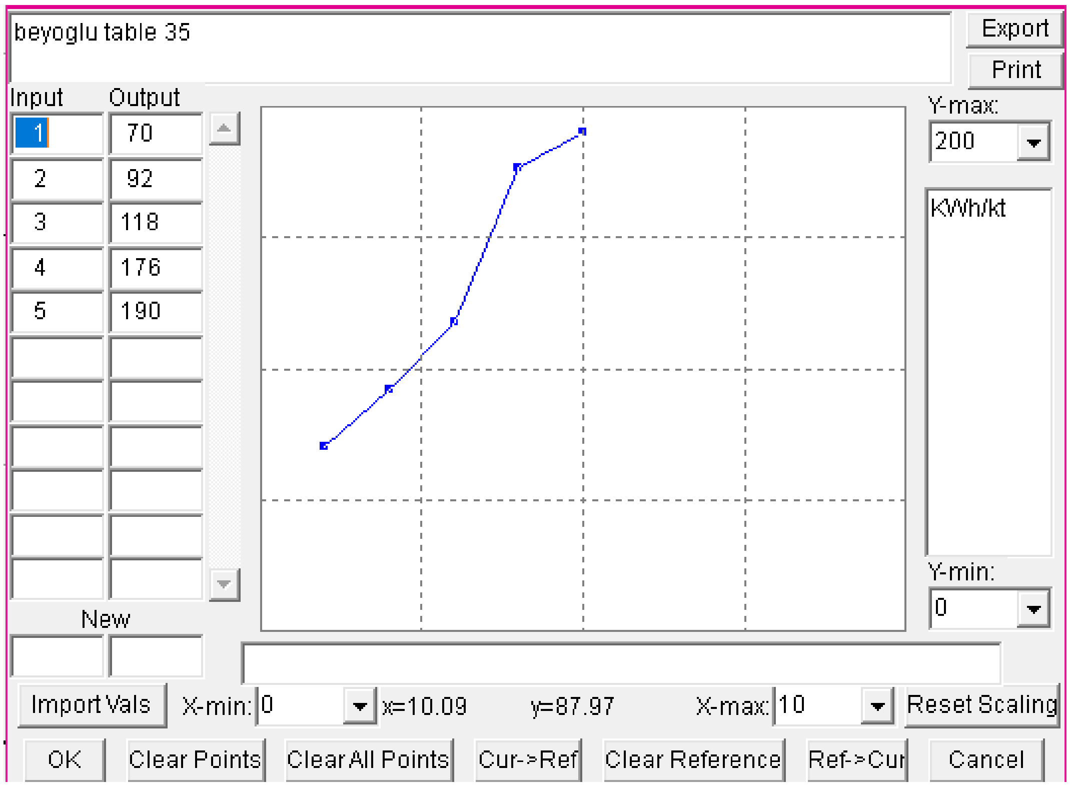


Figure 178 Crusher Power draw detemination- lookup table setup based on F80 class - graph view

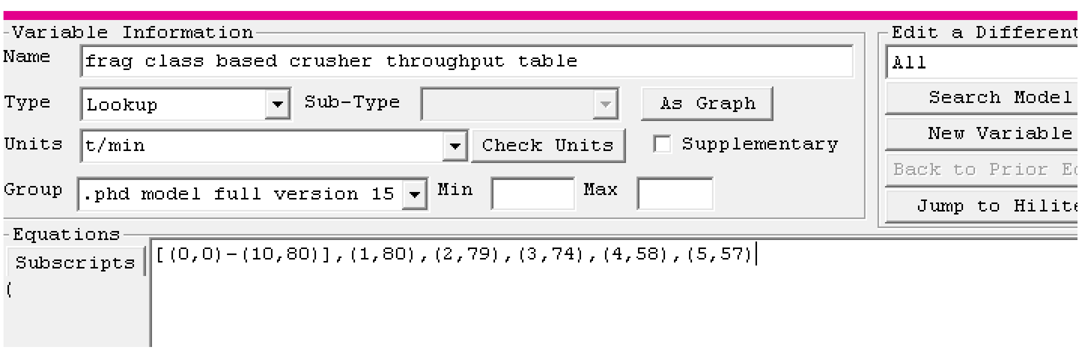


Figure 179 Throughput detemination- lookup table setup based on F80 class - formula view

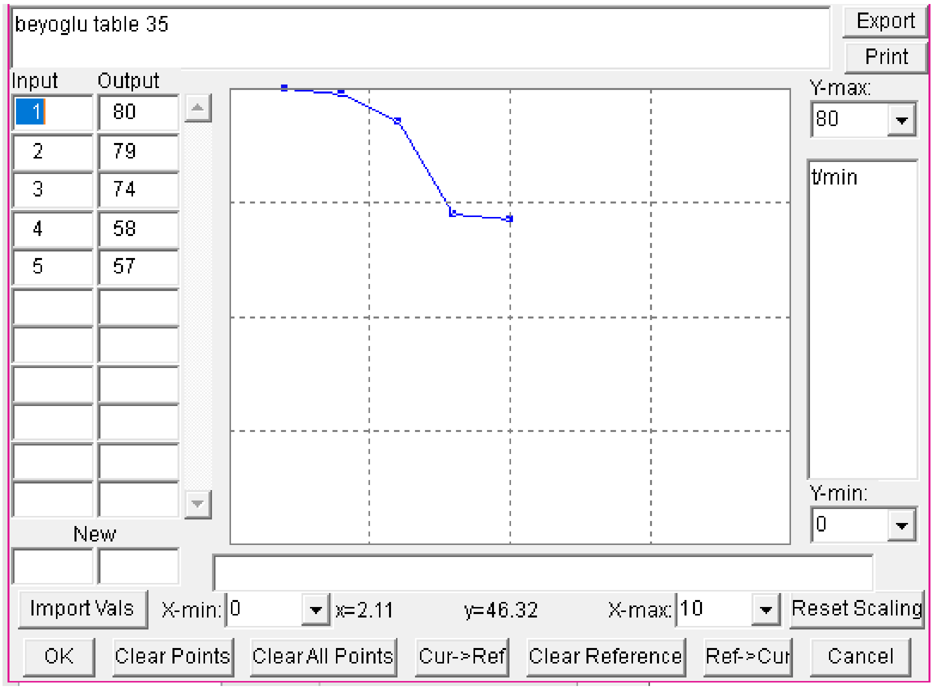


Figure 180 Throughput detemination- lookup table setup based on F80 class - graph view

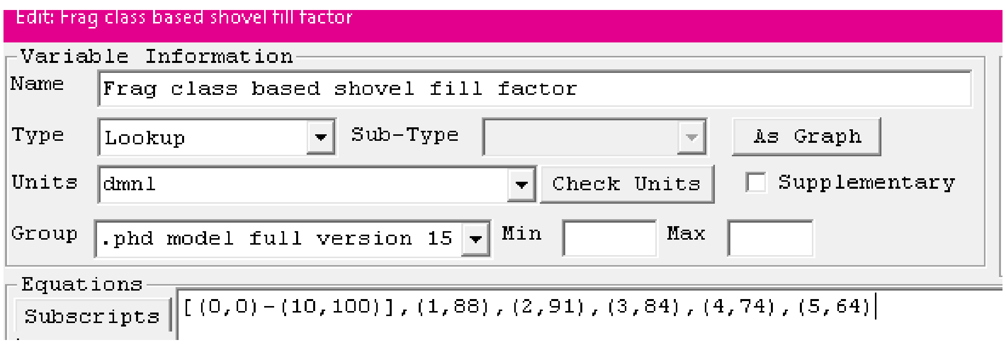


Figure 181 Shovel fill factor determination – lookup table setup based on F80 class - formula view

The runs based on these tables are not included as yet in the determination of energy consumption linked to fragmentation. Further studies are necessary whether this would be meaningful way of linking power draw of crusher via fragmentation classes therefore drilling efficiency.

A simple run on the financial revealed significant cost reductions when the automation switch was one. The result is not the total cost of mining but based on simple consumables such as fuel, tyres, drill consumables. The outcome of the simulation can be seen in the graphs as below. The change in cost does not seem to be significant. But it seems to be on the right track in terms of quantification efforts.

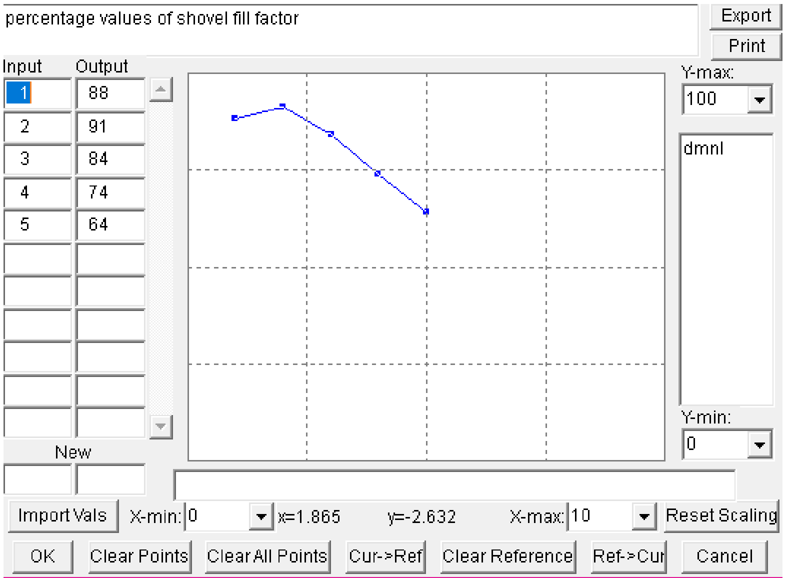


Figure 182 Shovel fill factor determination – lookup table setup based on F80 class - graph view

"If the building blocks are so shabby, is it worthwhile building integrated models at all? The answer is clearly yes, despite the present weaknesses of the models. The reason is that modelling forces us to reveal our assumptions and changing those assumptions show how important they are with respect to the outcome (Toth, 1995)". Toth is right to say that even if the building blocks are shabby total mine approach is going to pave the way to more accurate and detailed models.

It was interesting to note how sensitive the model was towards the determination of available direct drill time per year and the effect of parameters influencing the drill time. It could be an eye opener to the mining engineers that even a simple hot seat change time gives considerable gains in the production momentum.

The intention of this thesis was not to give a perfect answer for a specific mine regarding calculation of direct benefit/loss due to changes introduced but to have a tool that can do it. Any tool is as good as the inputs and relational formulae behind the tool setup. Therefore, the objective of having answers to most of the questions listed above has been achieved.

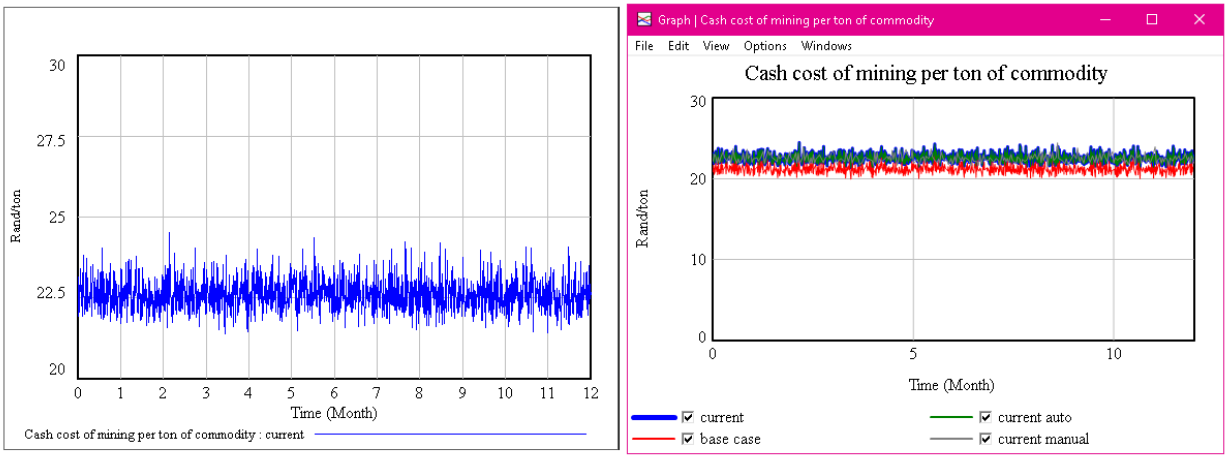


Figure 183 Cost of drill and blast changes due to imroved drilling cycle times and the resultant savings per ton

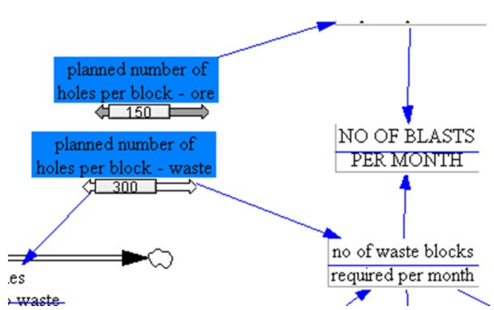
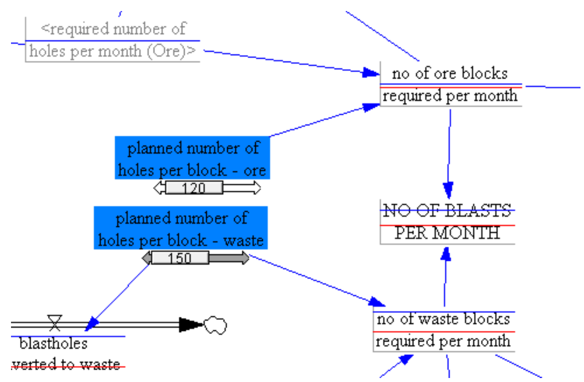
Scenario with more holes per block	**Scenario with less holes per block**
	
No of Holes Ore 150 No of Holes Waste 300 No of Blocks Total: 21 Lost production time due to standby: **32hr/month**	No of Holes Ore:120 No of Holes Waste: 150 No of Blocks:38 Lost production time due to standby: **76hr/month**

Figure 184 Drill block size impact on loss time

The simulation tool constructed during this study can be considered as a prototype. A more detailed mine setup can be geared towards a specific question or can be used as a generic quick mine planning tool to observe the variations in the totality due to parametric changes even in the deepest branches of an operation. The model created in this thesis opens the way to a world of possibilities in terms of connectivity and interaction of the different mining processes and the consequences thereof.

7.13 Quantification of Effects of Fragmentation

The empirical model created using Kuzram is firstly put to test if it creates meaningful output after adding random error margins to ‘burden’ and ‘stemming’ variables that follows a normal distribution. The output for P80 versus ‘burden’ and ‘spacing’ can be seen in the graph Figure 185. It shows that P80 value is proportional to Burden, and it increases as burden increases, which is meaningful since explosive energy radially expand away from explosive column and the fragmentation concentric are also radially increasing in size. However, we cannot say the same for stemming since the upward distribution of fragmentation should have no meaningful impact on the fragmentation distribution due to explosive energy being constant along the stemming axis this is also somewhat not reflected in the graph. But since stemming length is linked to ‘burden’ as a formulation then it makes sense to have an increasing trend. When errors are introduced the ‘stemming’ length, calculation based on ‘burden’ also changed but should actually stay constant. This alerted the author to the fundamental errors that could arise if not carefully thought about the way the model is constructed. Confidence in the setup of the formulation and output now exists based on this small exercise. And after the modification of the model in terms of not to make the stemming dependent on the burden but use the original stemming length planned in the setup the output of the graph changed as it should be. The error originated from the Stemming=Burden*0.7 relationship. If Burden changes so does stemming, but they should have been independent. Since stemming area theoretically should stay independent of the errors introduced to burden. The results are as follows as expected the stemming versus X80 stays constant.

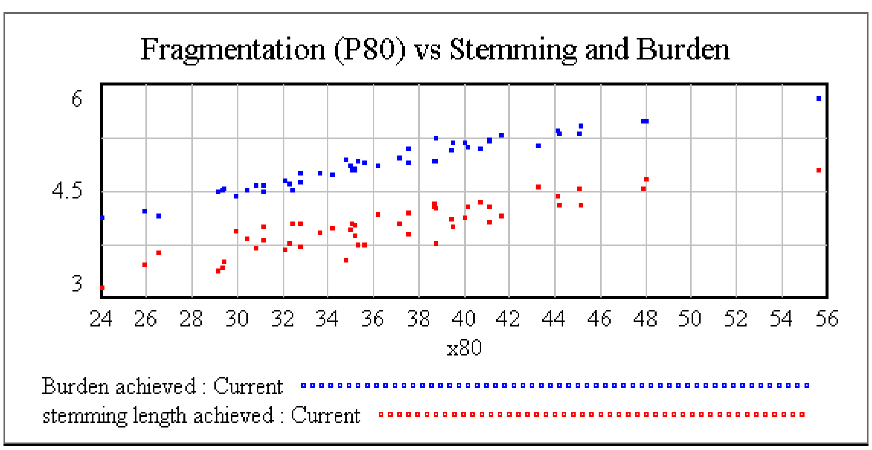


Figure 185 Faulty behaviour of P80 fragmentation versus burden and spacing after introduction of errors

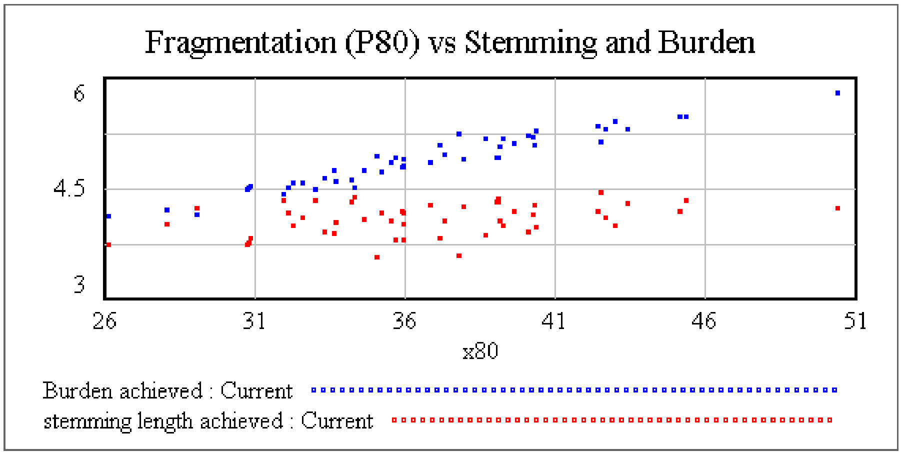


Figure 186 Correct behaviour of the X80 versus burden and stemming

If a user of the model wishes to see how P80 size influence the loading, hauling and crushing processes some manipulation to drilling and blasting parameters can be done during ‘synthesim’ mode. The budgetary impact in terms of additional costs incurred will be immediately visible as well. The P80 variability is achieved if an increase or decrease in the drill pattern parameters (burden and spacing and stemming) causing the area where most boulders are produced shrink and expand therefore P80 becomes adaptive to the changes on those drill pattern parameters. Similarly, some other parameters that are thought to be direct outcome of automation can be tested.

7.14 Quantification of Economic Benefits of Improvements in the Performance of Unit Processes

The only output that one can immediately realize in the model is the cost of mining component. This is the direct savings area.

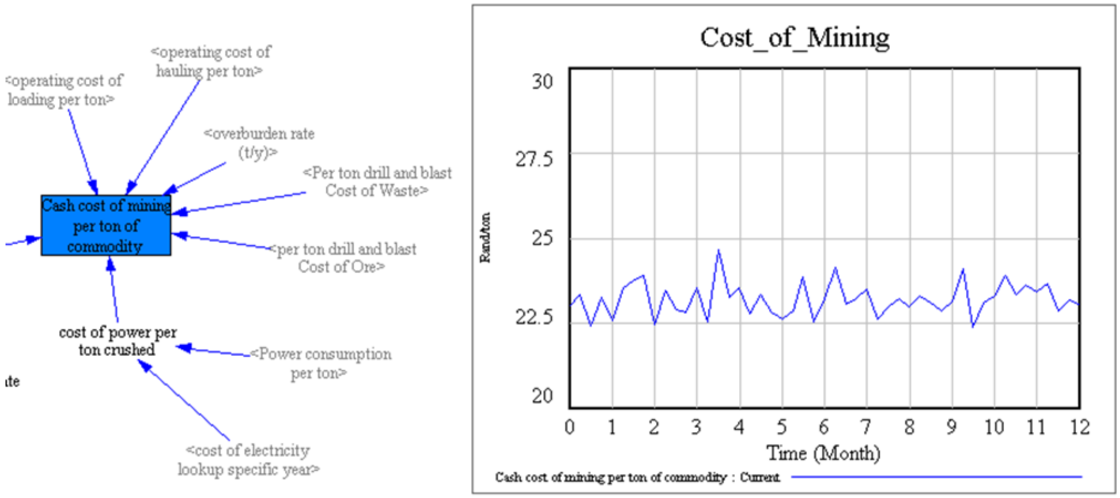


Figure 187 Mining cost graph in when manual drilling mode in the model setup

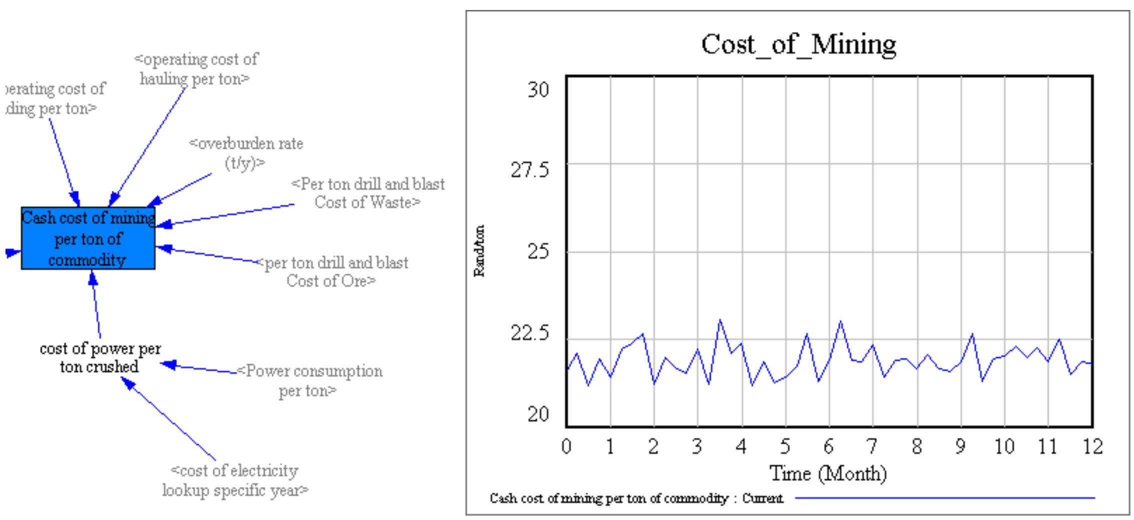


Figure 188 Miining cost graph when in auto drill mode in the model setup

The figures above (Figure 187 Figure 188) reflect before and after costs of drilling blasting and hauling. Loading is not included in the calculations for the above runs. The difference of cost benefit from before and after scenarios can be calculated based on the yearly cumulative. Obvious direct benefit of the costs is calculated as follows:

- Mining cost manual mode: 16.1 billion rands/year
- Mining costs auto mode: 15.49 billion rands/year
- Savings: 610 million rands/ year

The SD model developed here is a generic one that is applicable to any open pit mine where similar mining machinery is being used. Therefore, with the correct values used it can be used as a generalized surface mine model. The modeller however needs to be aware of the statistical definitions used in this study and modify as such to fit the model to the specific behaviours of machinery along with other mine specific parameters.

7.15 Chapter Summary

The chapter explored the contributing factors to the problem using the model developed. It also assessed the intended and unintended consequences of the technological changes introduced into an existing mine environment. The step by step approach from definition of the behaviour to the building of the total model required understanding patterns of behaviour. A value attached to automation related gains or losses was captured in this chapter for demonstration of the capability of a system dynamics model applied to a surface mine.

The next chapter will highlight the successes, challenges and future outlook revealed during the research carried out in this thesis.

"The matter is not too deep, but the ideas are often too shallow"

Chapter 8

8 Summary and Conclusions

8.1 Introduction

A new prototype system dynamics simulation model of a surface mine has been created using Vensim in order to meet the objective raised in Chapter 1 where the objective was to model the dynamic behaviour to study the behaviour and the consequence of a disruptive change to an existing system.

This section will highlight the important steps taken towards developing the system dynamics model and also discuss the contribution of this research in terms of quantifying the consequences of changes introduced to the mine parameters over the total mine value chain.

8.2 Research Questions and Resolutions

The mines may have well established systems in place for daily running of the operation which are generally run in various departments dedicated to each unit process. The interaction of the people and mining machinery within various departments have a combined impact on the overall performance and therefore revenue. The mining style has changed over the years adopting to new technology especially due to digital revolution popularly known as 4th industrial revolution. The pace at which the technology is being adopted also requires a quick adaptation to the changing work environment in terms of the policies but quantification of the impact of change is not easy with a traditional linear approach any longer when a variety of parameters are added to the existing due to technological interventions, installations or replacements. Therefore, the system dynamics modelling solution was selected to model such changes in the performance and behaviour of mine systems geared towards the research objective which is not necessarily easy to quantify with a linear spreadsheet type of application.

The purpose of the model was to demonstrate the consequences of technological changes to the various mining parameters by studying the causality of the processes with the help of a system dynamics method selected after an extensive literature review. From this objective stemmed some immediate questions that can be answered by the system dynamics model in a game like simulation environment. The questions raised in Chapter 1 was answered in chapter 7 in detail.

The main unit processes included in the simulation model are drilling, blasting, loading, hauling, and to some degree the crushing for capturing the variation in system behaviour while automatically capturing the cost and its variation over the time. The fragmentation profile of the mine was the most critical part of the modelling as it links all the main mining processes in terms of performance and fuel costs. In the meantime, time losses were also modelled where all performance criteria were defined to describe the behaviour of mining machinery under the changing circumstances due to disruptive changes. Mining machinery and operational performance measurement criteria were extensively covered in chapter 4.

The systems dynamics modelling is not new and has been used by many researchers including those in the medical field. Various industries use system dynamics, and the approach is generally similar since system dynamics is flexible enough to adapt to any process and condition provided there is a meaningful fundamental relationship behind each parameter observed.

The resolution was to create a model that will mimic the mine system based on the available data but not to have an "as is" hard data profile of a mine but to model at a reasonable abstraction level. Various data sources are being used to check the model's integrity of the results including case study related publications, similar mines with similar working environments, etc. The more the mining value chain gets complex the more data is created than it was before. It is not practical to model a mine with every small detail. There must be boundaries or summaries of processes. Therefore, data that is concise, but representative was essential for the system dynamics modelling

The conceptualisation and formulation of the system dynamics model is much wider than the numerical databases often used in operations research and statistical modelling. The modelling started with initial quantified values for the variables and stepped up from the simple to the complex while running validity tests in terms of results obtained at the simplest stage before building to the next step. The system dynamics model created using Vensim is by no means complete in terms of a true world reflection; however, it can be built on with further detailed formulae and input and additionally can be modified towards a specific purpose in mind. The models integrity is dependent on the relationships published in literature, meaning the formulae and assumptions captured from literature is what makes the model as realistic as possible.

8.3 Contribution of this Research

The novel approach to observe behaviour and understand consequences in a mine for testing scenarios resulting from technological changes was captured in a model created using a system dynamics tool. It was envisioned that automation may reduce the variability to some degree by managing predictable and measurable processes. This thesis demonstrates that a total mining process chain can be modelled from the relatively simple to the complex causal relationships in a dynamic way to study the causalities and interactions of the mine to capture the consequences of changes such as automation. It is a flexible way of studying mining dynamics. This research will pave the way for mining engineers to benefit from modelling the system dynamics to study concepts and new methods in “data poor” to very data intensive situations, where there are many techniques to represent both ends in the system dynamics modelling environment.

There is an increasing trend in mining engineering education to include system dynamics in their curriculum. This research will benefit such attempts in the future for mining engineering students. The nature of the model developed for this thesis demonstrates mining engineering concepts with good visuals almost in a game-like style to study the causalities and consequences of changes made in the mine design parameters even if it is minute changes.

8.4 Limitations of the Study

The biggest challenge of this study was to involve all the departments of the mining value chain. Typically, research could have been limited to one process, that is drilling, but then this would not meet the objective of changes in behaviour in the other units. The concept of variability and how this affects the rest of the process that are sequential was essential to this research. This requirement then leads to many constants and variables to be carefully selected and used in the model. The task of identifying a multitude of parameters was a long and tedious process. In a team environment this could have been completed much quicker.

The monetary results presented in the discussions chapter (Chapter 7) are based on direct savings due to less consumption of consumables, less drilling, less consumption of fuel and explosives savings with some marginal crushing power savings. The outcome could be much more pronounced if mine cost data were accessible to assess in terms of all the consumables. Essentially the assumptions made in terms of capturing behavioural changes within the mining system due to drilling errors comes from the literature based on observations of various authors. The model construction has some limitations based on the input data used, for example, there is the assumption that incorrect drill rig spotting is 100 % corrected due to automated GPS drill navigation. In reality even GNSS installed drilling may have errors and establishing this error requires independent verification by a field surveyor in terms of how correctly the drill rig navigated to the defined position. The costs of consumables and other complications are not considered. The damage of manual drilling however could be much more pronounced. It can be concluded that automation may improve general mining conditions which is not easy to quantify in the modelling environment.

All processes of mining are modelled with appropriate detail except for primary crushing from the mineral processing department. The mineral processing is out of bounds for this thesis. Mineral processing is dependent on the Run of Mine (ROM), in terms of size distribution. It would have been ideal to have power draw curves per fragmentation to calculate energy draw linked to fragmentation. Fragmentation is the main parameter that connects all processes to drilling quality. Numerous studies attempted to represent the output of blasting linked to drilling parameters, but it unfortunately stays at empirical level and out of the scope of this study.

At the beginning of the study, no prior knowledge of system dynamics existed for the author. It required extensive research on the concept including modelling. It has come to the author's attention that even though a systematic approach makes it look objective, the variables are defined by people and there is some subjectivity in the measured.

The bulk of the mining variables that were captured was the most lengthy task carried out in this study. It proved to be difficult when it comes to finding specific information related to one type of commodity or the machinery characteristics used in the study. It should be cautioned that machinery properly matched in an operation such as size of drill rigs to the size of loader and haulers changes the outlook of the selected input data for this research. The input selected are not necessarily matched ideally on the machinery level which can easily deviate the results from one extreme to the other. Despite these difficulties faced in this research, there is a surprising level of accuracy in the outcome of the simulation in manual and auto mode runs regarding available time, cycle times, number of drill rig requirements, etc as was discussed in Chapter 7.

8.5 Conclusion

This thesis demonstrated that dynamic simulation of mining processes is an effective method for quantification and for analysing the behaviour based on parametric causalities of the systems being studied.

The research does not give only one answer to the questions originally posed but many answers in terms of behavioural changes per process. These changes can be observed in the performance measurements of the machinery, cost of consumables and revenue linked to fragmentation quality. But it paved the way to include a more detailed and accurate answer, provided mine specific data is available to do so. The model requires in depth knowledge of the mine which is somewhat difficult for a researcher outside from the mining company being studied. The results accuracy is as accurate as the input data used.

8.6 Future Work

The weakest area of the research is that the cyclic effect of processes was not captured well in the spatial environment, such as deviation of mining benches from the planned elevations due to incorrect depth control. The main reason behind this error is the spatial nature of the mining blocks where mining blocks being mined is above another mining block that will be the next cycle. Meaning, the blasthole's subdrill section defines next blocks elevation as well as stemming area. Stemming area's intactness is often severed with the previous blocks incorrect subdrill levels. The quality of drilling if deviates from the intended position and depth results in undulations that will be increasingly pronounced for the second and third cycles as the mine progressively deepens. This is not easy to quantify in a system dynamics environment due to the spatial nature of the problem. The author however is aware of spatial modelling attempts of other system dynamists. This is an emerging new challenge in system dynamics and the future of mining industry modelling will likely benefit from this kind of research.

Although, the objective of developing a system dynamics model for measuring the impact of technological changes has been achieved, the study can be improved further with more detail on the formulae and the relationships provided with the field measurements supporting these relationships. A refined model with the support of the field work specific to a type of mine and commodity may lead to improving any gaps that may exist in the created model.

This study identified potential field work to determine crusher throughput based on fragmentation classes. This has been discussed in sections 6.3 and 7.12.

Mining engineers are using many complex and expensive mine planning and design tools during the initial stages of the mine design processes. Thereafter, some of these software packages are used for daily running of the mine in terms of short term planning, job allocations and dispatching. The proposed modelling method to study mining problems is not to be seen as a complete solution for daily running of a mine. The level of abstraction can be high in some areas and detailed in another in this type of modelling environment.

A limited number of mining engineers bought into the system dynamics methods as a tool to study various mining systems and behaviours. The research here will open new doors to mining engineers. The author is confident that anybody who has interest and has a general understanding of the system dynamics principles can improve or add more functionality to the system dynamics model applied to mining engineering model created in this study.

8.7 Final Word

The mining industry will thrive better when it is smart and connected with machines and systems but how smart and effective, they are can only be effectively measured with smart and effective tools.

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Appendix A – VENSIM Model Code

Frag class based shovel fill factor([(0,0)-(10,100)],(1,88),(2,91),(3,84),(4,74),(5,64))
~ dmnl
~ percentage values of shovel fill factor
|
fragmentation class= F80 based fragmentation class determination
~ [1,5,1]
f80 based fragmentation class determination=table of class definition based on P80 input(x80)
~
Crusher throughput= frag class based crusher throughput table(fragmentation class)
shovel factor percentage= Frag class based shovel fill factor(fragmentation class)
~ dmnl
~ in percentage
frag class based crusher power draw([(0,0)-(10,200)],(1,70),(2,92),(3,118),(4,176),(5,190))
~ KWh/kt
~ beyoglu table 35
|
energy draw=frag class based crusher power draw(fragmentation class)
planned versus actual cycle time switch= 1
~ [0,1,1]
~ Use the setting of 1 for actual
|
Loading Time minutes= IF THEN ELSE(planned versus actual cycle time switch>0, "loading cycle time {min)", "planned loading cycle time (min)")
~ min
frag class based crusher throughput table([(0,0)-(10,80)],(1,80),(2,79),(3,74),(4,58),(5,57))
~ t/min
~ beyoglu table 35
|
table of class definition based on P80 input([(0,0)-(1174,10)], (464,1), (615,2), (785,3), (994,4), (1174,5))
~ dmnl
~ the table here is based on the P80 based ore types classes as defined in
|
Block Volume=Burden achieved*Burden achieved*Spacing achieved*"required number of holes per month (Ore)"
~
muck=size fraction feed 1 to 2+size fraction feed 10 to 11+size fraction feed 11 to 12+size fraction feed 12 to 13\ +size fraction feed 13 to 14 +size fraction feed 14 to 15+size fraction feed 15 to 16+size fraction feed 16 to 17\ +size fraction feed 17 to 18+size fraction feed 18 to 19 +size fraction feed 19 to 20+size fraction feed 2 to 3+size fraction feed 3 to

4+size fraction feed 4 to 5 +size fraction feed 5 to 6 +size fraction feed 6 to 7+size fraction feed 7 to 8+size fraction feed 8 to 9+size fraction feed above 20
~
Stemming Length by design= stemming factor*Burden by Design
~ m
costs rate= Cash cost of mining per ton of commodity*"iron ore production planning (t/y)"/TIME STEP
~ Rands
cumulative costs for mining= INTEG (costs rate, 0)
~
uniformity index= IF THEN ELSE("Charge mass m"=0, 0,((2.2-14*Burden achieved/"effective charge diameter (mm)")*((1+Spacing achieved/Burden achieved)/2)^0.5*
(1-((hole depth achieved /100)*Burden achieved)/Burden achieved)*((actual column length+(air deck length/1.25)-actual subdrill)/Bench Height)*((ABS(actual column length+(air deck length/1.25)-actual subdrill))/(actual column length +air deck length/1.25)+0.1)^0.1)/(((Rock sonic velocity *"Rock Density g/cc")/((VOD^1.1)
*Average in hole density))^(LBI/100)))
~
fragments collected= INTEG (muck, 0)
~
Recovery rate= 0.95
~
New Reserves= "new reserves found (t/y)"/"number of months/y"
~ t/Week
"Iron Ore Reserves (t)"= INTEG (New Reserves-"Cumulative reserves mined (t)", "existing reserves (t)")
~
"iron ore production planning (t/y)"= "iron ore demand (t/y)"/Recovery rate
~ t/y
planned number of holes per month= ("Iron ore production planned (t/month)"/"Density of ore (t/m3)")/planned blasthole volume
~
planned blasthole volume= Bench Height*Burden by Design*Spacing by design
~
Per ton capital cost of drill rig= TOTAL NUMBER OF RIGS REQUIRED*((machine price/Total useful Life of Machine*Drilling Primary Production time per year)/annual tons)
~
stemming factor= 0.7
~ [0.7,1.2,0.1]
~ 0.7 for Crushed stone, 1 to 1.2 in all other types of stemming
|
operator proficiency= IF THEN ELSE(automation switch>0,Operator proficiency auto, operator proficiency manual)
~ dmnl [0.1,1,0.01]

~ how effective is the operator to use the mechanical availability of the \ drill rig
|
explosives cost per ton of ore= explosive cost per blasthole/"Breakage weight per hole (tons) - ore"
~ Rand/t
total cumulated fragments= total for defined lumps +total for oversize +total for the crusher 1+total for the crusher 2 +total fines
~
total for the crusher 2= INTEG (size fraction feed 13 to 14+size fraction feed 14 to 15+size fraction feed 15 to 16+\ size fraction feed 16 to 17+size fraction feed 17 to 18, 0)
~
total for defined lumps= INTEG (size fraction feed 4 to 5+size fraction feed 5 to 6+size fraction feed 6 to 7+size fraction feed 7 to 8 +size fraction feed 8 to 9, 0)
~ m3
total for oversize= INTEG (size fraction feed 18 to 19+size fraction feed 19 to 20+size fraction feed above 20, 0)
~
total for the crusher 1= INTEG (size fraction feed 10 to 11+size fraction feed 11 to 12+size fraction feed 12 to 13+ size fraction feed 9 to 10, 0)
~ m3
total fines= INTEG (size fraction feed 1 to 2+size fraction feed 2 to 3+size fraction feed 3 to 4, 0)
~ m3
Stock of Drilled Holes for waste= INTEG (drilled holes arriving waste, 0)
~ blastholes
"Waste drill tempo (min/hole)"="blast hole length(m)"*instantaneous ROP waste +IF THEN ELSE(automation switch>0, Operator activity sum per blast hole Auto , operator activity sum manual)
~ min/hole
"% Actual Quality AQ"= 0.7
~ dmnl
~ percentage
|
"Loading % Equipment availability (EQA)"= Loading Uptime/Controllable time
~ dmnl
~ A performance indicator of how much time equipment can be productive.
Percentage Uptime over Controllable time
|
"Loading % Equipment Utilisation (EQU)"= Loading Operating time/Loading Uptime
~ dmnl
"% loading Rate Achieved RT"= "Actual Production Rate APR m/month"/"Best Demonstrated Rate (BDR)(m/month)"
~ dmnl
"Loading % Overall Utilisation (OU)"=Loading Operating time/Total Calendar Hours T000

~ dmnl
"% Quality Achieved QT"= "% Actual Quality AQ"/"loading Target Quality (TQ)"
~ dmnl
"% Time (TL)"=Controllable time/Total Calendar Hours T000
~ dmnl
~ Controllable Time Drilling The percentage of total calendar time that is actually used by the operation
|
"(1/y)"=1
~
|
"100 ton load time based on number of passes lookup"([(4,2.13)-(18,8.43)],(4,2.13),(7,3.48),(12,5.73),(15,7.08),(18,8.43))
~ min
"100 ton loader switch"= 0
~ dmnl [0,1,1]
"100 ton number of passes required per Truck Payload lookup"([(0,0)-(230,20)],(49,4),(91,7),(146,12),(187,15),(230,18))~
"100 ton number of passes"= "100 ton number of passes required per Truck Payload lookup"(Truck payload)
~
"100 ton load time (min)"= IF THEN ELSE("100 ton loader switch"=1, "100 ton load time based on number of passes lookup" ("100 ton number of passes"),0)
~ min
"400 ton load time based on number of passes lookup"([(2,1.25)-(8,4.05)],(2,1.25),(3,1.72),(4,2.18),(5,2.65),(8,4.05))
~ min
"400 ton loader switch"= 0
~ dmnl [0,1,1]
"400 ton number of passes required per Truck Payload lookup"([(0,0)-(353,8)],(91,2),(146,3),(187,4),(230,5),(353,8))
~ min
"400 ton number of passes"= "400 ton number of passes required per Truck Payload lookup"(Truck payload)
~
"400 ton load time (min)"= IF THEN ELSE("400 ton loader switch"=1 , "400 ton load time based on number of passes lookup"\ ("400 ton number of passes"),0)
~ min
"650 ton load time based on number of passes lookup"([(2,1.25)-(6,3.12)],(2,1.28), (3,1.72), (4,2.18), (6,3.12))
~ min
"650 ton loader switch"= 1
~ dmnl [0,1,1]

"650 ton number of passes required per Truck Payload lookup"([(0,0)-353,6)],(91,2),(146,3),(187,3),(230,4),(353,6))
~
"650 ton number of passes"= "650 ton number of passes required per Truck Payload lookup"(Truck payload)
~
"650 ton load time (min)"= IF THEN ELSE("650 ton loader switch"=1, "650 ton load time based on number of passes lookup"\ ("650 ton number of passes"),0)
~ min
"Accident damage (hrs)"= 24
~ hr
~ All downtime as a result of damage or accidents, excluding equipment that forms part of Production downtime as specified in this document
|
Accumulated 1 to 2= INTEG (size fraction feed 1 to 2, 0)
Accumulated 10 to 11= INTEG (size fraction feed 10 to 11, 0)
Accumulated 11 to 12= INTEG (size fraction feed 11 to 12, 0)
Accumulated 12 to 13= INTEG (size fraction feed 12 to 13, 0)
Accumulated 13 to 14= INTEG (size fraction feed 13 to 14, 0)
Accumulated 14 to 15= INTEG (size fraction feed 14 to 15, 0)
Accumulated 15 to 16= INTEG (size fraction feed 15 to 16, 0)
Accumulated 16 to 17= INTEG (size fraction feed 16 to 17, 0)
Accumulated 17 to 18= INTEG (size fraction feed 17 to 18, 0)
Accumulated 18 to 19= INTEG (size fraction feed 18 to 19, 0)
Accumulated 19 to 20= INTEG (size fraction feed 19 to 20, 0)
Accumulated 2 to 3= INTEG (size fraction feed 2 to 3, 0)
Accumulated 3 to 4= INTEG (size fraction feed 3 to 4, 0)
Accumulated 4 to 5= INTEG (size fraction feed 4 to 5, 0)
Accumulated 5 to 6= INTEG (size fraction feed 5 to 6, 0)
Accumulated 6 to 7= INTEG (size fraction feed 6 to 7, 0)
Accumulated 7 to 8= INTEG (size fraction feed 7 to 8, 0)
Accumulated 8 to 9= INTEG (size fraction feed 8 to 9, 0)
Accumulated 9 to 10= INTEG (size fraction feed 9 to 10, 0)
Accumulated above 20= INTEG (size fraction feed above 20, 0)
"Actual Production Rate APR m/month"= 300
~ m/Month
~ This is the amount of production achieved either in tons or meters drilled depending on the mining machine type. Also the production per time needs to be defined. That is per month per week etc production
|Actual Time required to drill a block of ore= (ADJUSTED NO OF HOLES PER BLOCK OF ORE*"Ore Drill Tempo (min/hole)"/60)/Direct operational hours per day per drill rig
~ days/block [0,7,0.25]

~ Time required to drill one block of ore in months as units. This is discrete, therefore how many hours in a day per drill needs to be used in the calculation of duration of delay in completion of a block

|

Actual time required to drill a block of waste= (ADJUSTED NO OF HOLES PER BLOCK OF WASTE*"Waste drill tempo (min/hole)"/60)/Direct operational hours per day per drill rig

~ hr

additional drill rig= 1

~ [0,20,1]

~ additional resource

|additional drill rig waste=1

~ [0,20,1]

~ additional resource

|

ADJUSTED NO OF HOLES PER BLOCK OF ORE= INTEGER("required number of holes per month (Ore)"/no of ore blocks required per month)

~ dmnl

~ adjustment is required so that an integer number of blocks is obtained based on the needs of the production planning. The integer number obtained is then used to get to actual number of holes required per block for production demand.

|

ADJUSTED NO OF HOLES PER BLOCK OF WASTE= INTEGER("required number of holes per month(waste)"/no of waste blocks required per month)

~

"amortization (Rand/y)"=0

~ Rand/y

amount of anfo per hole= meters charged with ANFO*ANFO linear charge density

~ kg

amount of emulsion per hole= (actual column length-meters charged with ANFO)*"Emulsion linear charge density (kg/m)"

~ kg

ANFO linear charge density= "effective charge diameter (mm)"*"effective charge diameter (mm)"*"Density of ANFO (kg/cu.m)" /1273

~

Annual Shovel operating hours= 4000

~ hr

annual tons= "iron ore demand (t/y)"

~ t/year

Annual truck Loading Time available hr= Mechanical Truck Availability*Planned Truck Availability*Planned Truck Utilisation*Total Truck Time Available Annually

~ hr

annual truck loading time min=Annual truck Loading Time available hr*hrs to min

~ min

automation implementation period= 5

~ year
"Automation implementation time (y)"= IF THEN ELSE(automation switch=0, 0 , automation implementation period)
Automation Project cost= 2e+06
automation switch= 0
~ dmnl [0,1,1]
available number of blasts per month ore=Time available for blasting in a month/blasting cycle time Ore
~ Month
available number of blasts per month waste=Time available for blasting in a month/blasting cycle time waste
~ Month
ave add rod time=IF THEN ELSE("multipass rod \"1\" for yes">0 , no of rods*2.55,0)
~ min
Average Blocks in Service=SMOOTH(Blocks Being Drilled, "smoothing time (in months)")
~ blocks/Month
Average Blocks in Service waste= SMOOTH(Blocks Being Drilled waste, "smoothing time (in months) waste")
~ blocks/Month
Average Blocks Waiting= SMOOTH(Planned Blocks Queued, "smoothing time (in months)")
~ blocks/Month
Average Blocks Waiting waste=SMOOTH(Planned Blocks Queued waste, "smoothing time (in months) waste")
~ blocks
Average in hole density= 1.2
~ g/cm3
~ depth dependent explosive density in the blast hole
|
Bc= "bucket capacity (m3)"
~ m3
~ nominal bucket capacity in meter cube
|
Bench Height= 12.5
~ m
"Best Demonstrated Rate (BDR)(m/month)"=400
~ t/d
~ Best Demonstrated Rate is defined as the best demonstrated performance defined by calculating the average of the 5 best monthly production rates.
|
Bit cost= 2900
~ $
Bit cost per meter ore=Bit cost/Estimated bit life ore
~ $/m
Bit cost per meter waste= Bit cost/Estimated bit life waste

~ $/m
"Blast hole inclination from vertical (degree)"= 0
~ [0,30]
~ input Blasthole inclination from the vertical
|
"blast hole length(m)"= Bench Height/COS("Blast hole inclination from vertical (degree)")+(1-"Blast hole inclination from vertical (degree)"
/100)*actual subdrill
~ m/hole
Blasted ore=DELAY3(Progressing, no of ore blocks required per month)
~ blocks/Month
Blasted waste= DELAY3(Progressing waste, available number of blasts per month waste)
~ blocks/Month
blasthole charging tasks=Connecting the blastholes +Measuring depth of holes +Priming +Stemming
~
Blasting Blocks Queued Ore= INTEG (MAX(Progressing-Blasted ore,0), 0)
~ blocks
Blasting Blocks Queued waste= INTEG (MAX(Progressing waste -Blasted waste,0), 0)
~ blocks
Blasting clearance=20
~ min
~ preblast safety precautions siren, warning for evacuation
blasting cycle time Ore=(Blasting clearance +"Blasting Tasks ore (min)"/no of resources)/60
~ hr
blasting cycle time waste=(Blasting clearance +"blasting tasks waste (min)"/no of resources)/60
~ hr
"Blasting Tasks ore (min)"= (drill block prep with dozer +blasthole charging tasks)*planned number of holes per block ore +Preparing the site
~ min
"blasting tasks waste (min)"= (drill block prep with dozer +blasthole charging tasks)*planned number of holes per block waste +Preparing the site
~ min
blockage=10
~
~ All downtime as a result of blocked chutes, crushers, feeders, screens or stuck equipment.
|"blockage (hrs)"=10
~ hr
~ All downtime as a result of blocked chutes, crushers, feeders, screens or stuck equipment.
|
Blocks Being Drilled=Drilling Blocks Queued +Blasting Blocks Queued Ore

~ blocks/Month
Blocks Being Drilled waste= Drilling Blocks Queued waste + Blasting Blocks Queued waste
~ blocks/Month
"Breakage weight per hole (tons) - ore"=Variable Volume of rock per blast hole due to errors*"Density of ore (t/m3)"
~ ton/hole
~ Breakage weight per hole in tons
|
"Breakage weight per hole(tons)- waste"= "density of overburden (t/m3)"*Variable Volume of rock per blast hole due to errors
~ t
"Bucket capacity (m3)"=bucket capacity lookup(operating weight)
~ m3
bucket capacity lookup([(100,6.5)-(650,34)],(100,6.5),(400,20),(650,34))
~ dmnl
~ This lookup is based on the bucket capacity of the loader and as the loader size changes the bucket volume also changes. This graph is nothing to do with bucket fill factor but is based on the manufactured capacity
|
Bucket Filling Factor= IF THEN ELSE(automation switch>0,1,Effect of Fragmentation on Bucket Filling Factor)
~ dmnl [0,1,0.1]
~ fraction loaded every time a bucket is dumped into a truck
|
bucket passes= RANDOM NORMAL(min bucket pass, max bucket pass, mean bucket pass,std bucket pass, seed)
~ sec
bucket tons= "bucket capacity (m3)"*Bucket Filling Factor*"Density of ore (t/m3)"
~ t
Burden by Design= 5
~ m
Burden achieved= Burden by Design + IF THEN ELSE(automation switch>0,0, RANDOM NORMAL(-1,1 ,0 ,0.5,0.5 \ ))
~
"Capital cost (Rand/y)"="investment on new equipment (Rand/y)"+"operating time (y)"*"fix capital cost (Rand/y)"
~ Rand/y
capital cost of automation= IF THEN ELSE(automation switch=0,0,Automation Project cost +capital expenditure on system\
+capital expenditure per drill* number of drills automated)
~
Capital Cost Truck= 1.7e+06
~ dollar
capital expenditure=2353

~
capital expenditure on system= 2e+06
~ Rand
capital expenditure per drill=1.5e+07
~ Rand
capital shutdown=0
~
~ Scheduled Shutdown or Rebuild to which all the following criteria applies:
- budgeted for in the operational strategy, and
- included in the annual production plan, and

|
"Capital shutdown (days)"= 0
~ hrs
~ Scheduled Shutdown or Rebuild to which all the following criteria applies:
- budgeted for in the operational strategy, and
- included in the annual production plan, and

capital shutdown hrs= daytohrs*"capital shutdown (days)"
~ hr
"Caps per hole (cap/hole)"=1
~ caps/hole [1,6]
Cash cost of mining per ton of commodity= ((cost of power per ton crushed +operating cost of hauling per ton +operating cost of loading per ton\ +per ton drill and blast Cost of Ore)*"iron ore demand (t/y)"+(Per ton drill and blast Cost of Waste +operating cost of hauling per ton\ +operating cost of loading per ton)*"overburden rate (t/y)")/("iron ore demand (t/y)"+"overburden rate (t/y)")
~ Rand/ton
~ Cost of mining overburden is spread to a ton of mined ore for determining cash cost of producing one of commodity
|
"Cash flow after-tax (Rand/y)"= IF THEN ELSE("net profit (Rand/y)">0, "net profit (Rand/y)"+"non-cash deductions (Rand/y)", 0)
~ Rand/year
"Cash flow Rate (Rand/y)"= IF THEN ELSE("net profit (Rand/y)">0, "net profit (Rand/y)"-"non-cash deductions (Rand/y)" , 0)
~ Rand/y
cash generated from operations=17218
~ mRand
"Charge mass / m"= (Average in hole density*"effective charge diameter (mm)"^2*100)/127000
~ kg/m
charging the holes and blasting=1
~ Day
Check for unity of fractions= Fraction 1 Passing + size fraction 18 to 19+size fraction 19 to 20+size fraction 1 to 2\ +size fraction 3 to 4+size fraction 9 to 10+size fraction 4 to 5+size

fraction 10 to 11+size fraction 5 to 6+size fraction 11 to 12\ +size fraction 12 to 13+size fraction 13 to 14+size fraction 6 to 7+size fraction 14 to 15+size fraction 2 to 3+size fraction 15 to 16\ +size fraction 7 to 8+size fraction 16 to 17+size fraction 17 to 18+size fraction 8 to 9+size fraction more than sieve 20

~

"Collaring time (min)"=2

~ min [?,?,0.05]

actual column length= (Bench Height + actual subdrill-stemming length achieved-air deck length)

~ m

Connecting the blastholes= 0.7

~ min/hole

"Consumable Cost per ton (ore)"= "G.E.T. cost of ore (rand/m)"/"yield of broken ore (t/m)"

~ Rand/t

consumable cost per ton waste="G.E.T. cost of waste (rand/m)"/Yield of broken waste

~ Rand/t

control and instrumentation= 10

~

~ Time taken for control or instrumentation repairs to restore functionality.

|

"Control and instrumentation (days)"= 10

~ days

~ Time taken for control or instrumentation repairs to restore functionality.

|

Controllable time= Total Calendar Hours T000-Loading Non Controllable Time

~ hrs

~ Available equipment time attributable to any internal factors under the control of the operation that impacts production.

|

Cost of 1 Bit= 40000

~ R/bit

Cost of anfo per ton= 9000

~ Rand/t

Cost of Blasting Per month= "Cost of Caps per month (rand/month)"+"Total Cost of Explosives (rand/month)"

~ Rand

~ Total cost of blasting per day

|

Cost of booster= 50

~

cost of bulk explosive per blast hole=(amount of anfo per hole*cost of anfo per ton/1000+amount of emulsion per hole*cost of emulsion per ton /1000)

~ Rand/hole

"Cost of Caps per month (rand/month)"= "Number of caps per day (caps/month)"*"Unit cost of caps (rand/cap)"+Initiation system cost per month
~
~ Cost of caps per day
|
cost of electricity lookup specific year= price of electricity lookup(specific year)
~
cost of emulsion per ton= 10000
~ Rand/t
cost of initiating and accessories per blasthole=(actual column length +Spacing achieved)* slag factor of shock tube*shock tube cost per metre + "Caps per hole (cap/hole)"*"Unit cost of caps (rand/cap)"
+cost of booster*number of boosters per hole
~ Rand/hole
cost of operating truck in literature= "dollar & rand exchange"*("Operating cost of truck/hr"*Truck Cycle Time/60)/Truck payload
~ Rand/t
cost of power per ton crushed= Power consumption per ton*cost of electricity lookup specific year
"Coupling %"= 100
~
~ coupling percentage value, 100 for pumpable, less than 100 for cartridge explosives
|
crusher cost per ton= 0
~
crusher power consumption per month= Power consumption per ton*"Iron ore production planned (t/month)"
~ KWh/Month
crusher power cost per month= cost of electricity lookup specific year*crusher power consumption per month
~ Rand/Month
"cumulative overburden (tons)"= INTEG ("overburden rate (t/month)", 0)
~ t
cumulative power used= INTEG (crusher power consumption per month, 0)
~
"Cumulative reserves mined (t)"= INTEG ("Iron ore production planned (t/month)", 0)
~ t
Cumulative spent on crushing= INTEG (crusher power cost per month, 0)
~ Rand
daytohrs= 24
~ hr/Day
deferred stripping capitalised= 321
~ mRand
delays= 5

~
~ All external and internal delays experienced by the Maintenance department during Scheduled Maintenance that causes an interruption in service
delivery i.e. awaiting spares, awaiting transport, awaiting labour, awaiting supplier.
|
"Density of ANFO (kg/cu.m)"=0.9
~ g/cc
"Density of emulsion (g/cc)"= 1.1
~ g/cc [0.8,1.5]
~ Density of high explosive
|
"Density of ore (t/m3)"=
3.5
~ t/m3
"density of overburden (t/m3)"=2.35
~ t/m3
"depletion (Rand/y)"=0
~ Rand/y
Depreciation= 51
~ US$/hr
"depreciation (Rand/y)"=0
~ Rand/y
Desired no of blocks= scheduled ore blocks per month +Planned Blocks Queued
~ blocks/Month
~ This variable is sensitive to time step, AS WELL AS CHANGES TO THE NUMBER
OF HOLES PER BLOCK
|
Desired no of blocks waste= scheduled ore blocks per month waste + Planned Blocks Queued waste
~ blocks/Month
~ This variable is sensitive to time step, AS WELL AS CHANGES TO THE NUMBER OF HOLES PER BLOCK
diesel electricity air water= 10
~
~ no diesel or water
"diesel electricity air water (hrs)"= 10
~ hr
~ no diesel or water
"direct drilling hours required per month (total)"="Direct Drilling time required per month ore (hr)"+"Direct Drilling time required per month waste (hr)"
~ hr/Month
Direct Drilling time available per drill per month= Drilling Primary Production time per year/12
~ hr/Month

direct drilling time available per year= Drilling Primary Production time per year*TOTAL NUMBER OF RIGS REQUIRED
~
"Direct Drilling time required per month ore (hr)"= "required number of holes per month (Ore)"*"Ore Drill Tempo (min/hole)"/60
~ hr/Month
"Direct Drilling time required per month waste (hr)"= "required number of holes per month(waste)"*"Waste drill tempo (min/hole)"/60
~ hr/Month
Direct loading time available per loader per month= Loading Primary Production time/12
~ hr
"Direct loading time required per month ore (hr)"=0
~ hr
"Direct loading time required per month waste (hr)"= 0
~ hr
"direct loading hours required per month (total)"= 0
~ hr
Direct operational days per year=365-public holidays- sundays
~
Direct operational hours per day per drill rig= Drilling Primary Production time per year/Direct operational days per year
~ hr/Day
~ excluding public holiday
|
Direct operational hours per day per machine= Loading Primary Production time/loading Direct operational days per year
~ hrs/Day
~ excluding public holiday
|
"Discount payback period (y)"= IF THEN ELSE("Net Present Value (Rand)">0, 1, 0)
~ dmnl
"Discount rate (%)"= 10.26
~ dmnl
"dollar & rand exchange"= 15
~ Rand/$
drill block prep with dozer= Burden achieved*Spacing achieved*60/2000
~ min
~ 2000 meters square per hour
|
drill assistant cost to company= 12000
~ Rand
drill cycle fuel=drilling 80*"Fuel consumption per drill drilling (lt/hr)"
~ lt/hr
Drill pipe cost=9000

~ $
Drill Rig operator cost per month= IF THEN ELSE(automation switch>0,0, no of shifts per day*TOTAL NUMBER OF RIGS REQUIRED\
*(operator cost to company + no of helpers per drill rig *drill assistant cost to company))
~ Rand/Month
drilled holes arriving ore= "required number of holes per month (Ore)"
~ hole
drilled holes arriving waste= "required number of holes per month(waste)"
~ hole
drilling 80= 0.8
~ dmnl
~ percentage of the mechanical availability of the drill rig
|
drilling blocks processing time ore= "Ore Drill Tempo (min/hole)"/(1+stripping ratio)
~
Drilling Blocks Queued= INTEG (MAX(Starting-Progressing,0),0)
~ blocks
~ max function is used to avoid having negative numbers
|
Drilling Blocks Queued waste= INTEG (MAX(Starting waste-Progressing waste,0),0)
~ blocks
drilling capital shutdown hrs= daytohrs*"Drilling capital shutdown (days)"
~
"Drilling diesel electricity air water (hrs)"= 10
~ hr
~ no diesel or water
|
Drilling Fuel cost per ton of waste= fuel cost per meter drilled waste/Yield of broken waste
~ Rand/m
Drilling Fuel Cost per Ton ore= fuel cost per meter drilled ore/"yield of broken ore (t/m)"
~ Rand/t
"Drilling maintenance related delays (days)"=5
~ days
~ All external and internal delays experienced by the Maintenance department during Scheduled Maintenance that causes an interruption in service delivery i.e. awaiting spares, awaiting transport, awaiting labour, awaiting supplier.
|
"Drilling non production shift (hr)"= 0
~ hr
~ Shift defined as a non working shift by the operation – where operations \
do not operate a 7-day week or a 24-hour day.
|
Drilling Primary Production time per year= Drilling Direct Operating time-Drilling Secondary Non Production Work

~ hr/year
"Drilling section moves (hrs)"= 0
~ hr
~ Relocation of equipment due to operational requirements. Performed in non \
production time for those operations that do not operate 24/7/365. For \
example, mining section or major equipment moves where the impact on \
production is accounted for in the budgeted annual plan. Note: The time \
the move exceeds the plan must be recorder as a Production Delay.
|
"Drilling unscheduled production stop (hrs)"= 0
~ hr
~ All downtime experienced due to Unscheduled production activities \
required, excludes Tyres / Ropes / Chains.
|
"Drilling % Actual Quality AQ"=0.7
~ dmnl
~ percentage
|
"Drilling % Equipment availability (EQA)"=Drilling Uptime/Drilling Controllable time
~ dmnl
~ A performance indicator of how much time equipment can be productive.
Percentage Uptime over Controllable time
|
"Drilling % Equipment Utilisation (EQU)"=Drilling Direct Operating time/Drilling Uptime
~ dmnl
"Drilling% Overall Utilisation (OU)"=Drilling Direct Operating time/Drilling Total Calendar Hours
~ dmnl
"Drilling % Quality Achieved QT"="Drilling % Actual Quality AQ"/"Drilling Target Quality (TQ)"
~ dmnl
"Drilling % Rate Achieved RT"= "Drilling Actual Production Rate APR m/month"/"Drilling Best Demonstrated Rate (BDR)(m/month)"
~ dmnl
"Drilling% Time"= Drilling Controllable time/Drilling Total Calendar Hours
~ dmnl
~ Controllable Time Drilling The percentage of total calendar time that is actually used by the operation
|
"Drilling Accident damage (hrs)"=24
~
~ All downtime as a result of damage or accidents, excluding equipment that forms part of Production downtime as specified in this document
|

"Drilling Actual Production Rate APR m/month"=300
~ m/Month
~ This is the amount of production achieved either in tons or meters drilled depending on the mining machine type. Also the production per time needs to be defined. That is per month per week etc production
|
"Drilling Best Demonstrated Rate (BDR)(m/month)"= 400
~ t/d
~ Best Demonstrated Rate is defined as the best demonstrated performance defined by calculating the average of the 5 best monthly production rates.
|
"Drilling blockage (hrs)"= 10
~ hr
~ All downtime as a result of blocked chutes, crushers, feeders, screens or stuck equipment.
|
"Drilling capital shotdown (days)"=0
~ hr
~ Scheduled Shutdown or Rebuild to which all the following criteria applies:

- budgeted for in the operational strategy, and
- included in the annual production plan, and

|
"Drilling control and instrumentation (days)"= 10
~ days
~ Time taken for control or instrumentation repairs to restore functionality.
|
Drilling Controllable time= Drilling Total Calendar Hours-Drilling Non Controllable Time
~ hr
~ Available equipment time attributable to any internal factors under the control of the operation that impacts production.
|
Drilling Direct Operating time= Drilling Uptime-Drilling Lost Time
~ hr
~ Time during which the equipment is operating.
|
"Drilling electrical (days)"=10
~ days
~ Time taken for electrical repairs to restore functionality
|
"Drilling environmental (hrs)"= 0
~ hr
~ Lost production due to manageable environmental conditions, such as dust, stagnant water, rain, mist, ventilation, etc. as opposed to abnormal uncontrollable conditions
|

"Drilling environmental uncontrollable (days)"=0
~ days
~ Impact due to abnormal environmental occurrences which could not have been budgeted for, or exceeds the budgeted time period, such as excessive rain that causes flooding, earthquakes, etc. Only events greater than 24 hours in duration may be booked under this section
|
Drilling Equipment down time=Drilling Operational Stops hrs+ Drilling Scheduled maintenance time hrs+ Drilling Unscheduled maintenance time hrs
~ hr
~ Downtime attributable to Maintenance and Operational that renders the equipment inoperable
|
"Drilling external delays (hrs)"=1
~ hr
~ Production delays experienced due to external production interruptions. (Measured at section or module level and not at equipment level)
|
"Drilling external utilities (days)"= 1
~ days
~ Impact due to outside utilities such as power, water, rail systems, etc.
|
"Drilling follow-on work (hrs)"=5
~ hr
~ A follow-on task required as a result of the measured condition
|
Drilling ground engaging tools=50
~
~ drill bits, drill steel etc stuck, change required, tool broken unexpectedly
|
"Drilling hydraulic (days)"=10
~ days
~ Time taken for hydraulic repairs to restore functionality.
|
"Drilling inspections (hrs)"= 300
~ hr
~ Maintenance inspections initiated through predetermined intervals (including Visual and Condition Monitoring)
|
"Direct Drilling Delays (hrs)"= "Drilling environmental (hrs)"+"Drilling external delays (hrs)"+"Drilling no operator lost time (hrs)"
+"Drilling operational delays (hrs)"+Drilling safety +"Drilling service delays (hrs)"
+"Drilling staff related (hrs)"+Drilling supply chain
~ hr

~ Impact on equipment, section or modules as a result of upstream or downstream stoppages causing lost time.

|

"Drilling Lost Time Standby (hr)"=0

~ hr

~ some other drilling equipment is available for work therefore standby mode

|

"Drilling Consequential lost time (hr)"= 30

~ hr

~ Impact due to other equipment, modules or section/s experiencing Engineering Downtime

|

Drilling Long Term Productivity= Meters possible to be drilled in a year/ Drilling Total Calendar Hours

~ m/year

Drilling Lost Time="Drilling Consequential lost time (hr)"+"Direct Drilling Delays (hrs)"+"Drilling Lost Time Standby (hr)"

~ hr

~ Time during Uptime (T200) when the machine was available for production, but was not utilized

|

"Drilling market conditions (days)"=0

~ days

~ Loss of production time incurred as a result of market conditions.

|

Drilling Mean Time Between Maintenance Events MTBM=(Drilling Controllable time-Drilling Unscheduled maintenance time hrs-Drilling Scheduled maintenance time hrs)/Drilling number of maintenance events NEF

~ dmnl

~ mtbm=uptime/number of maintenance events

|

"Drilling mechanical (days)"=5

~ days

~ Time taken for mechanical repairs to restore functionality.

|

"Drilling modifications (days)"=1

~ days

~ Maintenance tasks initiated through applying a structured RCM-based process to modify the equipment or component to improve reliability, maintainability or availability

|

Drilling Non Scheduled Time= drilling capital shutdown hrs + "Drilling nonproduction shift (hr)"+"Drilling public holiday (hrs)" +"Drilling section moves (hrs)"

~ hr

~ Equipment time allocated as not being required in the production plan. \

(Only applicable to non-full calendar operations)

Drilling Uncontrollable time hrs=
("Drilling environmental uncontrollable (days)"+"Drilling external utilities (days)"\
+"Drilling market conditions (days)"+"Drilling non-scheduled holidays (days)"
+"Drilling uncontrollable labour (days)")*daytohrs
~ hr/year
~ Time attributable to external factors beyond the control of the operation \
that affects the whole operation e.g. outside utilities such as power, \
water, rail systems, environmental disasters
|
"Drilling no operator lost time (hrs)"= IF THEN ELSE(automation switch>0,0,no operator frequency per year*no operator related average lost time\ )
~ hr
Drilling Non Controllable Time= Drilling Non Scheduled Time+ Drilling Uncontrollable time hrs
~ hr
"Drilling non-scheduled holidays (days)"=1
~ days
~ Loss of production time due to additional national holidays, such as voting days or time off in lieu of time worked in.
|
Drilling number of maintenance events NEF=2
~ dmnl
"Drilling operational consumables (hrs)"= 0
~ hr
~ Internal controllable downtime associated with operational consumables not \
being available, i.e. Reagents, Magnetite, Flocculent. Excludes supply \
chain being responsible for it not being available.
|
"Drilling operational delays (hrs)"=NO OF BLASTS PER MONTH*Standby due to blasting*12
~ hr
~ Delays experienced due to necessary production activities, such as \
blasting, housekeeping, full silo, preparation for maintenance
|
Drilling Operational Stops hrs=
"Drilling Accident damage (hrs)"+"Drilling blockage (hrs)"+"Drilling diesel electricity air water (hrs)"\
+Drilling ground engaging tools +"Drilling operational consumables (hrs)"+"Drilling Production stoppage (hrs)"\
+"Drilling tyres ropes chains (hrs)"+"Drilling unallocated (hrs)"+"Drilling unscheduled production stop (hrs)"
~ hrs
~ Operational Stops Downtime: Necessary downtime attributable to Production \

that renders the equipment inoperable e.g. replacement of consumables
|
"Drilling Overall Equipment Effectiveness %"= "Drilling% Overall Utilisation (OU)"*"Drilling % Quality Achieved QT"*"Drilling % Rate Achieved RT"
~ dmnl
~ OEE = OEQU x (Actual Production Rate)/(Best Production Rate) x (Actual Quality)/(Target Quality)
|
"Drilling Production stoppage (hrs)"=5
~ hr
~ All Production reasons that render the equipment inoperative
|
"Drilling public holiday (hrs)"=public holidays*daytohrs
~ hr
~ National holiday defined as a non-working day by the operation
|
Drilling safety=10
~ hr
~ Downtime due to safety inspections and safety related interventions
|
Drilling Scheduled maintenance time hrs=(("Drilling maintenance related delays (days)"+"Drilling service and shutdowns (day)"+"Drilling modifications (days)")*daytohrs+"Drilling follow-on work (hrs)"+"Drilling inspections (hrs)")
~ hr
~ Scheduled Maintenance Downtime as a result of maintenance work included in the confirmed weekly maintenance plan
|
Drilling Secondary Non Production Work=10
~ hr
~ Equipment is operational, but performing non-production activities such as training,
|
"Drilling service and shutdown (day)"= 10
~ days
~ Maintenance tasks associated with predetermined services and shutdowns
|
"Drilling service delays (hrs)"=30
~ hr
~ Production delays experienced due to the installation of, or lack of, services, such as survey, ventilation, construction, etc.
|
"Drilling staff related (hrs)"= 365*"hot seat change time (min/day)"/60
~ hr
~ Delays experienced due to travelling, transport, shift change, etc. (Measured at section or module level and not at equipment level)

|
Drilling supply chain= 10
~ hr
~ Inability to perform work due to supply chain related delays, such as stock outs, incorrect materials purchased
|
"Drilling Target Quality (TQ)"=0.99
~ dmnl
~ Target Quality percent
|
Drilling Total Calendar Hours=365*24
~ hr
~ Total Time Annualized=24 x 365 = 8760
|
"Drilling tyres ropes chains (hrs)"= 0
~
~ All downtime related to tyres, ropes or chains or cable
|
"Drilling unallocated (hrs)"=5
~ hr
~ Downtime, which cannot be classified into the above or for when no reason has been recorded, such as balancing Direct Operating Hours with that of the Service Meter Reading.
|
"Drilling uncontrollable labour (days)"=8
~ days
~ Impact as a result of operation wide industrial action.
|
"Drilling unscheduled delays (days)"= 10
~ days
~ All external and internal delays experienced by the Maintenance department during Unscheduled Maintenance that causes an interruption in service delivery i.e. awaiting spares, awaiting transport, awaiting labour, awaiting supplier.
|
Drilling Unscheduled maintenance time hrs= ("Drilling control and instrumentation (days)"+"Drilling electrical (days)"+"Drilling hydraulic (days)"+"Drilling mechanical (days)"+"Drilling unscheduled delays (days)")*24
~ hrs
~ Unscheduled Maintenance Downtime as a result of maintenance work not included in the confirmed weekly maintenance plan
|

Drilling Uptime= Drilling Controllable time- Drilling Equipment down time
~ hr
~ Equipment time available for production activities.

|
Dump time= RANDOM NORMAL(min dump time, max dump time, mean dump time, std dump time, seed)
~
Effect of Fragmentation on Bucket Filling Factor= 0.8
~ [0,1]
Effect of fragmentation on loading cycle= 0.1
~ [0,1]
~ percent effort increased due to difficult digging conditions increasing this variable causes cycle time increases
|
Effect of operational site parameters on loading= 0.1
~ dmnl

"effective charge diameter (mm)"= Hole diameter by design*SQRT("Coupling %"/100)
~ mm
effective tax rate= 26
~
~ %
|
electrical=10
~
~ Time taken for electrical repairs to restore functionality
|
"electrical (days)"= 10
~ days
~ Time taken for electrical repairs to restore functionality
|
"Emulsion linear charge density (kg/m)"= "Density of emulsion (g/cc)"*"effective charge diameter (mm)"*"effective charge diameter (mm)" /1273
~
Energy factor= RWS*Powder factor/100
~ dmnl
"environmental (hrs)"=0
~ hr
~ Lost production due to manageable environmental conditions, such as dust, stagnant water, rain, mist, ventilation, etc. as opposed to abnormal uncontrollable conditions
|
"environmental uncontrollable (days)"= 0
~ days
~ Impact due to abnormal environmental occurrences which could not have been budgeted for, or exceeds the budgeted time / period, such as excessive rain that causes flooding, earthquakes, etc. Only events greater than 24 hours in duration may be booked under this section

|
Equipment down time= Loading Operational Stops hrs+ Loading Scheduled maintenance time hrs+ Unscheduled maintenance time hrs
~ hr
~ Downtime attributable to Maintenance and Operational that renders the equipment inoperable
|
equity holders= 8621
~ mRand
Estimated bit life ore= 1100
~ m
Estimated bit life waste= 1100
~ m
Estimated Drill Pipe life ore= 26000
~ m
Estimated Drill Pipe life waste= 26000
~ m
"existing reserves (t)"= 1.2e+08
~ t
Expected Mean fragment size= IF THEN ELSE(RWS=0, 0 ,"Rock Factor (A)"*((Energy factor^(-0.8))*explosive mass per hole ^(1/6))*((115/RWS)^0.633))
~ cm
explosive cost per blasthole=cost of bulk explosive per blast hole+ cost of initiating and accessories per blasthole
~ Rand/hole
explosive mass per hole= "Charge mass / m"*actual column length
~ kg/hole
explosives cost per ton of waste= cost of initiating and accessories per blasthole/"Breakage weight per hole(tons)- waste"
~ Rand/t
"external delays (hrs)"= 1
~ hr
~ Production delays experienced due to external production interruptions.
(Measured at section or module level and not at equipment level)
|
external utilities= 10
~
~ Impact due to outside utilities such as power, water, rail systems, etc.
|
"external utilities (days)"= 1
~ days
~ Impact due to outside utilities such as power, water, rail systems, etc.
|
F80 feed size= x80

~
~ feed size to the crusher
|
Feasible no of blocks=MAX(0,No of Rigs allocated to ORE drilling-Blocks Being Drilled)/TIME STEP+ additional drill rig
~ blocks/Month
Feasible no of blocks waste= MAX(0,No of Rigs Allocated to Waste Drilling-Blocks Being Drilled waste)/TIME STEP additional drill rig waste
~ blocks/Month
"fix capital cost (Rand/y)"= 0
~ Rand/y
"follow-on work (hrs)"= 5
~ hr
~ A follow-on task required as a result of the measured condition |
Fraction 1 Passing= 1-(EXP((-1)*Sieve1/10/Expected Mean fragment size)^uniformity index)
~
Fraction 10 Passing= 1-(EXP((-1)*Sieve10/10/Expected Mean fragment size)^uniformity index)
~
Fraction 11 Passing= 1-(EXP((-1)*Sieve11/10/Expected Mean fragment size)^uniformity index)
~
Fraction 12 Passing= 1-(EXP((-1)*Sieve12/10/Expected Mean fragment size)^uniformity index)
~
Fraction 13 Passing= 1-(EXP((-1)*Sieve13/10/Expected Mean fragment size)^uniformity index)
~
Fraction 14 Passing= 1-(EXP((-1)*Sieve14/10/Expected Mean fragment size)^uniformity index)
~
Fraction 15 Passing= 1-(EXP((-1)*Sieve 15/10/Expected Mean fragment size)^uniformity index)
~
Fraction 16 Passing= 1-(EXP((-1)*Sieve16/10/Expected Mean fragment size)^uniformity index)
~
Fraction 17 Passing= 1-(EXP((-1)*Sieve17/10/Expected Mean fragment size)^uniformity index)
~
Fraction 18 Passing= 1-(EXP((-1)*Sieve18/10/Expected Mean fragment size)^uniformity index)
~

Fraction 19 Passing= 1-(EXP((-1)*Sieve19/10/Expected Mean fragment size)^uniformity index)
~
Fraction 2 Passing= 1-(EXP((-1)*Sieve2/10/Expected Mean fragment size)^uniformity index)
~
Fraction 20 Passing= 1-(EXP((-1)*Sieve20/10/Expected Mean fragment size)^uniformity index)
~
Fraction 3 Passing= 1-(EXP((-1)*Sieve3/10/Expected Mean fragment size)^uniformity index)
~
Fraction 4 Passing= 1-(EXP((-1)*Sieve4/10/Expected Mean fragment size)^uniformity index)
~
Fraction 5 Passing= 1-(EXP((-1)*Sieve5/10/Expected Mean fragment size)^uniformity index)
~
Fraction 6 Passing= 1-(EXP((-1)*Sieve6/10/Expected Mean fragment size)^uniformity index)
~
Fraction 7 Passing=1-(EXP((-1)*Sieve7/10/Expected Mean fragment size)^uniformity index)
~
Fraction 8 Passing= 1-(EXP((-1)*Sieve8/10/Expected Mean fragment size)^uniformity index)
~
Fraction 9 Passing= 1-(EXP((-1)*Sieve9/10/Expected Mean fragment size)^uniformity index)
~
Fraction of over drill= RANDOM NORMAL(0 , 1,0.5 , mean subdrill fraction, seed)
~ [0,?,0.1]
fraction of stemming length variation= RANDOM NORMAL(0 , 1,0.7, mean stemming fraction ,seed)
~ dmnl
fuel consumption factor per drill rig type per hour=drill cycle fuel+ idle cycle fuel +level cycle fuel+ pipe handle cycle fuel +propel cycle fuel
~ lt/hr
Fuel consumption idling= 26
~ lt/hr
~ KOMATSU PC 4000
|
Fuel consumption loading= 255
~ lt/hr
~ KOMATSU PC4000

"Fuel consumption per drill drilling(lt/hr)"=101
~ lt/hr
"Fuel consumption per drill idle (lt/hr)"= 24.5
~
"Fuel consumption per drill levelling (lt/hr)"= 39 ~
"Fuel consumption per drill pipe handling (lt/hr)"= 50.9

~ lt/hr
"Fuel consumption per drill propel (lt/hr)"= 77.2
~
Fuel Consumption per truck per hour= 160
~ lt/hr
fuel cost per hole ore= Fuel Cost per hr drilled*"Ore Drill Tempo (min/hole)"/60
~ Rand/hole
fuel cost per hole waste= Fuel Cost per hr drilled*"Waste drill tempo (min/hole)"/60
~
Fuel Cost per hr drilled= fuel consumption factor per drill rig type per hour*petrol price with lookup
~ Rand/hr
fuel cost per meter drilled ore=fuel cost per hole ore/"blast hole length(m)"
~ Rand/m
fuel cost per meter drilled waste= fuel cost per hole waste/"blast hole length(m)"
~ Rand/m
fuel cost per ton for trucks=(fuel used per truck cycle/Truck payload)*petrol price with lookup
~ Rand/t
fuel used per truck cycle= Fuel Consumption per truck per hour*Truck Cycle Time/60
~ lt/payload
"G.E.T. cost of ore (rand/m)"=(Bit cost per meter ore+ Pipe cost per meter ore)* "dollar & rand exchange"
~ Rand/m
"G.E.T. cost of waste (rand/m)"=(Bit cost per meter waste+ Pipe cost per meter waste)*"dollar & rand exchange"
~ Rand/m
"Geometric Yield of broken volume per m drilled (m3/m)"=Variable Volume of rock per blast hole due to errors/"blast hole length(m)"
~ m3/m
~ Yield of broken rock in cubic meter per meter drilled
|
"gross profit (Rand/y)"= "iron ore production planning (t/y)"*("iron ore price (Rand/t)"-unit cash cost rand per ton ore\ )
~ Rand/y
ground engaging tools= 50
~
~ drill bits, drill steel etc stuck, change required, tool broken unexpectedly
|
Hauling Distance= 12.2
~ km
Hauling speed empty= 50
~ km/hr
Hauling Speed full= IF THEN ELSE(automation switch>0, 29 , 21.2)

~ km/hr
hole depth achieved= actual column length +stemming length achieved +air deck length
~ m
Hole diameter by design= 229
~ mm
"hot seat change time (min/day)"=IF THEN ELSE(automation switch>0,0,"drilling seat change time per day (min)")*no of shifts per day
~ min [0,200,0.1]
hrs per month loading= Direct loading time available per loader per month
~ hr/Month
hrs to min=60
~ min/hr
hydraulic=10
~
~ Time taken for hydraulic repairs to restore functionality.
|
"hydraulic (days)"= 10
~ days
~ Time taken for hydraulic repairs to restore functionality.
|
idle cycle fuel="Fuel consumption per drill idle (lt/hr)"*idle5
~
idle5= 0.05
~ dmnl
~ percentage of the mechanical availability of the drill rig
|
"income tax (Rand/y)"= ("taxable income (Rand/y)"*("tax rate (%)"/100))
Rand/y
individual bucket load Time=Dump time +Return time +Swing time
~ sec
Individual Dig time= 12.3699+0.0072*x80
~
~ based on fig 5 in Brunton's paper
|
Initiation system cost per month= number of holes per month drilled*shock tube cost per metre*"blast hole length(m)"
~ Rand/Month
instantaneous ROP waste= mean ROP waste
~ min/m
Instantaneous ROP ore=mean ROP
~ min/hole
"investment on new equipment (Rand/y)"= ZIDZ(capital cost of automation, "automation implementation time (y)")
~ Rand/y

"iron ore demand (t/y)"= 1.4e+07
~ t/y
"iron ore price (Rand/t)"= 72*"dollar & rand exchange"
~ Rand/t
~ 2018 price for iron ore
|
"Iron ore production planned (t/month)"= "iron ore demand (t/y)"/12
~ t/Month
"iron ore to be mined per day (t/d)"="iron ore production planning (t/y)"/"working day planning (days/y)"
~
JPO= 10
~ dmnl
~ Joint plane orientation, 10 for horizontal, 20 dipping out of face, 30 \ perpendicular to face, 40 dipping out of face
|
JPS=40
~ dmnl
~ Joint plane spacing, 10 for < 100 mm to 50 for > 1.0 m
|
"Loading Delays (hrs)"= "environmental (hrs)"+"external delays (hrs)"+"no operator lost time (hrs)"+"operational delays (hrs)"\
+loading safety +"loading service delays (hrs)"+"loading staff related (hrs)"+loading supply chain
~ hrs
~ Impact on equipment, section or modules as a result of upstream or \
downstream stoppages causing lost time.
|

"Loading Standby (hr)"=0
~ hr
~ some other drilling equipment is available for work therefore standby mode
|
LBI=0.5*(RMD+(Rock UCS*0.05)+(25*"Rock Density g/cc"-50)+JPS+JPO)
~
~ Lily blastability index
|
level cycle fuel="Fuel consumption per drill levelling (lt/hr)"*leveling3
~
"Level Time (min)"= 0.4
~ min
leveling3=0.03
~ dmnl
~ percentage of the mechanical availability of the drill rig

|
"Levelling -manual"=1.3
~ min
actual loading cycle time= bucket passes*(individual bucket load Time+ Individual Dig time)
~ sec
"Loading cycle time {min)"=actual loading cycle time/mintosecc
~ min
loading Direct operational days per year= 365-loading public holidays-loading sundays
~ days
"Loading hot seat change time (min/day)"= "loading seat change time per day (min)"*no of shifts options
~ min [0,200,0.1]
"Loading inspections (hrs)"= 300
~ hr
~ Maintenance inspections initiated through predetermined intervals (including Visual and Condition Monitoring)
|
"Loading Consequential (hr)"=30
~ hr
~ Impact due to other equipment, modules or section/s experiencing Engineering Downtime
|
loading Lost Time= "loading Consequential (hr)"+"Loading Delays (hrs)"+"Loading Standby (hr)"
~ hr
~ Time during Uptime (T200) when the machine was available for production, but was not utilized
|
loading Mean Time Between Maintenance Events MTBM= (Controllable time-Unscheduled maintenance time hrs-Loading Scheduled maintenance time hrs)/loading number of maintenance events NEF
~ dmnl
~ mtbm=uptime/number of maintenance events
|
loading no of months per year= 12
~ Month/y
loading no operator frequency per year= 1
~ dmnl
loading no operator related average lost time=8
~ hr
"Loading nonproduction shift (hr)"= 0
~ hr
~ Shift defined as a non-working shift by the operation – where operations do not operate a 7-day week or a 24-hour day.

|
loading number of maintenance events NEF=2
~ dmnl
loading public holidays=12
~ days
loading rate= ("bucket capacity (m3)"*Bucket Filling Factor*number of passes required/"planned loading cycle time (min)")/"Density of ore (t/m3)"
~ t/min
loading safety= 10
~ hr
~ Downtime due to safety inspections and safety related interventions
|
"loading seat change time per day (min)"= 20
~ min
"loading service delays (hrs)"= 30
~ hr
~ Production delays experienced due to the installation of, or lack of, services, such as survey, ventilation, construction, etc.
|
"loading staff related (hrs)"= 365*"loading hot seat change time (min/day)"/60
~ hr
~ Delays experienced due to travelling, transport, shift change, etc. (Measured at section or module level and not at equipment level)
|
loading Standby due to blasting= 2
~ hr
loading sundays=52
~ Day
loading supply chain=10
~ hr
~ Inability to perform work due to supply chain related delays, such as stock outs, incorrect materials purchased
|
"loading Target Quality (TQ)"= 0.99
~ dmnl
~ Target Quality percent
|
Long Term Productivity= tons possible to to be loaded in a year/Total Calendar Hours T000
~ton/year
LOOKUP Probability Density of X([(0,0)-(1,1)],(0,0),(0.0703364,0.0307018),(0.174312,0.114035),(0.262997,0.223684),(0.360856,0.337719),(0.422018,0.368421),(0.480122,0.346491),(0.571865,0.22807),(0.623853,0.114035),(0.70948,0.0570175),(0.819572,0.0263158),(1,0))
~ dmnl

~ Probability density for a random variable distributed over a 0-1 range. The scaling is arbitrary, so that this lookup may not be a true density function, in that its area does not integrate to 1.

|

lubricants cost per ton= "dollar & rand exchange"*Lubricants cost per year/Tons to be moved per year

~ Rand/t

Lubricants cost per year= 1100

~ US$/year

machine price= 5e+07

~ Rand

"Maintenance related delays (days)"= 5

~ days

~ All external and internal delays experienced by the Maintenance department during Scheduled Maintenance that causes an interruption in service delivery i.e., awaiting spares, awaiting transport, awaiting labour, awaiting supplier.

|

"Market conditions (days)"= 0

~ days

~ Loss of production time incurred as a result of market conditions.

|

max bucket pass= 10

~ sec

max dump time= 29

~ sec

max return time= 46

~ sec

max swing time= 18

~ sec

mean arrival rate ore= no of ore blocks required per month*PULSE TRAIN(0,Actual Time required to drill a block of ore ,1,100)

~ blocks/Month

~ production blocks required based on mine schedule per month

|

mean arrival rate waste= no of waste blocks required per month

~ blocks/Month

mean bucket pass= 5

~ sec

mean dump time=4.9

~ sec

mean return time= 10

~ sec

mean ROP= 3.4

~ min/m

mean ROP waste=2.2
~ min/m
mean subdrill fraction= 1
~ dmnl
mean stemming fraction= 0.5
~
mean swing time=6.6
~ sec
Measuring depth of holes= 1.3
~ min/hole
mechanical=10
~
~ Time taken for mechanical repairs to restore functionality.
|
"mechanical (days)"= 5
~ days
~ Time taken for mechanical repairs to restore functionality.
|
Mechanical Truck Availability= 0.85
~ [0.5,1,0.1]
meters charged with ANFO= 0
~ [0,15,0.1]
"Meters Drilled (y)"=100000
~
Meters possible to be drilled in a year= "monthly Drilling Requirement m/month"*no of months per year
~
min bucket pass=6
~ sec
min dump time= 1
~ sec
min return time=2
~ sec
min swing time= 2
~ sec
mineral royalty= 986
~
mintosecc=60
~ sec/min
"modifications (days)"= 1
~ days
~ Maintenance tasks initiated through applying a structured RCM-based process to modify the equipment or component to improve reliability, maintainability or availability
|

"monthly Drilling Requirement m/month"= ("required number of holes per month (Ore)"+"required number of holes per month(waste)")*"blast hole length(m)"
~ m
"multipass rod \"1\" for yes"=0
~ [0,1,1]
Loading Uncontrollable time hrs=("environmental uncontrollable (days)"+"external utilities (days)"+"market conditions (days)" +"non-scheduled holidays (days)"+"uncontrollable labour (days)")*daytohrs
~ hr/year
~ Time attributable to external factors beyond the control of the operation that affects the whole operation e.g., outside utilities such as power, water, rail systems, environmental disasters
|
"Net Cash Flow (Rand)"= INTEG ("Cash flow Rate (Rand/y)", 0)
~ Rand/y
"Net Present Value (Rand)"= INTEG ("Present Value (Rand/y)", 0)
~ Rand
"net profit (Rand/y)"="taxable income (Rand/y)"+"income tax (Rand/y)"
~ Rand/y
"new reserves found (t/y)"= 0
~ t/y
NO OF BLASTS PER MONTH= no of waste blocks required per month +no of ore blocks required per month
~ block
no of helpers per drill rig= 2
~
No of Loaders allocated to ORE loading= INTEGER("Direct loading time required per month ore (hr)"/Direct loading time available per loader per month+0.5)
~ hr
No of Loaders Allocated to Waste Loading= INTEGER("Direct loading time required per month waste (hr)"/Direct loading time available per loader per month +0.5)
~ hr
no of months per year= 12
~ Month/year
no of ore blocks required per month= INTEGER("required number of holes per month (Ore)"/planned number of holes per block ore\ +0.5)
~ blocks
no of resources= 2
~ [1,?,1]
~ no of people doing blasting tasks
|
No of Rigs allocated to ORE drilling=INTEGER("Direct Drilling time required per month ore (hr)"/Direct Drilling time available per drill per month +0.5)
~ dmnl

No of Rigs Allocated to Waste Drilling= INTEGER("Direct Drilling time required per month waste (hr)"/Direct Drilling time available per drill per month +0.5)
~ dmnl
no of rods=INTEGER("blast hole length(m)"/rod length+0.5)
~
no of shifts options= 2
~ dmnl [0,3,1]
no of shifts per day= IF THEN ELSE(automation switch>0, 0 ,no of shifts options)
~ [1,3,1]
no of waste blocks required per month= INTEGER("required number of holes per month(waste)"/planned number of holes per block waste +0.5)
~ block
no operator= 10
~
no operator frequency per year= 10
~ dmnl
"no operator lost time (hrs)"= loading no operator frequency per year*loading no operator related average lost time
~ hrs
no operator related average lost time= 8
~ hr
Loading Non Controllable Time= Non Scheduled Time+ Loading Uncontrollable time hrs
~ hrs
non controlling interest= 2523
~ mRand
"non scheduled holidays (days)"= 1
~ days
~ Loss of production time due to additional national holidays, such as \
voting days or time off in lieu of time worked in.
|
Non Scheduled Time=capital shotdown hrs+"loading nonproduction shift (hr)"+"public holiday (hrs)"+"section moves (hrs)"
~ hrs
~ Equipment time allocated as not being required in the production plan. (Only applicable to non-full calendar operations)
|
"non-cash deductions (Rand/y)"= "depreciation(Rand/y)"+"depletion (Rand/y)"+"amortization (Rand/y)"
~ Rand/y
number of boosters per hole= 1
~
"Number of caps per day (caps/month)"= "Caps per hole (cap/hole)"*number of holes per month drilled
~ caps/Day

~ number of caps required per day
|
umber of drills automated=6
~ [0,50,1]
"Number of holes per day(hole/day)"= ("overburden blasting per day (m3/day)"+"Ore blasting per day (m3/day)")/Variable Volume of rock per blast hole due to errors
~ holes/Day
~ number of holes to drill in a day based on production requirements
|
number of holes per month drilled="required number of holes per month (Ore)"+"required number of holes per month(waste)"
~
number of loads per hr=60/"planned loading cycle time (min)"
~
number of loads per month ore= bucket tons*hrs per month loading*number of loads per hr
~ t/Month
Number of Loads required annually= Tons to be moved per year/Truck payload
~ dmnl
number of maintenance events NEF=2
~ dmnl
"number of months/y"=12
~
number of passes required= INTEGER("100 ton loader switch"*"100 ton number of passes"+"400 ton loader switch"* "400 ton number of passes"+"650 ton loader switch"*"650 ton number of passes")
~ dmnl
~ number of passes per loader depends on the size of the machine as well as selected truck type, therefore, switch set 1 will determine the number of passes based on loader capacity therefore truck payload.
|
Number of trips possible per truck= annual truck loading time min/Truck Cycle Time
~ t/min
~ number of trips per truck
|
Number of trucks required= INTEGER(0.5+Number of Loads required annually/Number of trips possible per truck)
~ dmnl
operating cost of hauling per ton= fuel cost per ton for trucks+ lubricants cost per ton +tyre cost per ton moved
~ Rand/t
operating cost of loading per ton= 0
~ Rand/t
"Operating cost of truck/hr"=219

~ dollar/hr
~ literature reported cost
|
Operating expenses= 2.5451e+10
~ Rand
operating margin=0.38
~
~ percentage
|
operating profit=15316
~ mRand
"operating time (y)"= 14
~ year
~ reserve life is 14 nyears as reported in 2018 by company
|
Loading Operating time= Loading Uptime-loading Lost Time
~ hrs
~ Time during which the equipment is operating.
|
operating weight=100*"100 ton loader switch"+400*"400 ton loader switch"+650*"650 ton loader switch"
~ t [100,650,50]
~ value changes based on selected switch, or active loader type.
|
operational consumables=0
~
~ Internal controllable downtime associated with operational consumables not being available, i.e. Reagents, Magnetite, Flocculent. Excludes supply chain being responsible for it not being available.
|
"operational consumables (hrs)"=0
~ hr
~ Internal controllable downtime associated with operational consumables not being available, i.e. Reagents, Magnetite, Flocculent. Excludes supply chain being responsible for it not being available.
|
"operational delays (hrs)"= NO OF BLASTS PER MONTH*loading Standby due to blasting*12
~ hr
~ Delays experienced due to necessary production activities, such as blasting, housekeeping, full silo, preparation for maintenance
|
Loading Operational Stops hrs="Accident damage (hrs)"+"blockage (hrs)"+"diesel electricity air water (hrs)"+"operational consumables (hrs)"

+"Production stoppage (hrs)"+"tyres ropes chains (hrs)"+"unallocated (hrs)"+"unscheduled production stop (hrs)"
~ hr
~ Operational Stops Downtime: Necessary downtime attributable to Production
that renders the equipment inoperable e.g. replacement of consumables
|
operator activity sum manual= "Collaring time (min)"+"levelling -manual"+ "Reset Time (min)"+"Retract time (min)"+propel time manual +ave add rod time
~
Operator activity sum per blast hole Auto=("Collaring time (min)"+"Level Time (min)"+"Propel time auto (min)"+"Reset Time (min)"
+"Retract time (min)"+ave add rod time)
~ min/hole
operator cost to company=20000
~ Rand/Month
Operator proficiency auto= 0.95
~
operator proficiency manual=0.7
~
"optimum mean fragment size (m)"=0.15*Bc^(1/3)
~
"Ore blasting per day (m3/day)"="iron ore to be mined per day (t/d)"/"Density of ore (t/m3)"
~ m3/Day
~ blasting production per day in cubic meters
|
"Ore Drill Tempo (min/hole)"= "blast hole length(m)"*Instantaneous ROP ore+ IF THEN ELSE(automation switch>0, Operator activity sum per blast hole Auto , operator activity sum manual)
~ min/drill hole
~ Actual Tempo for Ore Drilling per drill hole in automated drilling
|
ore loaded per loader on available time= INTEG (number of loads per month ore, 0)
~ ton
"Loading Overall Equipment Effectiveness %"="Loading % Overall Utilisation (OU)"*"% Quality Achieved QT"*"% loading Rate Achieved RT"
~ dmnl
~ OEE = OEQU x (Actual Production Rate)/(Best Production Rate) x (Actual \
Quality)/(Target Quality)
|
"overburden blasting per day (m3/day)"=
"overburden planning (t/d)"/"density of overburden (t/m3)"
~ m3/Day
"overburden planning (t/d)"= "overburden rate (t/y)"/"working day planning (days/y)"

~
"overburden rate (t/month)"=stripping ratio*"Iron ore production planned (t/month)"
~ t/y
"overburden rate (t/y)"= "overburden rate (t/month)"*12
~
p80 fragmentation table([(0,0)-
(1400,30)],(0,12),(195.719,13.4211),(556.575,16.1404),(796.33,17.8947),(997.554\
,19.3421),(1200,21),(1395.72,22.5))
~ dmnl
P80 product size= 10
~ cm
~ final product size for crusher
|
"payback period (y)"= IF THEN ELSE("Net Cash Flow (Rand)">0, 1, 0)
~ dmnl
per ton drill and blast Cost of Ore=
"Consumable Cost per ton (ore)"+Drilling Fuel Cost per Ton ore + explosives cost per ton of ore
~ Rand/t
Per ton drill and blast Cost of Waste= consumable cost per ton waste + Drilling Fuel cost per ton of waste+explosives cost per ton of waste
~ Rand/t
petrol price with lookup=
price of petrol lookup(specific year)
~ Rand/lt
Pipe cost per meter ore= Drill pipe cost/Estimated Drill Pipe life ore
~ $/m
Pipe cost per meter waste= Drill pipe cost/Estimated Drill Pipe life waste
~ $/m
pipe handling5=0.05
~ dmnl
~ percentage of the mechanical availability of the drill rig
|
Pipe handle cycle fuel= "Fuel consumption per drill pipe handling (lt/hr)"*pipe handling5
~
Planned Blocks Queued= INTEG (scheduled ore blocks per month-Starting, 0)
~ blocks
Planned Blocks Queued waste= INTEG (scheduled ore blocks per month waste-Starting waste, 0)
~ blocks
"planned loading cycle time (min)"=
MAX((("100 ton load time (min)"+"400 ton load time (min)"+"650 ton load time (min)")*(1\
+Effect of fragmentation on loading cycle)),"loading cycle time {min}")
~ min

~ direct loading time based on the truck pay load capacity
|
planned number of holes per block ore= 150
~ blastholes/block
planned number of holes per block waste=300
~ blastholes/block
Planned Truck Availability= 0.87
~ dmnl [0.87,0.92,0.01]
~ Expected maximum when autonomous
|
Planned Truck Utilisation= 0.76
~ dmnl [0.76,0.85,0.01]
Powder factor= explosive mass per hole/(Bench Height*Burden achieved*Spacing achieved)
~ kg/m3
"Powder Factor (kg/cu.m)"= "Total bulk explosive (kg/hole)"/Variable Volume of rock per blast hole due to errors
~
~ powder factor
|
Power consumption per ton= The bond work index*11*(1/SQRT(P80 product size)-1/SQRT(F80 feed size))
~ KWh/t
Preparing the site= 20
~ min
~ barricades, inspections, sign boards, etc
|
"Present Value (Rand/y)"=(("capital cost (Rand/y)"+"cash flow after-tax (Rand/y)")/(1+("discount rate (%)"/100\
))^(Time*"(1/y)"))
~ Rand/y
"price (Rand/t)"=17.1
~ Rand/t
price of electricity lookup([(2015,60)-(2020,100)],(2015,70),(2016,80),(2017,85),(2018,90),(2019,92),(2020,94))
~ Rand/kW
price of petrol lookup([(2015,11.05)-(2021,17)],
(2015,11.05),(2016,11.46),(2017,13.3),(2018,13.54),(2019,14.69),(2020,15.52),
(2021,16.15))
~ Rand/lt
~ PRICE OF PETROL
|
Loading Primary Production time= Loading Operating time-Secondary Non Production Work
~ hr/year
Priming=2.36

~ min/hole
Production stoppage= 5
~
~ All Production reasons that render the equipment inoperative
|
"Production stoppage (hrs)"= 5
~ hr
~ All Production reasons that render the equipment inoperative
|
Progressing= DELAY3(Starting, Actual Time required to drill a block of ore)
~ blocks/Month
Progressing waste= DELAY3(Starting waste, Actual time required to drill a block of waste)
~ blocks/Month
propel cycle fuel= "Fuel consumption per drill propel (lt/hr)"*propel7
~
"Propel time auto (min)"= 0.9
~ min
propel time manual= 2.4
~ min
propel7=0.07
~ dmnl
~ percentage of the mechanical availability of the drill rig
|
"public holiday (hrs)"= loading public holidays*daytohrs
~ hr
~ National holiday defined as a non-working day by the operation
|
public holidays=12
~
Queueing at shovel time= IF THEN ELSE(automation switch>0, 4.9 , 5.7)
~ min [0,?]
~ it was reported that queuing times reduced from 5.7 to 4.9 at the loader \
(assuming this is due to shovel loading efficiencies linked to improved \
fragmentation of the muckpile)
|
queuing at crusher time= 0
~
"required number of holes per month (Ore)"=("Iron ore production planned (t/month)" /"Density of ore (t/m3)")/Variable Volume of rock per blast hole due to errors
~ hole/Month
"required number of holes per month(waste)"=("overburden rate (t/month)"/"density of overburden (t/m3)")/Variable Volume of rock per blast hole due to errors
~ hole/Month
"Reset Time (min)"= 0.53

~ min
"Retract time (min)"= 0.85
~ min
Return time= RANDOM NORMAL(min return time, max return time, mean return time, std return time, seed)
~ sec
RMD= 50
~ dmnl
~ Rock Mass Description,10 for friable to 50 for hard massive
|
"Rock Density g/cc"=2.58
~ gr/cc
"Rock Factor (A)"=0.06*(JPS+JPO+25*"Rock Density g/cc"-50+Rock UCS/5)
~ dmnl
Rock sonic velocity= 6000
~ m/s
~ Use about 4000 for soft weathered rock and 6000 for hard brittle rock
|
Rock UCS= 250
~ MPa
rod length= 19
~ m
RWS= 98
~ dmnl
~ Relative Weight Strenth, 100 f0r ANFO, 98 for Emulsion
|
safety= 10
~
~ Downtime due to safety inspections and safety related interventions
|
Loading Scheduled maintenance time hrs= (("maintenance related delays (days)"+"service and shotdowns (day)"+"modifications (days)")*daytohrs+"follow-on work (hrs)"+"loading inspections (hrs)")
~ hrs
~ Scheduled Maintenance Downtime as a result of maintenance work included in \
the confirmed weekly maintenance plan
|
scheduled ore blocks per month=mean arrival rate ore
~ blocks/Month
scheduled ore blocks per month waste=mean arrival rate waste
~ blocks/Month
"drilling seat change time per day (min)"=20
~ min
Secondary Non Production Work= 10

~ hrs
~ Equipment is operational, but performing non-production activities such as training,
|
section moves= 8
~
~ Relocation of equipment due to operational requirements. Performed in nonproduction time for those operations that do not operate 24/7/365. For example, mining section or major equipment moves where the impact on production is accounted for in the budgeted annual plan. Note: The time the move exceeds the plan must be recorder as a Production Delay.
|
"section moves (hrs)"= 0
~ hr
~ Relocation of equipment due to operational requirements. Performed in nonproduction time for those operations that do not operate 24/7/365. For example, mining section or major equipment moves where the impact on \
production is accounted for in the budgeted annual plan. Note: The time the move exceeds the plan must be recorder as a Production Delay.
|
seed= 0
~ dmnl
"service and shutdowns (day)"=10
~ days
~ Maintenance tasks associated with predetermined services and shutdowns
|
service delays= 30
~
~ Production delays experienced due to the installation of, or lack of, services, such as survey, ventilation, construction, etc.
|
shock tube cost per metre= 1
~ [1,30]
shovel loading capacity= Shovel Mechanical Availability*Shovel theoretical Loading capacity per hr*Utilisation of Shovel
~
Shovel maintenance cost per hr loading= 203
~ US$/hr
Shovel Maintenance cost per hour idling=18.65
~ US$/hr
Shovel Mechanical Availability= 0.95
~ dmnl [?,1,0.1]
Shovel theoretical Loading capacity per hr= 4395
~ t/hr
~ 50 minute operating time per hour is a typical time

|
Sieve 15= 250
~ mm
Sieve1= 3.35
~ mm [0,1,0.1]
Sieve10= 53
~ mm
Sieve11= 75
~ mm
Sieve12= 100
~ mm
Sieve13= 150
~ mm
Sieve14= 200
~ mm
Sieve16= 300
~ mm
Sieve17= 350
~ mm
Sieve18= 425
~ mm
Sieve19= 600
~ mm
Sieve2= 4.75
~ mm [0,5,0.1]
Sieve20= 850
~ mm
Sieve3= 6.3
~ mm
Sieve4= 8
~ mm
Sieve5= 13.2
~ mm
Sieve6=19
~ mm
Sieve7= 25
~ mm
Sieve8= 28
~ mm
Sieve9= 38
~ mm
size fraction 1 to 2=Fraction 2 Passing-Fraction 1 Passing
~ dmnl
size fraction 10 to 11= Fraction 11 Passing-Fraction 10 Passing

~
size fraction 11 to 12= Fraction 12 Passing-Fraction 11 Passing
~
size fraction 12 to 13= Fraction 13 Passing-Fraction 12 Passing
~
size fraction 13 to 14= Fraction 14 Passing-Fraction 13 Passing
~
size fraction 14 to 15= Fraction 15 Passing-Fraction 14 Passing
~
size fraction 15 to 16= Fraction 16 Passing-Fraction 15 Passing
~
size fraction 16 to 17= Fraction 17 Passing-Fraction 16 Passing
~
size fraction 17 to 18= Fraction 18 Passing-Fraction 17 Passing
~
size fraction 18 to 19= Fraction 19 Passing-Fraction 18 Passing
~
size fraction 19 to 20= Fraction 20 Passing-Fraction 19 Passing
~
size fraction 2 to 3=Fraction 3 Passing-Fraction 2 Passing
~
size fraction 3 to 4=Fraction 4 Passing-Fraction 3 Passing
~
size fraction 4 to 5=Fraction 5 Passing-Fraction 4 Passing
~
size fraction 5 to 6=Fraction 6 Passing-Fraction 5 Passing
~
size fraction 6 to 7=Fraction 7 Passing-Fraction 6 Passing
~
size fraction 7 to 8=Fraction 8 Passing-Fraction 7 Passing
~
size fraction 8 to 9=Fraction 9 Passing-Fraction 8 Passing
~
size fraction 9 to 10=Fraction 10 Passing-Fraction 9 Passing
~
size fraction feed 1 to 2= size fraction 1 to 2*Block Volume
~ m3
size fraction feed 10 to 11= size fraction 10 to 11*Block Volume
~ m3
size fraction feed 11 to 12= size fraction 11 to 12*Block Volume
~ m3
size fraction feed 12 to 13= size fraction 12 to 13*Block Volume
~ m3
size fraction feed 13 to 14= size fraction 13 to 14*Block Volume

~ m3
size fraction feed 14 to 15= size fraction 14 to 15*Block Volume
~ m3
size fraction feed 15 to 16= size fraction 15 to 16*Block Volume
~ m3
size fraction feed 16 to 17= size fraction 16 to 17*Block Volume
~ m3
size fraction feed 17 to 18= size fraction 17 to 18*Block Volume
~ m3
size fraction feed 18 to 19= size fraction 18 to 19*Block Volume
~ m3
size fraction feed 19 to 20= size fraction 19 to 20*Block Volume
~ m3
size fraction feed 2 to 3= size fraction 2 to 3*Block Volume
~ m3
size fraction feed 3 to 4= size fraction 3 to 4*Block Volume
~ m3
size fraction feed 4 to 5= size fraction 4 to 5*Block Volume
~ m3
size fraction feed 5 to 6= size fraction 5 to 6*Block Volume
~ m3
size fraction feed 6 to 7= size fraction 6 to 7*Block Volume
~ m3
size fraction feed 7 to 8= size fraction 7 to 8*Block Volume
~ m3
size fraction feed 8 to 9= size fraction 8 to 9*Block Volume
~ m3
size fraction feed 9 to 10= size fraction 9 to 10*Block Volume
~ m3
size fraction feed above 20= size fraction more than sieve 20*Block Volume
~ m3
size fraction more than sieve 20= 1-Fraction 20 Passing
~
slag factor of shock tube= 1.2
~
~ extra length for tying and other wastage
|
"Smoothing time (in months) waste"= 1
~ Month [1,7,1]
"Smoothing time (in months)"=1
~ Month [1,7,1]
Spacing by design= 6
~

Spacing achieved=Spacing by design+ IF THEN ELSE(automation switch>0,0,RANDOM NORMAL(-1, 1,0 ,0.5, 0.1\))
~
"Specific charge per ton of rock (kg/ton)"= "Total bulk explosive (kg/hole)"/"Breakage weight per hole (tons) - ore"
~ kg/ton
~ Specific charge in kg per ton of rock
|
specific year=2018
~ [2015,2025,1]
staff related= 365*0.25
~
~ Delays experienced due to travelling, transport, shift change, etc. (Measured at section or module level and not at equipment level)
|
Standby due to blasting= 2
~ hr
Starting=MIN(Desired no of blocks, Feasible no of blocks)
~ blocks/Month
Starting waste= MIN(Desired no of blocks waste, Feasible no of blocks waste)
~ blocks/Month
std bucket pass= 1
~ dmnl
std dump time=2.7
~ dmnl
std return time= 4.7
~
std swing time=2.2
~ dmnl
Stemming=1.25
~ min/hole
~ with the help of bobcat
|
stemming length achieved=Stemming Length by design+ IF THEN ELSE(automation switch>0,1, fraction of stemming length variation)
~
Stock of Drilled holes for ore= INTEG (drilled holes arriving ore, 0)
~ blastholes
stripping ratio= 4.7
~ m3/m3
|
Subdrill by design= 2.5
~ m

actual subdrill= Subdrill by design+ IF THEN ELSE(automation switch>0,1, fraction of over drill)
~
sundays= 52
~
supply chain=10
~
~ Inability to perform work due to supply chain related delays, such as stock outs, incorrect materials purchased
|
Swing time= RANDOM NORMAL(min swing time, max swing time, mean swing time, std swing time, seed)
~ sec
switch control mine life= 0
~ [0,1,1]
"Target Quality (TQ)"= 0.12
~
~ Target Quality percent
|
"tax rate (%)"= 28
~ dmnl
~ tax rate in SA in 2008
|
"Taxable income (Rand/y)"= IF THEN ELSE("gross profit (Rand/y)">"non-cash deductions (Rand/y)","gross profit (Rand/y)" -"non-cash deductions (Rand/y)" , 0)
~

The bond work index= 0.15
~ KWh/t
"Throoughput tonnage (tons/h)"= "Iron ore production planned (t/month)"/(16*30)
~
throughput tons per month="Throoughput tonnage (tons/h)"*24*30
~ t/Day
Time available for blasting in a month= 30*24/2
~ hr/Month
~ legally actual charging and blasting can be done during day times
|
Tipping Time= 2.5
~ min
Tire cost= 38000
~ US$
Tire life in hours= 5600
~ hr/tire
~ komatsu 730 7200hrs tyre life tyre size 40 R7

cat777 5600 hrs 27x49
|
Toe Energy Factor= (((Bench Height*0.3+actual subdrill)*"Charge mass / m")/(Bench Height*0.3*Burden achieved*Spacing achieved))*(RWS/100)
ton2 kg= 1000
~ kg
"tons loaded per shovel (y)"= Annual Shovel operating hours*shovel loading capacity
~ t/year
tons possible to be loaded in a year= ("Iron ore production planned (t/month)" +"overburden rate (t/month)") *loading no of months per year
|
Tons to be moved per year= "iron ore demand (t/y)"+"overburden rate (t/y)"
~ t/year
"Total bulk explosive (kg/hole)"= amount of anfo per hole +amount of emulsion per hole
~ Blasthole charge
Total Calendar Hours T000=365*24
~ hrs
~ Total Time Annualized=24 x 365 = 8760
"Total Cost of Explosives (rand/month)"= cost of bulk explosive per blast hole*number of holes per month drilled
~ Rand/Month
~ total cost of explosives per day
|
Total Drilling Time required per year= no of months per year*"direct drilling hours required per month (total)"
~ hr/y
Total Loading Time required per year=loading no of months per year*"direct loading hours required per month (total)"
~ hr
TOTAL NUMBER OF Loaders REQUIRED= INTEGER("direct loading hours required per month total)"/Direct loading time available per loader per month +0.5)
~ dmnl
~ in order to avoid drill time shortage nearest integer should always be rounded to the higher end of the range. For that reason, 0.5 is added to the formula
TOTAL NUMBER OF RIGS REQUIRED=INTEGER ("direct drilling hours required per month (total)"/Direct Drilling time available per drill per month +0.5)
~ dmnl
~ in order to avoid drill time shortage nearest integer should always be
rounded to the higher end of the range. For that reason, 0.5 is added to
the formula
Total Truck Time Available Annually=8760
~ hr
total trucking time based on production= Number of trips possible per truck*Truck Cycle Time/60

~ hr/year
Total Useful Life of Machine= IF THEN ELSE(automation switch>0,useful life autodrill, useful life manual)
~ hr
Total Useful Life of Machine Automatic= 100000
~ hr
Truck Cycle Time= Loading Time minutes +Queueing at shovel time +queuing at crusher time +Tipping Time +Two way travel time
~ min
~ truck cycle time
Truck payload= 186
~ t
~ Komatsu 730E, 30 of them in the fleet
|
Two way travel time= Hauling Distance/(Hauling speed empty+ Hauling Speed full)*hrs to min
~ hr
tyre cost per hour= "dollar & rand exchange"*Tire cost/Tire life in hours
~ Rand/hr
tyre cost per ton moved=tyre cost per hour*Truck Cycle Time/Truck payload
~ Rand/t
tyres ropes chains= 100
~
~ All downtime related to tyres, ropes or chains or cable
|
"Tyres ropes chains (hrs)"= 0
~ hr
~ All downtime related to tyres, ropes or chains or cable
|
unallocated= 5
~ Downtime, which cannot be classified into the above or for when no reason has been recorded, such as balancing Direct Operating Hours with that of the Service Meter Reading.
|
"Unallocated (hrs)"= 5
~ hr
~ Downtime, which cannot be classified into the above or for when no reason has been recorded, such as balancing Direct Operating Hours with that of the Service Meter Reading.
|
uncontrollable labour= 8
~ Impact as a result of operation wide industrial action.
"Uncontrollable labour (days)"= 8
~ days
~ Impact as a result of operation wide industrial action.
|

unit cash cost rand per ton ore= 240
~ Rand/t
~ 2017 cost of mining per ton mined
"Unit cost of caps (rand/cap)"= 150
~ Rand [1,?]
~ Unit cost of cap
|
unscheduled delays= 10
~ All external and internal delays experienced by the Maintenance department during Unscheduled Maintenance that causes an interruption in service delivery i.e., awaiting spares, awaiting transport, awaiting labour, awaiting supplier.
|
"Unscheduled delays (days)"= 10
~ days
~ All external and internal delays experienced by the Maintenance department during Unscheduled Maintenance that causes an interruption in service delivery i.e., awaiting spares, awaiting transport, awaiting labour, awaiting supplier.
|
Unscheduled maintenance time D100= control and instrumentation +electrical +hydraulic +mechanical +unscheduled delays
~ hr
~ Unscheduled Maintenance Downtime as a result of maintenance work not included in the confirmed weekly maintenance plan
|
Unscheduled maintenance time hrs= ("control and instrumentation (days)"+"electrical (days)"+"hydraulic (days)"+"mechanical (days)" +"unscheduled delays (days)")*24
~ hrs
~ Unscheduled Maintenance Downtime as a result of maintenance work not included in the confirmed weekly maintenance plan
|
unscheduled production stop= 0
~ All downtime experienced due to Unscheduled production activities required, excludes Tyres / Ropes / Chains.
|
"Unscheduled production stop (hrs)"= 0
~ hr
~ All downtime experienced due to Unscheduled production activities required, excludes Tyres / Ropes / Chains.
|
Loading Uptime= Controllable time- Equipment down time
~ hr
~ Equipment time available for production activities.
useful life autodrill= 120000
~

useful life manual= 90000
~
Utilisation of Shovel= 0.85
~ [?,1,0.1]
VOD=4500
~ m/s
~ explosive velocity of detonation
|
Variable Volume of rock per blast hole due to errors= Burden achieved*Spacing achieved*Bench Height/COS ("Blast hole inclination from vertical (degree)")
~ m3
"working day planning (days/y)"= 300
~ days/y
x80= Expected Mean fragment size/(0.4306)^(1/uniformity index)
~ cm
~ Formula is based on Rajpot Muhammed's thesis, pg 39)
|"yield of broken ore (t/m)"= "Density of ore (t/m3)"*"Geometric Yield of broken volume per m drilled (m3/m)"
~ t/m
Yield of broken waste= "density of overburden (t/m3)"*"Geometric Yield of broken volume per m drilled (m3/m)"
~ t/m
"yield of broken waste (t/m)"= "density of overburden (t/m3)"*"Geometric Yield of broken volume per m drilled (m3/m)"