A THEORETICAL ANALYSIS OF THE IMPLICATIONS OF COMMINUTION PRACTICES ON OPEN PIT MINE PLANNING

by

RORISANG GOMOLEMO THAGE

submitted in accordance with the requirements for
the degree of

MAGISTER TECHNOLOGIAE

in the subject

ENGINEERING: CHEMICAL

at the

UNIVERSITY OF SOUTH AFRICA

SUPERVISOR: PROF FK MULENGA

December 2020

DECLARATION

I hereby declare that this dissertation submitted for the degree M Tech: Chemical Engineering, at the University of South Africa, is my original work and has not previously been submitted to any other institution for any degree or examination.

I further declare that all sources cited or quoted are indicated and acknowledged through a comprehensive list of references.

Rorisang Gomolemo Thage

University of South Africa
Johannesburg, South Africa

DEDICATION

I dedicate this work to my beloved mother Mrs. Anna Thage for the countless prayers, sacrifices, unconditional support, love, and patience throughout the execution of this work. Without her support, I would not have been able to complete this work. I am forever indebted to her and can never ask for a greater mother.

I further dedicate this dissertation to my late father Mr. Tshone “Ditao” Thage. He passed on when I was fourteen years old and never lived to see my capabilities. He is the architect of all that I do because I know that if he were alive, he would be proud to see the fruits of his work and love.

To my siblings Rebecca and Refilwe, this Master’s degree is the result of your encouragement and unconditional love. You have been and always will be the greatest sisters any younger sister would wish to have, and I thank you so much for everything.

To my niece, Keorapetse, and nephews Oratile, Reratilwe, and Lore, may this work remind you that you are capable of anything you want to achieve. If Aunty can do it, so can you!

To my friends Katlego Modise and Lehlogonolo Matonya, this dissertation is the result of all your long late-night video calls showing immerse support and believing in me. Our friendship is very close to my heart thanks you so much for everything.

All that I am and hope to be, I owe it to my beloved family.

ACKNOWLEDGEMENTS

Firstly, I would like to thank my supervisor, Prof Francois Mulenga, for his guidance, support, and patience throughout the execution of this research. His never-ending words of encouragement and motivation instilled self-belief and confidence in me throughout my research work. I am indebted to him and the University of South Africa for allowing me to do this work.

I am ever grateful to my mentor Ms. Rorisang Lebethe for her selfless guidance and continued support. She has always been there whenever I needed her, and always believed in me even when I wanted to give up. I am thankful to have her in my life.

Deepest gratitude to my colleagues Mr. Thapelo Mosetlhe and Prof Adedayo Yusuff for reading through some of my work and encouraging me to soldier on.

Finally, I take this opportunity to express my heartfelt gratitude to those who have contributed towards the successful completion of my research in different ways including the MiningMath Associates who provided me with the opportunity to freely use SimSched and supported me all the way through.

ABSTRACT

The implications of comminution practices on the planning of a typical open pit mine was investigated in this study by means of computer simulation. The objective was to assess the effects of mining costs as well as processing costs on the production plan of a typical open pit mine.

For the purpose of the research, MineLib, an open library of ore body models was consulted. This led to the selection of a copper-gold ore body named "*Newman1*" for use in the strategic mine optimisation. Various scenarios were considered in order to highlight the contribution of comminution costs to the mine plan. In all the simulated scenarios, the objective function was to maximise the Net Present Value (NPV). And in terms of simulation setup, the comminution costs and cut-off grades were systematically varied from 70 % to 140 %. It was hence possible to investigate their effects on the NPV of the *Newman1* ore body using SimSched, a freeware for mine optimisation and planning.

Results showed that there is a great opportunity to increase the NPV of the *Newman1* block model by adjusting the contribution of processing costs in general and comminution costs in particular. This can be achieved for instance by controlling the policy of cut-off grades, lowering production costs, and increasing throughput.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

ACRONYMS

AG	Autogenous Grinding
ANBA	Ammonium Nitrate Blasting Agent
ANFO	Ammonium Nitrate Fuel Oil
ar2tech	Advanced Resources and Risk Technology
ASM	Small-Scale Mining
Au	Gold
COG	Cut-off grade
CSV	Comma-Separated Values
Cu	Copper
DBS	Direct Block Scheduling
GPS	Global Positioning System
HPGR	High-Pressure Grinding Rolls
LG	Lerchs-Grossmann algorithm
LP	Linear programming
NG	Nitro-Glycerine
NPV	Net Present Value
OPOPS	Open Pit Design and Production Scheduling
SABC	Semi-Autogenous and Ball milling with Crusher
SAG	Semi-Autogenous Grinding
SGeMS	Stanford Geostatistical Modelling Software
SimSched DBS	SimSched Direct Block Scheduler
UG	Underground mine
USA	United States of America
VOD	Velocity of Detonation

Chapter 1 Introduction

1.1 Background

Comminution is one of the most important phases in the extraction and processing of mineral ores. The verb “to comminute” means *to reduce to minute particles* (Soanes and Hawker, 2005). Comminution is therefore a generic term that refers to the reduction in rock size. In underground and surface mining operations, comminution or size reduction is required to expose valuable minerals from the rock and enable their easy recovery (Moema et al., 2009).

Concordant studies have shown that comminution alone represents 50 – 70 % of the processing costs (Musingwini, 2016; Daniel and Lewis-Gray, 2011; Ballantyne and Powell, 2014). As such, an opportunity exists to systematically investigate the effects of comminution costs on a typical surface mining operation.

Three broad comminution stages are available to the mineral processing industry: blasting, crushing, and milling. Since much energy is used during comminution, enormous cost savings can be obtained through improved comminution practices.

Several comminution techniques have been used with varied degrees of success. However, the economic implications of choosing a particular comminution strategy on mine planning are yet to be explored. Indeed, mine planning involves determining the maximum profitable excavation sequence throughout the lifespan of the mine. It is proposed in this research work to consider a computer-based block model of a mine for simulation. The ore body model was selected from MineLib, a database of publicly available block models. The amount of ore contained in each discretized block of the selected ore body was characterized in terms of waste and total mineable tonnage. Mining and processing costs were then accounted for in the estimation of the expected profit of individual blocks. For this, the relative proportion of processing costs was varied to mimic the contribution of comminution costs to the profitability of the generic open pit mining project. The findings led to a better appreciation of the comminution-mining interrelation from the point of view of mine planning. The findings finally had the potential to influence how ore bodies are evaluated for effective and profitable mineral exploitation.

1.2 Problem statement

Comminution is an essential step to the liberation and recovery of minerals. However, an enormous amount of energy is expended especially when milling is involved. This is because existing milling technologies waste energy as a result of the unnecessary and unavoidable over-grinding of material. This in turn has a bearing on the economics and the effectiveness of downstream processes (Hlabangana et al., 2016).

Khumalo (2007) reported that comminution is responsible for roughly 50 % of the total energy of a mineral processing plant. When considered as part of the entire mine production chain, comminution accounts for up to 70 % of the total operational costs (Radziszewski, 2013; Nadolski et al., 2014). Here, the mine production chain is considered to entail mining, comminution, and concentration.

Now consider this, the decision to mine a block and send it the comminution plant instead of the tailings dam is dictated by the economic value of the mined block. However, the value of the block is dependent on the operational costs incurred along the chain. These costs then affect the mine production plan as it seeks to optimize the sequence of extraction of waste and ore mined out over time. Ultimately, the net value of the ore body over the entire span of the mine may be reduced. The problem is that little has been done to explore the contribution of comminution costs to open pit mine optimization (Napier, 2015); hence, the present research study.

1.3 Objectives and purpose of the research study

The purpose of this dissertation is:

- To optimize the proportional influence of mining and processing costs on the production plan of an open pit mine.
- Explore the relationship between processing costs and specifically comminution costs on the anticipated net profit value of the ore body.
- To determine the contribution of reduced comminution costs to open pit mine plan and the associated implications in terms of comminution practice.

1.4 Outline of the dissertation

The present dissertation is organised as follows:

Chapter 2 provides a literature review on the principles of and relevant research done on open pit mine optimisation. It also presents the impact comminution has on the overall costs of the mine. It further looks at the Lerchs-Grossman algorithm used in pit optimization.

Chapter 3 discusses how the procedure adopted in the collection of data for pit optimization. It also presents SimSched Direct Block Scheduler, the software used for mine optimization. Finally, simulation scenarios and optimization are presented in the chapter.

Chapter 4 presents the results generated from the simulation work. These results are then made sense of in Chapter 5. Finally, a summary of the findings from this research dissertation is covered in Chapter 6. In addition to this, concluding remarks and suggestions for future work are made.

Chapter 2 Literature review

2.1 Introduction

Open pit mining is well established in shallow and intermediate ore bodies where it is considered to be the most economic compared to underground mining methods. The economics of open pit mining is governed by several factors such as the flexibility and safety of the operation, mining costs, processing costs, and the price of minerals and metals. The uncertainty attached with commodity prices for instance affects the sustainability of the mining project as well as the confidence with which optimal operating conditions can be reached. The problem is exacerbated by the fixed market price of minerals and metals. Several studies have argued that mining operations can only control operational and processing costs but not economic factors such as metal price (e.g. Asad, 2005; Asad and Topal, 2011; Ramazan and Dimitrakopoulos, 2013). Although these studies have explored the possibility controlling the major economic factors, but still, their rendition revolved around mining and processing costs.

Studies by King (2011) and Ataei et al. (2008) are interesting in that they report the processing costs to be the most important economic factor of an open pit operation. The processing costs in these studies include crushing, grinding, and concentration/segregation.

In the mining industry, crushing and milling are collectively referred to as comminution. This is despite the fact that the comminution or size reduction of rock essentially commences with blasting. That is why in this dissertation, the term “comminution” is used to mean the reduction of the size of blasted rock fragments by crushing and/or milling with the view to liberating the mineral of interest from the unwanted fraction (Wills and Finch, 2015).

The level of size reduction required of a comminution operation is dictated by the demand and specifications of the market or by the requirement of subsequent separation stages to be undergone. However, comminution alone is responsible for 30 – 70% of the total operational costs of typical mining operations (Radziszewski, 2013; Nadolski et al., 2014). Because of this, comminution has drawn a lot of research initiatives designed to reduce processing costs and energy consumption. A detailed review of various factors contributing to the poor efficiency of comminution operations is documented in this chapter. This is followed by a description of open pit mine planning. Finally, algorithms and tools used for the optimization of the net profit value of ore bodies exploited by open pit mining are reviewed.

2.2 Comminution

Comminution operations entails the reduction of rock fragments as large as 1 meter or larger to particles as small as 25 microns (Powell et al., 2011). Unfortunately, a large fraction of energy needed is wasted in the process. For example, approximately 11 % of the energy available is actually utilized for the comminution by milling of particles from 20 mm to 100 microns (Powell et al., 2011; Tromans and Meech, 2002). A large fraction of the world electric power usage is attributed to this inefficient and costly method (Roth and Ambs, 2004). In 1981, comminution was estimated to amount to about 2 % of the entire electric usage of power in the United States of America. This has since increased (Kawatra et al., 2005). So, improvements however minute may imply significant economic savings not only for the comminution process itself but also for the mine production chain.

Some facts are self-evident; first, commodity prices fluctuate erratically and are driven by the market. Second, large high-grade and easy-to-process-ore deposits are uncommon. Last, energy conservation is a matter of civil and often national strategic interest. However, mineral processors adapt to the changing economic environment by specifically adjusting operating conditions at the comminution stage. The challenge is that in current low-grade mining operations, the magnitude of consumables and energy used is substantial, especially around the comminution section (Abouzeid and Fuerstenau, 2009; Charles and Gallagher, 1982). Appropriate design of the comminution circuit is therefore critical, particularly for large-scale hard-rock mining projects. Numerous options can be resorted to when such a circuit is designed. Some design strategies rely on long-established technologies while others are based on more recent advancement. And in some cases, technologies available have been improved from the experience of current mining ventures (Barratt and Sherman, 2002; Labys and Thomas, 1975).

In the next subsections, the fundamental concepts and commonly used technologies in comminution operations are succinctly reviewed. This background knowledge generally guides the selection of appropriate configuration of comminution circuits and associated equipment. A description is also made of supporting theories behind drilling and blasting with the understanding that these activities are ahead of comminution.

2.2.1 Fundamental concepts

The meaning of the verb comminute in the dictionary is to reduce to minute particles (Soanes and Hawker, 2005). In the mining and mineral processing industry, the term comminution applies primarily to crushing and grinding even though the size reduction of rocks begins with blasting (Rosario, 2010).

Comminution is an important stage in the processing of minerals as it is necessary to release the valuable minerals from the gangue. The breakage action can also be regarded in the light of a new surface of mineral particles being formed. For metallurgical extraction processes such as leaching and flotation, growing the mineral surface is important.

Comminution encompasses physical methods of reducing the ore/rock to the desired size with the view to releasing mineral species without altering the physical and chemical equities of the rock.

There exist many techniques of comminution. Figure 2.1 illustrates a typical circuit entailing a sequence of the following comminution equipment: a gyratory crusher, a cone crusher, a semi-autogenous (SAG) mill, and a ball mill. The comminution circuit typically reduces the size of rock in stages from approximately 1 m to below 0.5 mm.

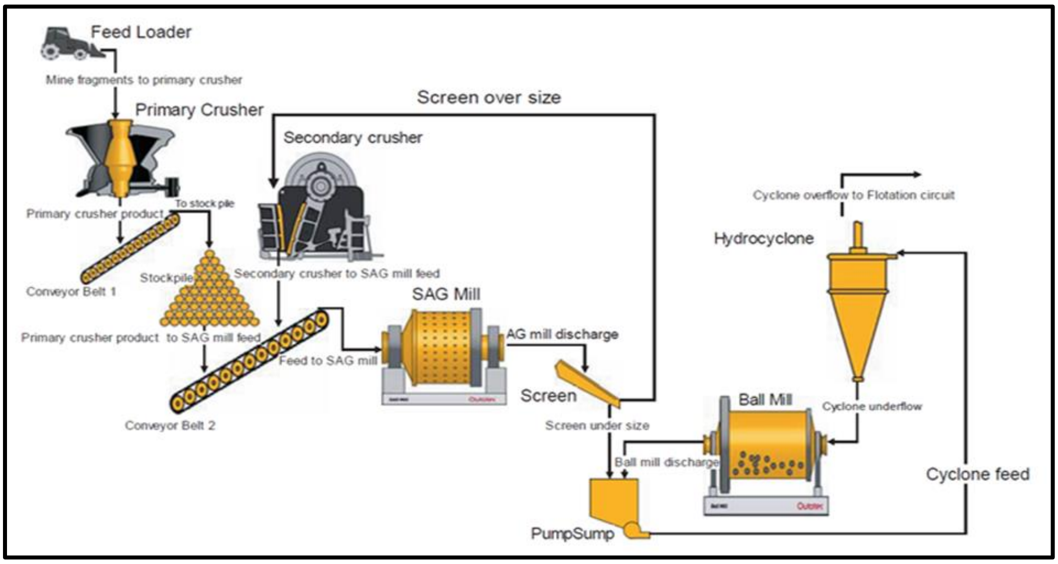

Figure 2.1 A basic flowsheet of an ordinary comminution circuit (Rosario et al., 2009)

Crushing is generally done by impacting or compressing the run-of-mine rock against heavy-duty metallic plates. SAG milling on the other hand takes advantage of the presence of lumps of rock that tumble inside the cylindrical rotary vessel and break rock particles by abrasion, attrition, and impact. The product from the SAG mill is subsequently broken inside a ball mill. The latter uses spherical steel balls as grinding media that subsequently pulverise the material to less than 0.5 mm.

Comminution circuits are always built around comminution equipment and size classifiers as exemplified in Figure 2.1. Size classifiers are a family of equipment aimed at separating particle fragments based on their size. This ensures that particles ready to move to the next stage are not unnecessarily crushed or milled further thereby wasting energy. The separation of coarse particles is done using vibrating screens. In Figure 2.1 for example, the SAG mill discharge is processed through a screen so that the oversize fraction is sent back to the pebble crusher. For smaller particles as is characteristic of the ball mill discharge, hydrocyclones are most commonly used.

The comminution circuit in Figure 2.1 would be used to break a run-of-mine ore and release or liberate mineral particles from the rock matrix containing them. The crushing section would be run dry whereas SAG and ball milling would typically be done wet. The addition of water improves the efficiency of comminution while ensuring that the slurried material easily flow through the tumbling mills. The final product coming out of the circuit makes its way what is known as concentration or mineral upgrading. Concentration basically separate liberated mineral particles from the gangue. Flotation, leaching, gravity separation are some concentration operations that can be used for the purpose. The choice of concentration depends on the nature and type of orebody being mined.

An important point to make is that comminution circuits are the bottleneck of any mineral processing plant. This is because comminution circuits define the throughput and efficiency of the plant. In addition to this, the objective of comminution circuits is to maximize throughput while producing a targeted size distribution. If the material is hard, this typically means that throughput must be limited to generate the targeted particle size. This is done by adjusting parameters such as the classification cut-off size, the volume of grinding media in the mill, and the rotational speed of tumbling mills.

2.2.2 Drilling and blasting

The production of run-of-mine rock to be fed into a comminution circuit hinges on drilling and blasting of the rock mass. The purpose of rock drilling is to open holes within the rock mass with appropriate distribution and geometry. Rock drilling is done nowadays with the assistance of rotary-percussive mechanical rockdrills echoing hand-boring where a hammer batters the chisel and rolls it into the pit. Drilled holes are charged with explosives and a designed sequence is performed by detonating the holes (Jimeno et al., 1995). The detonation is aimed at causing cracks in the surrounding rock fragmenting the rock while making exploitation possible.

It is evident that the goal of blasting is to achieve ample fragmentation cost-effectively and safely. Fragmentation should produce a large volume of broken fragments of rock of average size without excessive dust or fine content. This enables the operation to dig out (i.e. load and remove) a planned volume of rock from within specified limits.

From a principle point of view, mining explosives or fracture explosives produce a high-intensity shock movement and a large volume of gas upon explosion. The gas expands rapidly within the confined cavity of the drill hole. It then travels through small cracks present in the rock and produces new fractures thereby breaking the rock.

The first development of mining explosives was gunpowder (or black powder). It is a low-powered product that is still used in specific situations such as the processing of dimensional stones. Next, Nitro-Glycerine-based explosives became prevalent in mining for around eight years until the early twentieth century (Persson et al., 1993). Despite being kept in controlled environments, Nitro-Glycerine (NG) items were inherently unstable and dangerous to use, especially with age and when exposed to the sun and heat. NG-based goods have been almost entirely replaced by products such as emulsions, micro balloons, tiny glass, or plastic spheres that absorb oxygen and sensitize the substance in the blend. They are routinely prepared in modern practice only when they are charged through a hole and are non-explosive before the moment (Persson et al., 1993).

Ammonium Nitrate Fuel Oil (ANFO) is the most widely used mining explosive worldwide. ANFO and occasionally called ANFEX (fuel explosive) is a mixture of ammonium nitrate with petrol typically integrated with almost 6 % diesel as an oxidizing driver (ammonium nitrate blasting agent). It is economical and easy to manufacture, ship, and maintain.

Density, initiation sensitivity, water resistance, and velocity of detonation are the properties of interest used in the selection of fracture explosives. The velocity of detonation or VOD is the velocity at which an explosion spreads along a hole. In the functioning of explosives, VOD plays a key role of facilitating detonation. ANFO for example has a VOD of approximately 4 400 m/s under ideal conditions. This explosion wave is generated by an accessory known as a detonator. As shown in Figure 2.2, a shock tube is fitted through the detonator for safety and efficiency purposes. The shock tube also ensures good initiation, precise timing while the many detonators buried in each drill hole are typically set off by a single electric detonator (Hummel and McCann, 2011).

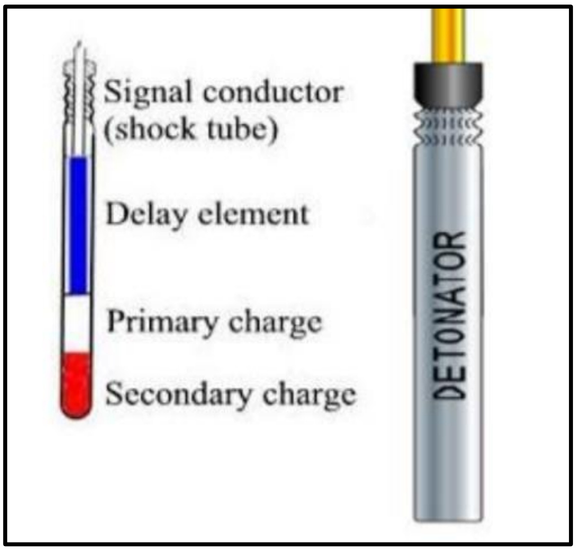

Figure 2.2 A typical detonator (Adapted from Pande et al., 2015)

Drilling and blasting can be argued to be the first stage of comminution in the mine value chain. This is because the primary focus of blasting in the past was to enable the excavation equipment to dig the blasted rock effectively while keeping the number of oversize chunks (or boulders) created at the lowest possible. Now, the impact of blasting on subsequent operations is receiving much attention in recent years. These operations (i.e. milling and crushing) are presented next.

2.2.3 Crushing and milling

Depending on the school of thought, the blasted rock fragmented are subjected to the first (or second) stage of comminution, that is, primary crushing.

From the 1920s to the 1950s, many comminution circuits were devised as a sequence of stages of crushing, succeeded by rod milling and finally by ball milling. During the 1960s, the usage of rod mills failed as ball mills of greater diameter for processing coarser feeds became available. The decade also saw the arrival of mills for Autogenous Grinding (AG) and Semi-Autogenous Grinding (SAG). Large-diameter AG and SAG mills often coupled with ball mills then became the standard comminution circuit in the early 1970s. Even though power consumption was generally higher, the simplicity of the SAG/Ball mill circuit or SAB circuit, the low number of components, and the small footprint made the complete economics of the circuit greater than that of the three-stage crushing configuration (Wills and Finch, 2015). Over the next several decades, SAG-based circuits opened the way to high-tonnage and low-grade operations typical of the base metal industry (Rose et al., 2015). The adoption of large tumbling mills extended in such a way that most Greenfield mining ventures or those expanding from the early 1980s to the early 2000s have nominated circuit configurations that include AG/SAG mills (Barratt and Sherman, 2002). It is in this light that three generic types of tumbling mills are presented in the subsequent paragraphs: ball mill, AG mill, and SAG mill.

Tumbling mills are the most widely used comminution technology in the mining industry. A tumbling mill is a drum that rotates about it longitudinal axis. The rotary motion lifts the load made of large particles and grinding media that then drops in a cascading and cataracting motion. The falling fraction impacts particles in the lower region of the load and breaks large solid particles into small-sized particles. Depending on the design of the tumbling mill, the type and volume of grinding media used (i.e. steel balls or rock lumps), the mill is known as ball mill, autogenous mill, or semi-autogenous mill.

Ball mills rely exclusively on grinding balls to effect comminution with bed of balls making up as much as 45 % of the internal volume of the cylindrical drum. SAG mills use 8 % to 21 % of the volume of the mill while AG mills only use the rock for grinding.

It should be stated that two factors have encouraged the departure from SAB circuits especially in hard-ore operations. The first is the pressing need to decrease energy consumption herded not only through economics, but also through civic attention to climate change. The second factor is the appearance of High-Pressure Grinding Rolls (HPGR). As their manufacturer developed roll-wear safety systems to deal with abrasive and hard ores, this comminution technology became attractive (Casteel, 2006). Furthermore, HPGR mills are more energy-efficient than conventional mills as they deliver greater unit throughput at higher reduction ratio (Valery and Jankovic, 2002).

The HPGR mill is a system built around a pair of counter-rotating and high compression milling rolls riding on a robust lip. In the frame is a single immovable roll while the other can drift on rails using pneumo-hydraulic springs. The feed is provided to the opening between the rolls and is crushed by the inter-particle breakage mechanism (see Figure 2.3).

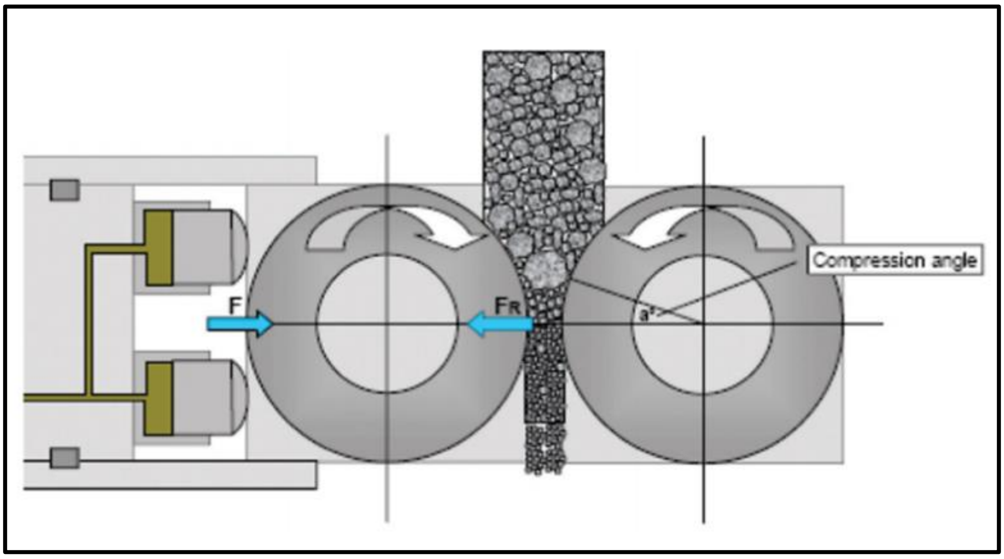

Figure 2.3 HPGR design principle (Morley, 2010)

The operating pressure varies usually from 50 to 150 bars but can exceed 180 bars. This pressure creates compression forces of up to 25 000 kN for the largest devices.

The main catalyst for the use of HPGR mill in hard-rock comminution is its energy efficiency compared to conventional mills and crushers. In addition to this, benchmarked against SAB circuits, HPGR-based circuits provide significant reductions in the grinding media consumption and associated costs (Morley, 2010; Saramak, 2011; Daniel and Morrell, 2004; von Michaelis, 2005).

The latest wave of technological development in comminution is what is known as stirred milling. Stirred milling technology was introduced in 1950s but has penetrated the mining industry only in the last few decades. There exist various types of stirred milling machines on the market covering applications in primary and secondary milling as well as regrinding. Stirred mills exhibit high energy efficiency especially in fine grinding; however, encouraging results are being reported also in the coarser grinding range (Valery and Jankovic, 2002).

The economics and design of comminution circuits are primarily influenced, by tumbling mills, i.e. SAG, AG and ball mills. This is because tumbling mills are energy-inefficient and energy-intensive. They account for up to 80 % of the total energy consumption of the processing plant (Abouzeid and Fuerstenau, 2009; Fuerstenau and Abouzeid, 2002). Efforts for gradual step-changes have given rise to the idea of combining HPGR and stirred mills in one flowsheet (Valery and Jankovic, 2002). This paved the way for future energy-conscious comminution circuits. An opportunity to evaluate the stirred/HPGR milling circuit and understand its potential benefits is being explored at the mill testing facility of the Norman B. Keevil Institute of Mining Engineering, University of British Columbia. Concurrently, a joint stirred and HPGR milling circuit is being investigated with the two machines operated outside their commonly accepted operating conditions. Irrespective of what the future holds, Drozdiak (2011) has been able to prove that an HPGR-stirred mill circuit is theoretically feasible. The researcher also showed that the circuit has promising advantages over both the standard crushing/ball milling circuit and the HPGR/ball mill circuit. Finally, the performance of the current SAG and Ball milling circuit closed with a Cone Crusher (or SABC circuit) at the Huckleberry Mine was compared to two alternative circuits. The idea was to see whether the new HPGR/stirred mill circuit arrangement could achieve energy savings. Great strides were hence made in the quest for more energy-efficient comminution circuits (Wang et al., 2013).

2.2.4 Comminution economics

In the USA, 39 % of the energy due to mining activities is used for processing operations, of which 75 % is accounted for by comminution. This figure is also likely to apply to most mining countries (Tromans, 2008). One of the reason for this is that comminution is energy-extensive and inherently inefficient (Austin, 1984; Fuerstenau and Abouzeid, 2002; Hukki, 1975). The state of affairs is pronounced in large hard-rock mines. Indeed, tremendous quantities of energy are expended on comminution operations alone. And while energy and costs associated with crushing is not negligible, milling in general is responsible for half of the figures.

There is substantial evidence that suggests that blasting can be used to incur major cost savings around comminution circuits (e.g. Eloranta, 1995; Paley and Kojovic, 2001). One may postulate that crushing and grinding performance can be influenced by the size distribution of blasted fragments and the internal softening of individual fragments. There may therefore be room for improvement when comminution circuits are aligned with blasting practices. A direct consequence of this would be greater mineral liberation and enhanced downstream recovery.

There have been reports on the research and implementation of drill-to-mill ventures. In the majority of these studies, there is evidence that drilling-and-blasting staff ought to be weary of the effects of blasting practices on the economics of comminution. To achieve the best cost of service, blasting engineers need to work closely with process engineers. This will ensure that the greatest possible savings in energy are incurred at the comminution stages (Tromans, 2008).

Other benefits of improved blasting-to-comminution include increased productivity in crushing and grinding, more undersized material that bypasses the crushing stages, reduced consumable wear in crushing, grinding, loading, and hauling, increased shovel production and less energy expenditure in loading, and ability to use lightweight truck boxes to haul more uniformly sized blasted fragments.

It can therefore be argued from the above that the configuration of comminution circuits and smarter blasting practices have the potential to improve the contribution of comminution to plant economics. Most importantly, even if not all energy savings in crushing and grinding are realized, substantial cost savings are still possible.

2.3 Open-pit mine planning

The pioneering work of Lerchs and Grossmann (1965) epitomises a precise and computationally tractable network-based technique for cracking the ultimate pit limit problem. The work was extended later by Hochbaum (2001) as well as Chandran and Hochbaum (2009) amongst others.

The key to the ultimate pit limit problem is to determine the economic enclosure of profitable blocks given constraints of pit slopes. Time is disregarded from the point of view of the production scheduling and the expected revenue from the extraction of a block.

Fundamentals underpinning open-pit mine planning are reviewed in this section. They are to lay the foundation of the Lerchs-Grossman algorithm covered in the next section.

2.3.1 Geometrical definitions of the pit

The different mineral deposits mined by open-pit techniques today vary considerably in scale, shape, orientation, and depth. There are several geometry-based planning and design considerations essential to different topographies. Figure 2.4 is a graphical rendering of the earth's surface before and after the construction of the open pit mine.

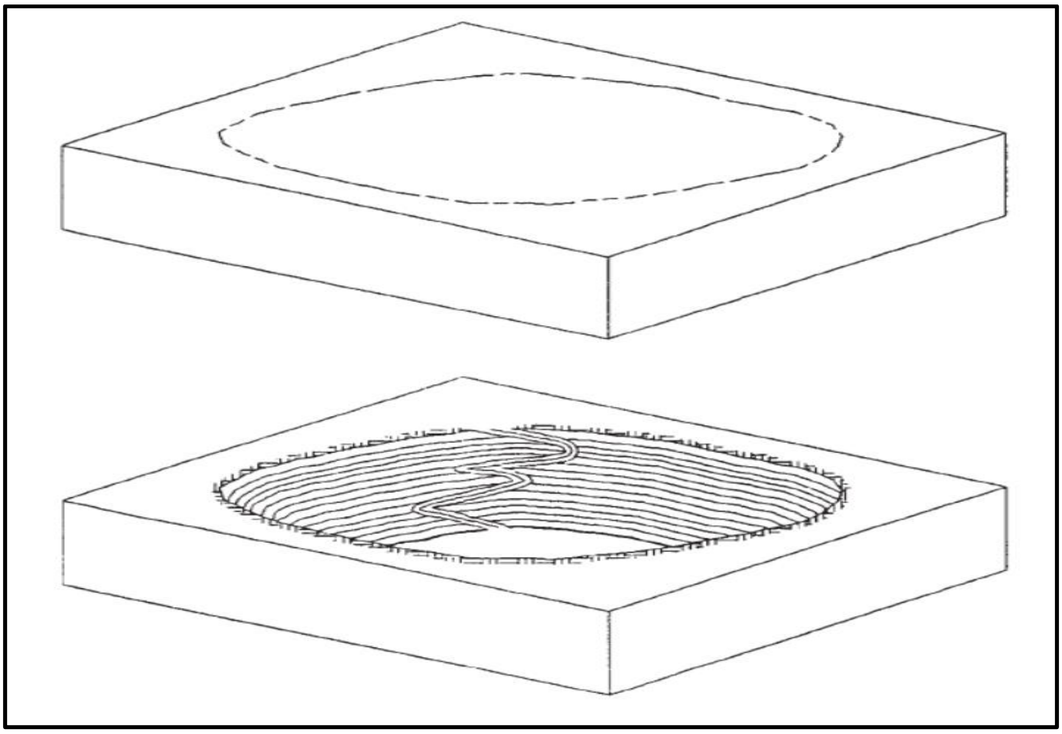

Figure 2.4 Simple open-pit mine geometry (Adapted from Hustrulid et al., 2013)

It can be seen from Figure 2.4 that the ore body is mined from the top down in a sequence of horizontal levels of the same width or benches. Mining begins with the top bench, and after a sufficient floor area has been uncovered the next layer of mining begins. The method continues until the height of the bottom bench is achieved, and the final pit outline is reached. A path or ramp must be built to reach various benches as shown in the bottom drawing of Figure 2.4. The width and slope of the ramp depend on the type of lauding and hauling equipment used.

Stable pit slopes must be planned and maintained throughout the construction and operation of the pit. The angle of the pit slope is a critical geometric parameter with an important techno-economic influence. Open-pit mining is highly mechanized. Each mining machine has a geometry linked to its physical size and space needed for efficient operation (Hustrulid et al., 2013). The following sections address the geometrical aspects involved in the planning and construction of open pit mines.

2.3.1.1 Bench geometry

The key extraction feature in open pit mining is the bench. A bench is a narrow strip of land cut to the side of an open pit mine. These step-like zones are built along the walls of the open pit mine for access and mining (Jackson, 1997). Each bench has a higher and lower surface separated by a distance H equal to the height of the bench. Visible sub-vertical surfaces are referred to as bench faces. They are defined by the angle of the toe, the crest, and the angle of the face. The latter is the average angle of the face with the horizontal angle (refer to Figure 2.5).

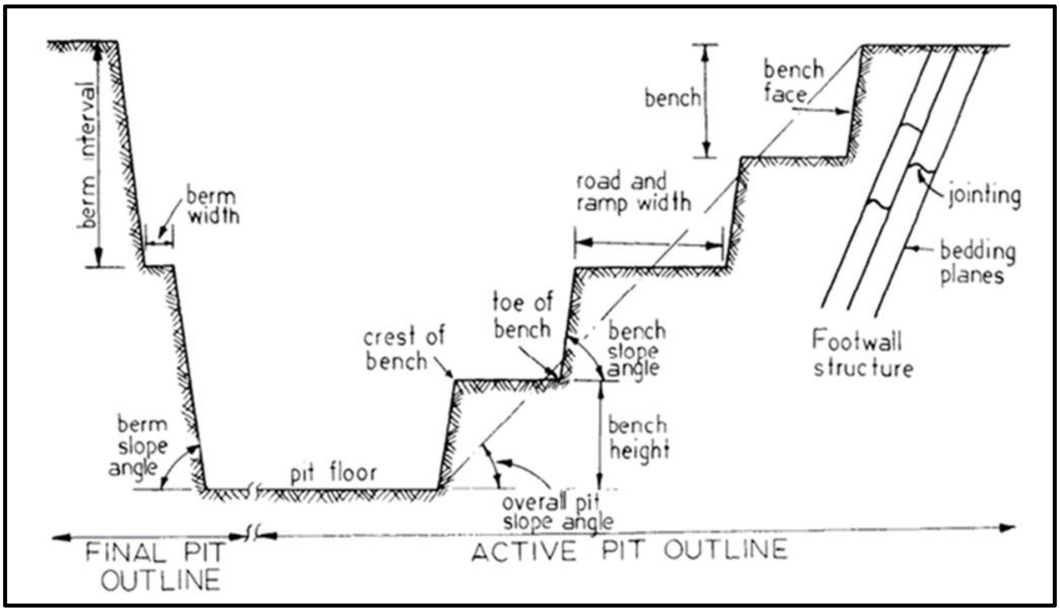

Figure 2.5 Benches showing various components (Jackson, 1997)

The angle of the bench face can vary considerably with the characteristics of the rock, the orientation of the face, and the blasting practices. In most hard rock pits, it ranges from around 55° to 80°. The typical initial design estimate could be 65°. However, this value must be used with caution as the angle of the bench face can have an immense impact on the overall angle of the pit slope (Hustrulid et al., 2013). Usually, bench faces are mined as sharply as possible, at a 45° maximum angle. However, there is a certain amount of back-break due to several reasons. This is known as the distance between the actual bench crest and the designed crest.

The visible bench on the lower surface is called the bench floor. The width of the bench is the distance between the crest and the toe measured along the upper surface. The bank width is the horizontal projection of the face of the bench. There are a variety of types of benches. A working bench is one in the process of being mined. The width to be separated from the operating bench is called a break.

Below are steps to be considered when defining the geometry of the bench:

- Properties of the deposit determines a particular geometrical method and production approach. These include grade distribution, total tonnes, and value of the deposit.
- The production approach that will lead to regular ore-waste production rates, selective mining and blending criteria, and workplaces.
- Production requirements guiding the specifications of the equipment to be used and associated operation geometry.
- Consequences regarding stripping ratios, processing and mining costs including the assessment of slope stability features and the suitable geometry to support the operation.

2.3.1.2 Access to the ore

In order to gain initial access to the orebody, the covering vegetation and overburden must be removed. A vertical digging face must be established in the orebody before the main production can commence. Furthermore, a ramp must be created to permit truck and loader access. A drop cut is used to create the vertical breaking face and the ramp access concurrently. Because vertical blast holes are being fired without a vertical free face, the blast conditions are highly constrained. Rock movement is primarily vertically upwards with a very limited sideways motion. To establish adequate digging conditions, the blast holes are typically very closely spaced.

Geometric considerations regarding ore access indicate the following:

- There can be substantial volumes associated with the main ramp system.
- The setting of the ramp varies with time.
- In the higher levels of the pit, the ramp is underlain by waste while the lower ranges are underlain by mineral.
- Cash flow considerations are significantly affected by ramp timing.
- The stripping ratio, the percent extraction, and the overall extraction are greatly affected by the haul road geometry (road width and road grade).

2.3.2 Orebody modelling

There exist various definitions of orebodies. They may be classified into three distinct components:

1. The physical geometry of the geological units hosting the orebody.
2. The attributes of all material to be mined characterised in terms of assays and geo-mechanical properties of all material to be mined.
3. The economic value model of the mineral deposit.

The aim of orebody modelling is to determine the value of the mineral deposit and the potential of making a return by analysing the values of grade, tonnage, and other designated geological entities (SME, 2005).

Although the description can be brief and compact, determining the value of an underground resource requires a great deal of work. The final orebody model should provide information on the physical, technological (reliability of projected mineral beneficiation and mine production operations), and economic properties of the resource (Kennedy, 1990).

Orebody modelling is aimed at reproducing the reality of the mineral deposit as closely as possible using available data. The challenge with the exercise is that underground deposits are intangible entities whose forms, compositions of quality, and quantity are not always well understood. The goal of geological investigations and explorations is to classify all these unknowns. Topographic and lithological data are collected at the beginning of the process and a database is created. Variations in degree, thickness and depth, overload structure, ore volume, form and extension, footwall, and hanging wall properties are calculated by the various geometric approaches used in this database. Both numerical calculations and visual aids help to bring out an orebody model (Singer and Menzie, 2010).

Orebody modelling techniques can be categorized broadly into three forms: mathematical, geostatistical, and traditional. Most of these techniques are used as software packages in one form or another providing high speed computation through the vast number of blocks making up the model (Erarslan, 2012).

The most tangible data to describe the location, form, quality, and quantity of the ore is the centre of the drill hole. Indeed, that is where one gets as much information regarding the ore in that particular drill hole. Global Positioning System (GPS) data is also used in drawing topographic maps, surfaces and subterranean maps such as thickness and grade contours. When topographic coordinates are combined with stratigraphic information, a three-dimensional data set is then obtained. In the end, after extensive mathematical processing, a three-dimensional orebody model can be generated (Hustrulid and Kuchta, 2006). The model describes the physical orebody as well as the distribution of various attributes key to the future economic valuation of the deposit. The design of the mine and the production schedule can now follow from the physical structure, and the quality composition of the model of the ore deposit (Hartman, 1992).

Three-dimensional components x, y, z (easting, northing, altitude/elevation) comprise survey data and allow surface modelling. Drill hole data containing depth and layer data helps to explain how the geological structure is in the three dimensional space (Torries, 1998). Drill holes often contain the ore grade or calorific value data. A three-dimensional orebody model provides a geological understanding of stratigraphical layers (Nieuwland, 2003).

Several methods have been proposed in recent decades to address challenges relating to geological modelling (Agoston, 2005). This is because geological modelling attempts to estimate unknown values with limited data at hand. In Figure 2.6, a universal work plan and flowchart for computer-aided orebody modelling are shown.

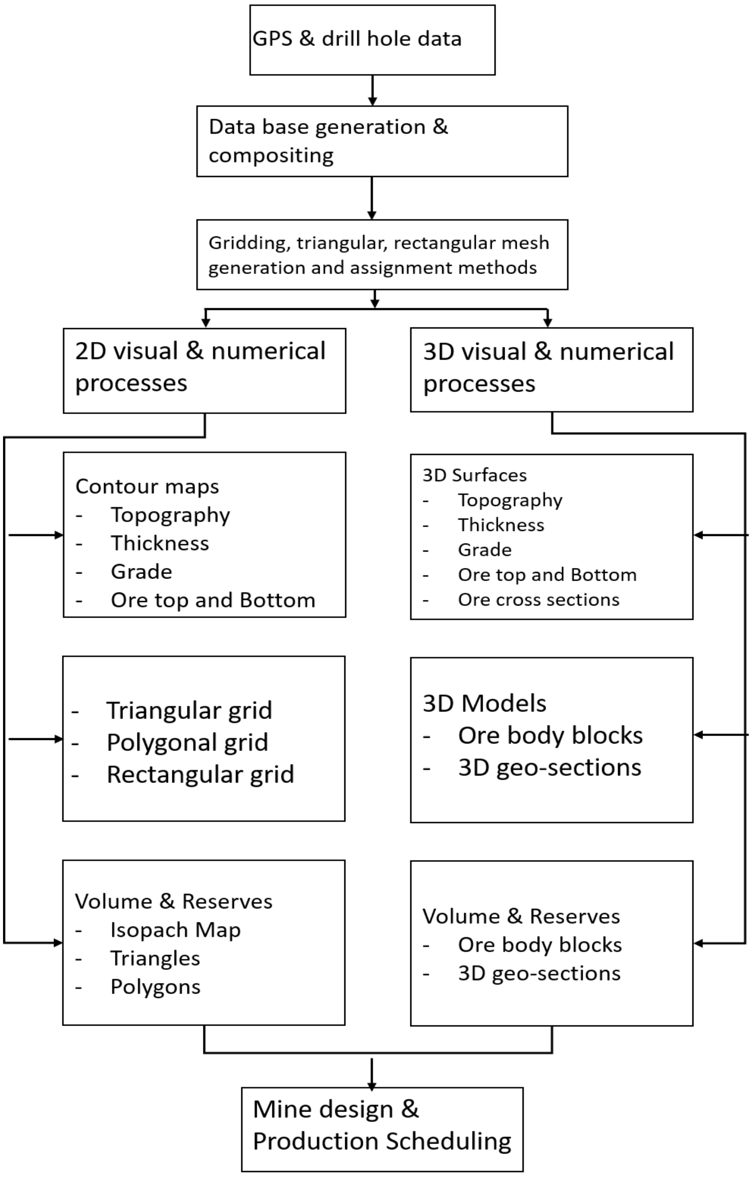

Figure 2.6 Computer-aided orebody modelling work flowchart

Once the ore deposit is modelled visually and numerically following the workflow in Figure 2.6, the design and production schedule of the mine can follow. At this point, engineering economics and optimization principles are contemplated. Here, optimization and simulation techniques such as graph theory, linear and target programming, dynamic programming, mixed integer programming, moving cones, genetic algorithm, and network analysis are considered (Erarslan and Celebi, 2001).

Graph theory applied to mine planning is introduced in the next section in what is known as the Lerchs-Grossman algorithm. It is the most widely used optimization and simulation technique.

2.4 Lerchs-Grossman algorithm

In surface mining projects, successful open pit design and output scheduling (OPOPS) are crucial. The consequences of pit design, mine scheduling, and associated forecasts have a significant influence on the management of cash flow. Modern OPOPS is based on the well-known Lerchs-Grossman algorithm implemented as the nested Lerchs-Grossman algorithm (Lerchs and Grossman, 1965; Whittle, 1988 & 1997; Whittle and Rozman, 1991).

Given a set of geological, technical, and mining considerations, the Lerchs-Grossman algorithm offers an optimal scenario of how an orebody should be best mined economically. This optimal environment is susceptible to the uncertainties associated with the optimization process input and the uncertainties of the following (Refsgaard et al., 2007):

1. Orebody model and associated inconsistency of in-situ grade and distribution of material form.
2. Technical specifications such as excavation capacity and slope angles.
3. Capital and operational expenditures and the commodity prices.

Various problems of uncertainty and risk involved in OPOPS have been raised in the past years (e.g. Ravenscroft, 1992; Onur and Dowd, 1993;

Halatchev and Moustakerov, 1994; Dowd, 1994; Rossi and Van Brunt, 1997). Orebody models and their geological attributes are the main source of uncertainty and risk. In most cases, sensitivity to grade variability or metal values is analysed with global changes that cannot account for the critical regional block grade variability. Be that as it may, the Lerchs-Grossman algorithm still find use in mining practice. It is succinctly presented in the subsequent sections.

2.4.1 Pit optimization – Definition

For a given set of economic parameters, pit optimization is used to define the most profitable pit shells (or nested pit shells). What is economically mineable from a given deposit is determined by the final pit limits. It determines which blocks are to be mined and which ones are to be left unmined. The economic block model is developed first from the geological grade model to classify the blocks to be mined. This is achieved by assuming, at present economic parameters, output and process costs and commodity prices. The economic parameters include the prices of metals, the recovery of processes, and operating costs.

Each positive block is tested by using the economic block values to decide if its value will compensate for the overburden removal. The estimation is based on the breakeven calculation that tests whether the undiscounted income earned from a given ore block will pay for the undiscounted cost of waste block mining. This research is carried out using computer programs that implement either the “cone mining” technique or the Lerchs-Grossman algorithm (Lerchs and Grossmann 1964; Zhao and Kim, 1992).

The Lerchs-Grossmann (LG) algorithm ensures the optimality of determining pit limits that maximize the undiscounted profit whereas the routine of cone mining is heuristic and can yield sub-optimal results. The determination as to what should be mined within the final pit limits is time-dependent and the correct outcome must take into account the understanding of the time when a specific block will be mined and how long the waste needs to be extracted.

Achireko (1998) identified some of the factors that can lead to poor predictive ability of the pit optimization process. For example, the optimization process can produce large pits with a long mine life. Fluctuations in metal prices will inevitably have unpredictable bearing on the optimised solution. In another instance, pit optimization can be done at the beginning of the feasibility project when detailed operation costs are not available yet. As a result of this, rough estimates are made for future costing with unintended future consequences. Operating costs can also change with time when the initial optimization is based on current information. Another interesting example is that smaller pits are made up of smaller tasks that may have different running costs than the optimization assumed. Similarly, larger pits might have different operating costs and throughput rates than anticipated in the optimization. Finally, pit optimization is generally constructed on the assumption of fixed metal price, fixed cut-off grade for the life of the mine (The cut-off grade is the grade that is used during scheduling to distinguish between ore and waste; Dagdelen, 1992), and operating costs adjusted with the depth of the pit. However, the metallurgical response of the plant changes with the ore type being mined. And despite all the above, the pit optimization is a modelled solution to the complex problem of the ultimate pit limit problem. It should be taken in this light as it provides great insight into the mining problem.

2.4.2 Optimization of the net present value

A geologic block model is the starting point of any open-pit mine planning. From the model, a determination needs to be made as to whether or not a given block should be mined, when this should be done, and whether the block should be sent to the processing plant or the tailings dam.

The combined response to the above questions determines the annual progression of the pit surface, and the annual cash flows over the life of the mine to be generated. The solution to the scheduling problem depends on the attributes associated with individual blocks. In addition to determining the cash flows for that year, the decision about which blocks should be mined in a given year and how they should be processed (i.e. leaching, comminution, or waste) also influences future annual schedules. The scrutiny of pit limits that maximize Net Present Value (NPV) requires that in defining the mining sequence of blocks during the life of the mine, the time value of the money should be taken into account. This is because the NPV of the mine is not be maximized by the pit limit that maximize the undiscounted income for a given project. The intermediate pits leading to the ultimate pit limit are calculated as part of the planning and scheduling process to see how the pit surface changes over time. The method used in current software packages to generate nested pits is to gradually vary costs, cut-off grades, or product prices from a low value to a high value. For example, by varying the product price from a low value to a high value, many pits can be produced by increasing the size and decreasing the average value per ton of ore in the pit (Dagdelen, 1992 & 2001).

Note that the cut-off grade that maximizes the NPV of the cash flows is a function of limitations on mining, milling, and refining capacity as well as the distribution of grades within the deposit. An algorithm for the calculation of cut-off grades that optimize the NPV of a project subject to limitations on mine, mill, and refinery capability was proposed by Lane (1964). During the initial years of the deposit, the cut-off grade method that results in a higher NPV for a given project is used. This gradually decreases to a break-even cut-off grade as the deposit grows depending on the distribution of the deposit by grade. Different computer packages have been produced using the Lane algorithm (Lane, 1988; Dagdelen 1992; Whittle, 1999). Their implementation to determine the optimal cut-off grade strategy has led to significant improvements in the NPV of mining projects.

2.4.3 Limitations of the Lerchs and Grossmann algorithm

The greatest limitation of the LG algorithm reside in the difficulty of numerically implementing the method without simplifying assumptions, the complexity in incorporating variable pit slopes, and the extended computing times required to converge to a feasible solution.

2.5 Effects of operational costs on mine optimization

Comminution alone is responsible for 30 – 70 % of the total operational costs of typical mining operations (Radziszewski, 2013; Nadolski et al., 2014). This has consequently drawn most of the initiatives designed to reduce processing costs and energy consumption in the mining industry. Various factors contribute to the poor efficiency of comminution operations which in turn affect the solution to the mine optimization problem.

Worldwide economic crises and uncertainties in the mining sector have forced engineers and researchers to look for ways of compressing overall mining costs. One can consider that the costs associated with blasting and drilling operations contribute about 15 % of the overall costs of mining hard-rock deposits (Božić, 1998; Gokhal, 2010; Palangio et al., 2005). By increasing the volume of explosives per ton of rock to be blasted, it is possible to drastically cut down on the processing costs. This increases the blasting and drilling costs but with a benefit on the comminution side that outweighs the initial costs of using more explosives (Božić, 1998; Eloranta, 1995; Napier-Munn, 2015; Paley and Kojovic, 2001; Workman and Eloranta, 2003). High-intensity blasting, however, needs to be done to maintain good fragmentation while ensuring safe wall control (Gokhal, 2010; Olofsson, 1988).

It is important to point out that cost savings have seldom been considered through optimizing drill and blast geometric parameters. Indeed, the cost of fragmenting a piece of in situ rock is influenced by several factors. These include but are not limited to blast geometric parameters and patterns, density and type of explosives, and the geological nature of the rock formation. But the latest research show that the type and amount of explosives used per ton of rock have the greatest potential for cost-effective mining and comminution (Mulenga and Mwashi, 2018; Napier-Munn, 2015; Paley and Kojovic, 2001; Workman and Eloranta, 2003).

Finally, as can be calculated in terms of environmental and fragmentation issues, the real cost of damaged blasting can be many times the cost of the blast itself. Examination of various activities indicates that while mine blasts normally fragment rock to be treated by the mining process, optimum fragmentation is possible to increase efficiency and reduce the cost of all downstream processes (Božić, 1998).

2.6 Concluding summary

From the literature review conducted so far, it is evident that open pit mines have been striving to find a way to deal with the economics of their operation. They have identified comminution circuits as the bottleneck that can lead to reduced operational costs when tuned with energy-intensive blasting. Other comminution technologies such as high-pressure grinding roller and stirred mills also have a potential to result in cost-effective circuits compared to classical semi-autogenous and ball milling circuits. All these cost saving strategies may positively affect the economics of the mining project. However, this is yet to be investigated in detail. It is against this background that the present dissertation attempts to establish the extent to which reduced comminution costs may impact open pit mine planning. The Net Present Value is used as the guiding criterion while optimisation is also done on the undiscounted profits as would normally do mine planners. In the end, the simulation work is expected to shed some light on the effects of comminution practices on open pit mine planning.

Chapter 3 Simulation tool and data collection programme

This chapter describes the techniques employed to collect the data for analysis in line with the objectives set out for the research dissertation. The methodology adopted in this study revolves around computer simulation work (Goddard and Melville, 2001). This was used to investigate the effects of variable comminution costs on mine planning.

3.1 Introduction

Finding the best long-term production scheduling is important in open pit mining projects. Ordinarily, it should result in maximized NPV under several operational constraints. The issue is that solving the open pit optimization problem does not always lead to the optimal solution (Osanloo et al., 2008). This impasse can simply be described in these terms: First, until the block values are known, the pit outline with the highest value cannot be determined. Second, until the mining sequence is determined, the block values are not known. And last, until the pit outline is available, the mining sequence cannot be determined.

Several simulation packages have been put forward for the optimization of open pit mining. In general, most packages are proprietary with varying licensing options while few are freely available. It is in the second group that SimSched Direct Block Scheduler (DBS) falls under and is used for this research. The dataset descriptive of the orebody model is also publically available through the MineLib database. This database contains block models for use in studying open pit mining problems (Espinoza et al., 2013).

In this research work, a small academic dataset named *Newman1* is extracted from the MineLib database. The *Newman1* block model is fed into SimSched DBS and set up to be solved using the optimization problem paradigm. SimSched DBS then iteratively computes the pit optimization problem under various combinations of mining and processing costs.

Two optimization approaches are considered in this work. The first relies on the stepwise approach based on the LG algorithm (Caccetta and Hill, 2003). A series of nested pits are produced this way followed by the application of a scheduling algorithm to subsets of the block. The second strategy is based on direct block preparation. Here, the idea is to solve linear optimization problems for a given extraction period under slope and mining constraints.

3.2 SimSched, the Direct Block Scheduler

SimSched DBS is a plugin of the Stanford Geostatistical Modelling Software (SGeMS). Currently in a beta stage, SimSched DBS is computer software hosted by MiningMath and used for pit optimisation (Chaves et al., 2020). Upon assigning extraction periods and destinations to blocks, SimSched DBS can perform NPV maximization for mine scheduling purposes. Here, the NPV maximization problem is considered under constraints of milling capacity and time value of money amongst others.

As a freeware, SimSched DBS enables one to do pit optimization, pushback design, and scheduling simultaneously in a single process. Moreover, the flexibility of the mixed-integer programming algorithms deployed as part of the freeware allows for the addition of blending restrictions and constraints relating to excavation hours, metal production, and average haul distance.

Another key aspect of SimSched DBS is that it incorporates surfaces to generate scheduling plans as geometric parameters. In other words, it is possible to set minimum values not only for the bottom mining widths but also for a range of vertical rates of advance. Importing surfaces for a custom geometric restriction of the pit is also possible. SimSched DBS allows the use of stockpiles, multiple destination routes, variable slope angle, and block-by-block recovery at the processing plant. SimSched DBS can finally determine the global optimal mine scheduling in a single step.

3.2.1 Definition of the mine project

Current practices in open pit mine planning normally challenge the planners and managers to make strategic decisions at different stages of the project. The ultimate goal of these decisions is to achieve the best long-term production scheduling. This section presents the basics of the software used in this study for mine planning. The NPV is used as an index to show the optimized extraction of the commodity.

Simulations are carried out using the SimSched DBS software. Once SimSched DBS has opened, the SGeMS window developed by Advanced Resources and Risk Technology (ar2tech) pops up as shown in Figure 3.1. SGeMS is an open-source programming language that is used for solving problems involving spatially dependent variables.

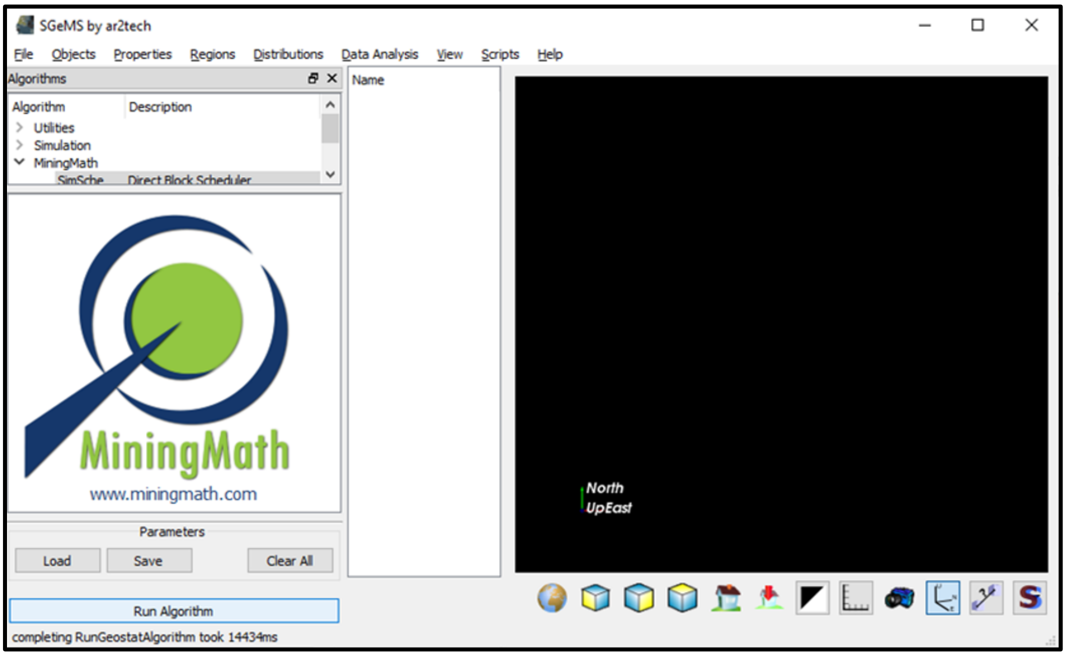

Figure 3.1 SGeMS by ar2tech interface

SimSched DBS tools are now available under MiningMath as depicted in Figure 3.1. After running the SimSched DBS, a new window pops up as shown in Figure 3.2. It has a green panel on the left-hand side of the screen where several operations can be invoked. For example, a model in a CSV format can be imported to start a new job. Next to the green panel is a pane displaying previously saved jobs; one can also save or remove jobs done in the pane. Other operations include opening a scenario that one had worked on before, exporting the model into a single file for easy transfer, and license details about the software version. An extensive library and instruction manual on the use of SimSched is also available with illustrative examples.

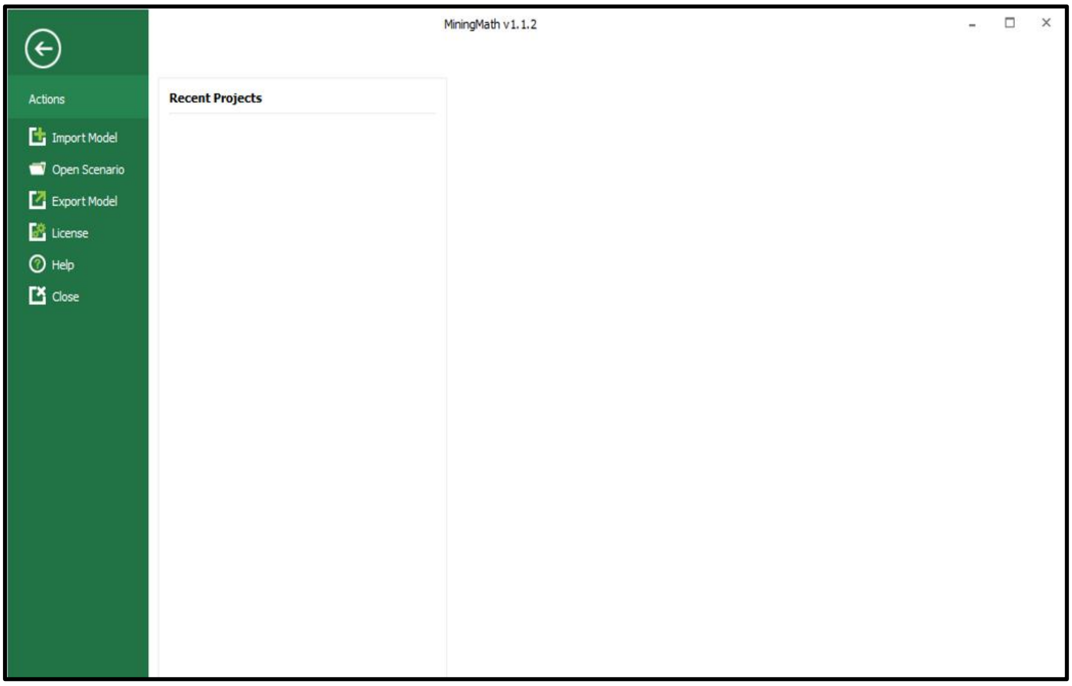

Figure 3.2 Main window for SimSched DBS under MiningMath

At this stage, the data that is in CSV format can be imported into SimSched DBS to start the process of mine scheduling.

3.2.2 Block model of the mine project

It should be recalled in Figure 3.2 that the *Import Model* option under the green panel is to be used to import block models of the project. Only a block model that is in line with the minimum requirements and specifications of the software discussed in Figure 3.2 can be successfully imported. By clicking on *Import Model*, a new window similar to Figure 3.3 appears listing all block models available for importing.

The SimSched DBS software has the flexibility of importing different block models of one or different projects at the same time. Figure 3.3 shows that the *file name* input field is shown in red, indicating a mandatory field while the *Newman1* block model highlighted in blue is the one being imported.

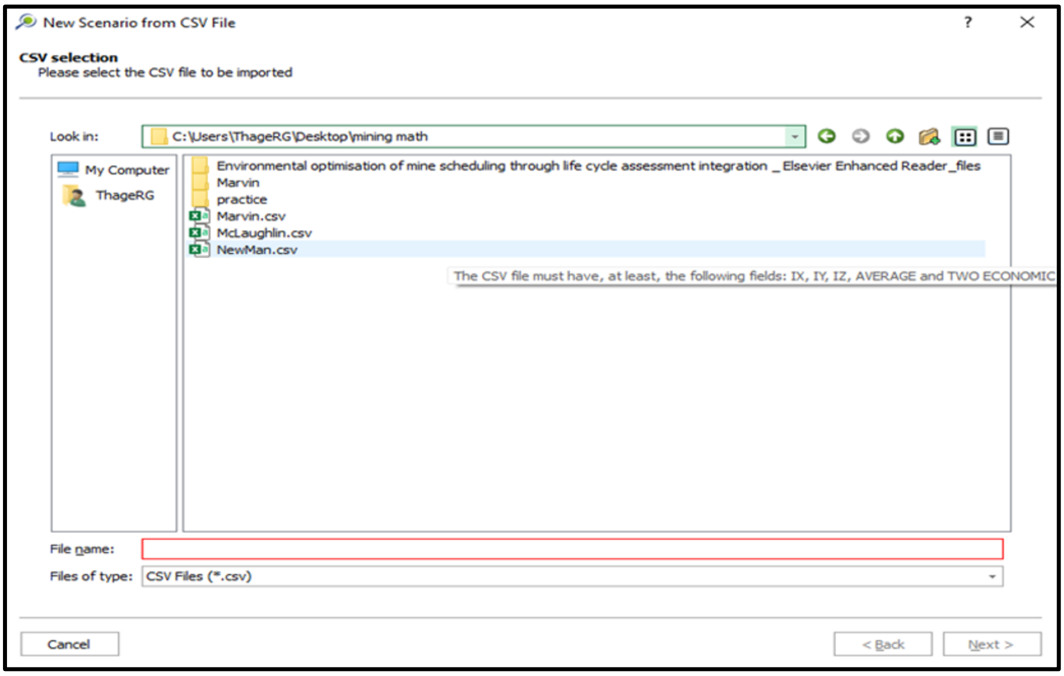

Figure 3.3 Importing a CSV model

After completing all steps required in Figure 3.3, the name of the first simulation scenario of the *Newman1* model can be entered. Upon clicking the *Next* button, the window in Figure 3.4 appears with all the statistics pertaining to the imported block model. It can now be seen that the characteristics of the *Newman1* model downloaded from the MineLib database are summarised in terms of the following:

- Minimum and maximum values of X, Y, and Z indices
- Minimum and maximum grades of copper (Cu)
- Minimum and maximum grades of gold (Au)
- Minimum and maximum values of ore density (t/m³)
- The minimum and maximum value of the economic value process
- The minimum and maximum value of economic value waste

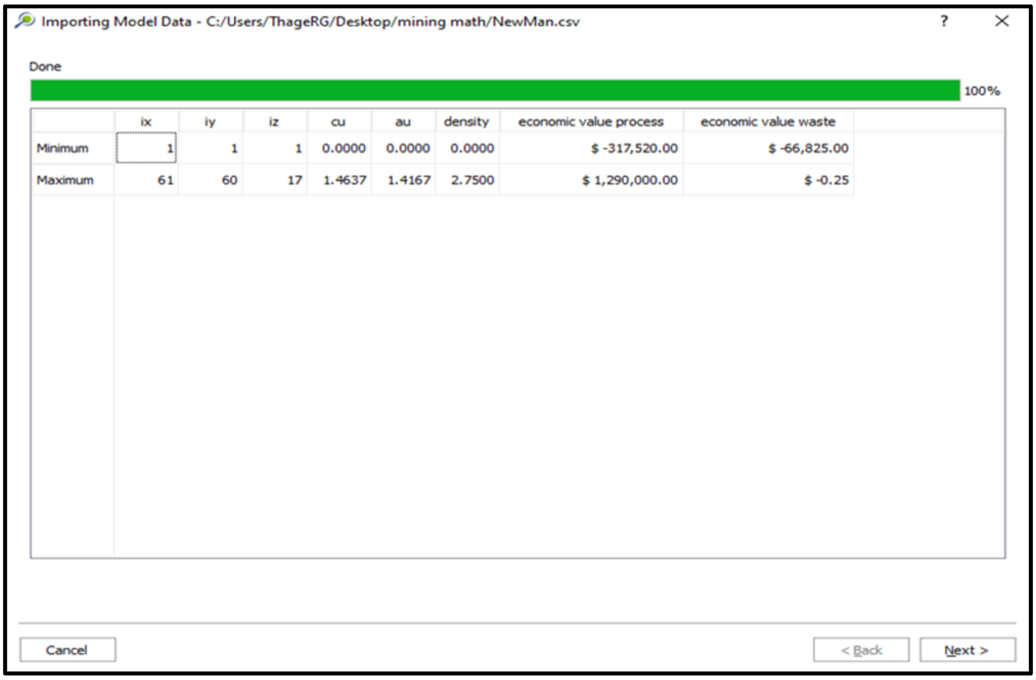

Figure 3.4 Data validation for NewMan1

Each field is linked to a financial value (Economic Value Waste/Process). Each block value should be accounted for by the linked field as a function of its terminus, grades, recovery, mining costs, transport, treatment, and sale price. The subsequent formulas demonstrate exactly how the value of a block is determined:

$$Block\ Tonnes = Block\ Volume * Block\ Density \tag{3.1}$$

$$Tonnes\ Cu = Block\ Tonnes * Grade\ Cu/100 \tag{3.2}$$

$$Mass\ \mathrm{Co} = Block\ Tonnes * Grade\ Co \tag{3.3}$$

$$\boldsymbol{Economic\ Value\ Process} = (Tonnes\ Cu * Recov\ Cu * (Selling\ Price\ Cu - Selling\ Cost\ Cu)) + (Mass\ \mathrm{Co} * Recov\ \mathrm{Co} * (Selling\ Price\ \mathrm{Co} - Selling\ Cost\ \mathrm{Co})) - (Block\ Tonnes * (Processing\ Cost + Mining\ Cost)) \tag{3.4}$$

$$\boldsymbol{Economic\ Value\ Waste} = -\ Block\ Tonnes * Mining\ Cost \tag{3.5}$$

After data validation, pressing the *Next* button prompts the form in Figure 3.5 showing different field types. The form in Figure 3.5 displays a preview of the imported block model and two header rows. The top header row has various field types, and the bottom one shows the headers of the CSV file.

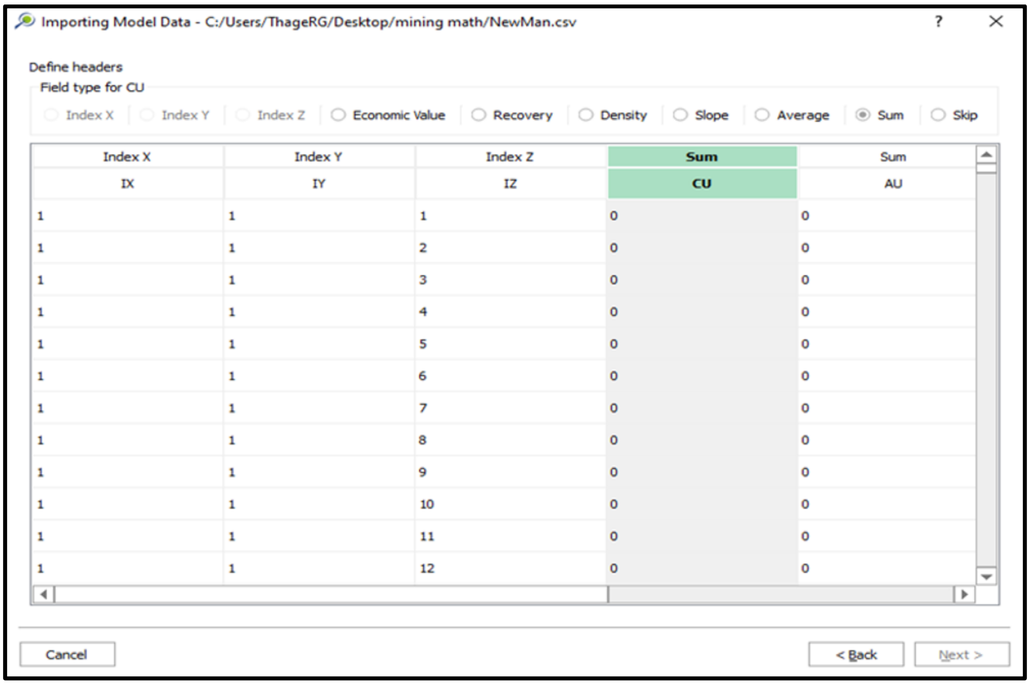

Figure 3.5 A form showing various field types

Now that the CSV file is imported, the additional data relating to the dimensions and grade units of the imported blocks can be defined. Grade units are imported from the same initial CSV file as illustrated in Figure 3.6 with the grades in copper and gold of individual blocks. Note here that copper grade is in % while gold grade is in ppm (equivalent to g/t).

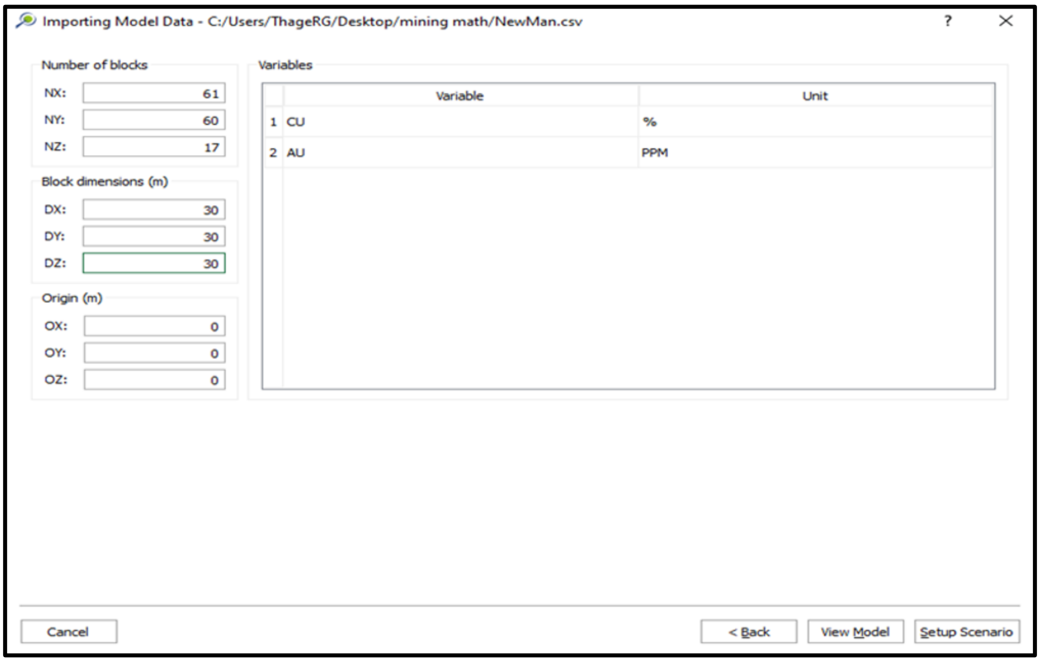

Figure 3.6 Grade units and block dimensions

After filling in the required fields with the number of blocks, block dimensions, and the origin (see left-hand side of Figure 3.6), the options *View Model* is enabled. The required fields are derived by analysing the raw data. Before proceeding with the direct block scheduling (DBS), the imported model can be viewed by clicking on *View Model*. When this is done, MiningMath closes while the preview is made available on the SGeMS interface as shown in Figure 3.7.

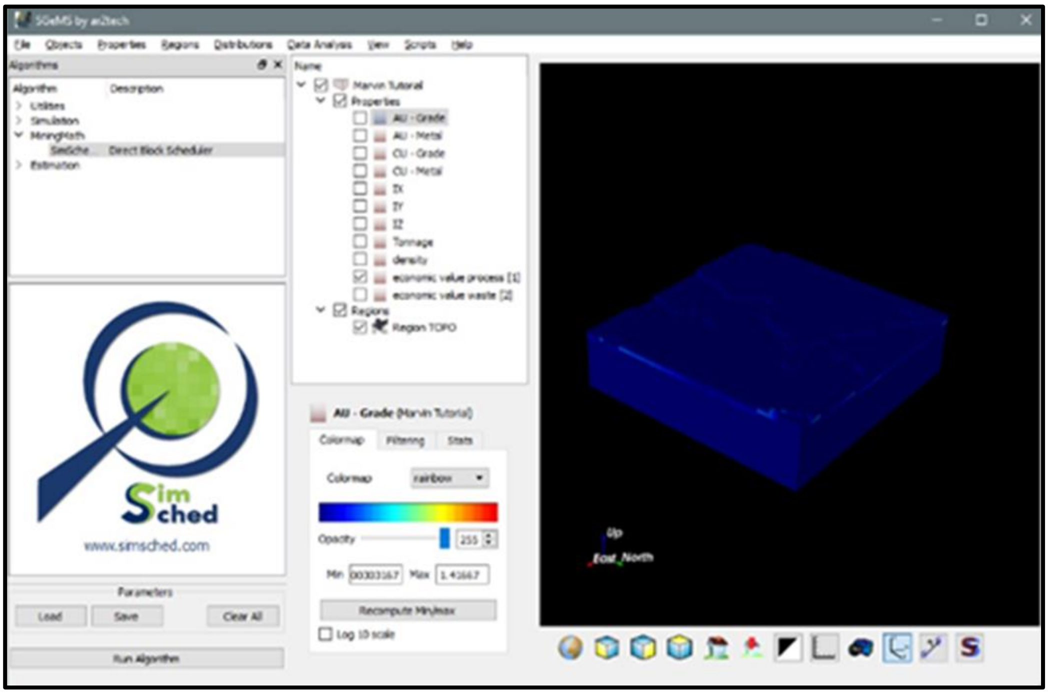

Figure 3.7 Typical SGeMS display of a block model.

3.2.3 Optimization and planning of open pit mines in SimSched DBS

For any mining enterprise, the preparation of output for long-term planning is an essential activity. Mining blocks must be prepared for extraction over several years and individually allocated a destination. This is done to optimize the net present value of the open pit project subject to capacity and organizational limitations. Optimization is then carried out by generating nested pits using the LG algorithm. Here, DBS is considered with the understanding that individual blocks are selected for extraction and destinations are allocated for certain periods (Morales et al., 2015).

Operational parameters associated with the mine scheduling problem are non-linear in nature. This non-linearity results in cumbersome computations and increased processing time for the solution to convergence (Osanloo et al., 2008). In this regard, SimSched DBS deploys a Linear Programming (LP) engine that linearizes parameters thereby accelerating the search. Mathematically, the objective function used for the purpose is written as:

$$max \sum_{b \in B} \sum_{t \in T} \sum_{d \in D} \vartheta_{btd} y_{bt} \tag{3.6}$$

Where $b \varepsilon B$: a setting of all blocks b

$t \varepsilon T$: a set of periods inside the horizon

$d \varepsilon D$: a set of all destinations d

ϑ: discounted value linked with the final destination of a block b in period t

y: 1 if block b is mined in period t, 0 otherwise.

Depending on the application, the objective function in Equation (3.6) is subjected to constraints of slope, process recoveries, and time amongst others. Generically, these constraints can be represented as follows:

$$\sum_{b \in B} c_b y_{bt} \leq \bar{C} \tag{3.7}$$

Where c and $\bar{C}$ are the consumption of resource associated with the extraction of block and minimum (maximum) resource bound in any period (tons) respectively.

Figure 3.8 shows the process taken by SimSched DBS to iteratively solve the optimization problem. The first step in the process is linearization. The linearization process is evoked to transform the operational parameters into linear constraints. At this point, SimSched DBS executes the model and checks its feasibility. Should the solution not be feasible, Equations (3.6) and (3.7) are revisited by applying simplifying assumptions to the problem. On the other hand, should the solution be feasible, SimSched DBS will run the optimization procedure until the NPV is maximized; then, the solution is stored for later use.

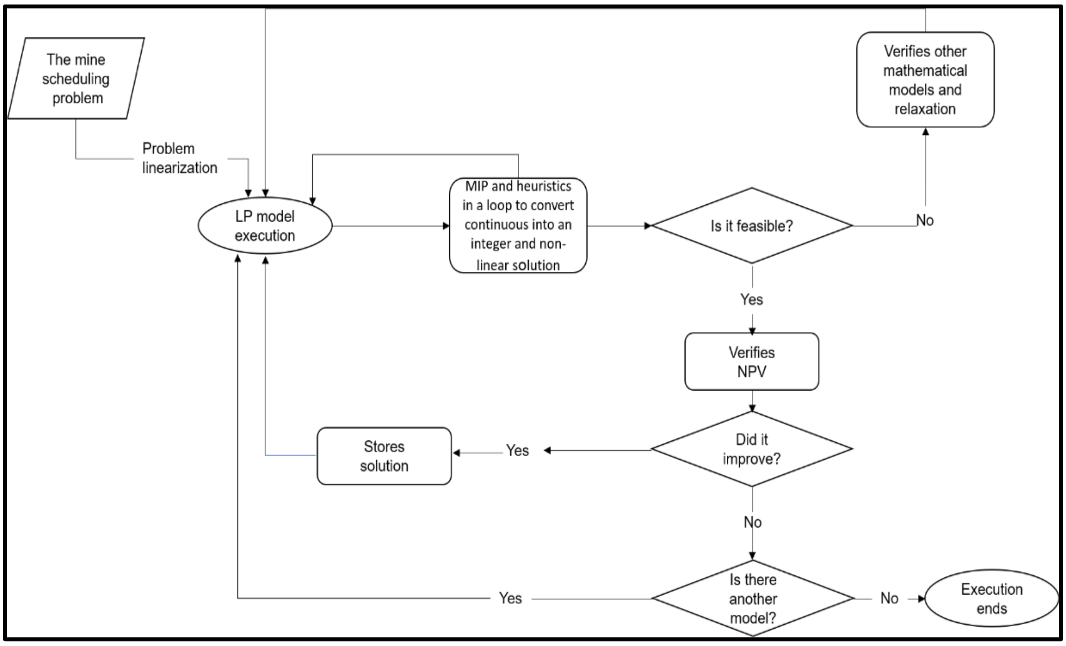

Figure 3.8 Mine scheduling problem

3.2.4 Optimization report

SimSched DBS can generate a capability report directly into Microsoft® Excel® and an optimized block and surface pit in SGeMS for the *Newman1* block model. The reports will clearly show the number of periods for the optimized pit as well as the output metal and input grade for Cu and Au respectively at various periods. The number of tons for each period of both minerals is also calculated and rendered as part of the optimization report.

3.3 Definition of the ore body model

Several comminution techniques and circuit designs have been used in industry with varying degrees of success (Tromans, 2008). However, the economic implications of the choice of a particular comminution strategy on mine planning are still yet to be explored. This is because the primary objective of mine planning is to select the maximum profitable excavation sequence throughout the lifespan of the mine irrespective of the costs incurred.

From the point of view of this study, the *Newman1* block model is considered for simulation whereby comminution and processing costs are varied while corresponding mine plans are generated. The *Newman1* block model was selected because of the scale of analysis required of the research work. The block model consists of 1 060 standard blocks. Attributes such as rock type, tonnage, ore grade, and economic values are available as part of the dataset. Once the user defines the mining and processing costs, the destination of each block can be determined (i.e. waste dump or processing plant). This is where the concept of cut-off grade discussed next is required.

3.3.1 Cut-off grade

Grade is a factor used to define the value of a rock block (Hall, 2014). The word “rock” refers to the total material that is mined until it is divided into ore and waste fractions. The term “ore” is defined as the mineralized material that is extracted for treatment. The concept of cut-off grade is used to differentiate between ore and waste (Lane, 1988; Rendu, 2008). The point of departure in the categorization of material in an open pit mine is the following mathematical expression (Hall, 2012):

Rock = ore (treated or stockpiled) + waste = total material moved (3.8)

Note that the components of Equation (3.8) are the main drivers of mining and processing costs accounted for in mine planning (Hall, 2014). Defining the cut-off grade therefore has a bearing on the mining and processing costs as well as on the economics of the mine plan. A high cut-off grade amplifies the NPV but shortens the life of the mine.

3.3.2 Types of cut-off grade

There are two groups of cut-off grades in open-pit mining: internal and external cut-off grade (Baird and Satchwell, 2001). The difference between the two is that the internal cut-off grade is applied after pit streamlining for blocks that are in the ideal pit to characterize mineral reserves. This type of cut-off grade classifies material as ore or waste. It finds application in the SimSched DBS software. The external cut-off grade, on the other hand, is applied during pit streamlining to recognize blocks that create income and characterize a definitive pit. Blocks beneath the external cut-off grade are treated as waste. When the pit optimization is done, and a plan is created, post-pit enhancement is completed to enhance the life of the mine extraction technique of the asset inside the planned pit.

3.3.3 Break-even cut-off grade

This is the grade whose income covers all cash-dependent expenses including fixed and variable expenses, corporate and mining assessments, and other allocated capital uses (Pasieka and Sotirow, 1985). Dagdelen and Kawahata (2007) describe the monetary break-even cut-off grade as the grade which can be used to distinguish the metal from the squander. It is managed by comparing the incentive at the plant to the incentive at the dump. This break-even cut-off grade is generally used to determine the last pit limits. The same is true of the initial investment point in the life of a mine where the working expenses are equal to the estimate of the item sold.

3.3.4 Minimum (marginal) cut-off grade

The marginal cut-off grade is what meets just the variable working costs (prohibiting assigned managerial and other fixed working and capital expenses). It is utilized to decide the lowest evaluation that could be mined without misfortunes if there is no other mineralized material accessible for the predefined ability to create a positive net income (Pasieka and Sotirow, 1985). This grade is utilized to isolate the metal from squandering inside the ideal pit limit.

3.3.5 Cut-off grades for polymetallic deposits

Polymetallic deposits are mineral existences that contain more than one metal of financial worth (Rendu, 2008). Estimating the cut-off grade in this case should account for the contribution of every metal to the income.

The cut-off grade of a multi-mineral ore deposit can be completed using parametric cut-off grades. This intrinsically means that for a deposit like *Newman1* with copper being the fundamental mineral and gold the by-product, the grade can be stated in terms of copper equivalent as follows:

$$\textbf{Cu Equivalent Grade} = \text{Grade}Cu + \frac{GradeAu * (RecoveryAu * (PriceAu - Selling\ CostAu) - Element\ Processing\ CostAu)}{\text{Recovery}Cu * (\text{Price}Cu - \text{Selling Cost}Cu) - \text{Element Processing Cost}Cu} \tag{3.9}$$

From the estimate in Equation (3.9), a single parametric cut-off grade can now be applied to copper and gold simultaneously. This cut-off grade is used both to assess the final pit and to decide whether to process or discard a certain tonnage of material.

3.4 Design and programme of simulation work

This section explains the approach that was employed for the simulation work. Geotechnical parameters, assumptions for pit optimization, and simulation scenarios considered amongst others are presented. The information was used to set up SimSched DBS so that the NPV of the *Newman1* model could be maximized.

3.4.1 Definition of geotechnical parameters of the pit

The final slopes of an open pit mine are usually excavated to the steepest possible angle of about 45° – 50°. This is to reduce the volume of waste rock that must be extracted before the ore is retrieved. However, open pit mining is a complex, risky and capital-intensive operation that may extend over many years (Lerchs and Grossmann, 1964).

The potentially hazardous nature of the operation needs to be considered when defining a safe and economic pit design. The main challenge resides in the variability of the soil conditions and excavation methods in use on site. This makes it difficult to come up with a single solution for the geotechnical design and operation of any mine to the point that detailed site-specific enquiry are resorted to instead. To stay within the scope of this research, key assumptions are made around the pit wall design and the life of mine.

Before mining begins, an appropriate geometry of the excavation design on which the overall mine plan is based should be created. The slopes of the mine are then cut to the steepest and safest angle so as to reduce the volume of waste rock to be excavated when an ore is recovered (Hoek and Bray, 1981). While economics generally guide the choice of the slope angle, the need for wide benches on which mining equipment can move freely impose a limit on how steep one can go (Steward and Kennedy, 1971). The stability of the individual bench is controlled by the local geological conditions, the overall shape of the field, groundwater conditions and the excavation technique used. These controlling factors vary for different mining situations (Hoek and Bray, 1981).

For the purpose of this research study and considering the fact that blocks forming the *Newman1* model are cubical, it was decided to set the slope angle at the value of 45°. The sequence for pit wall design can now be illustrated in Figures 3.9 – 3.11 using the floating cone method (i.e. LG method) applied in two dimensions (Lerchs and Grossman, 1964).

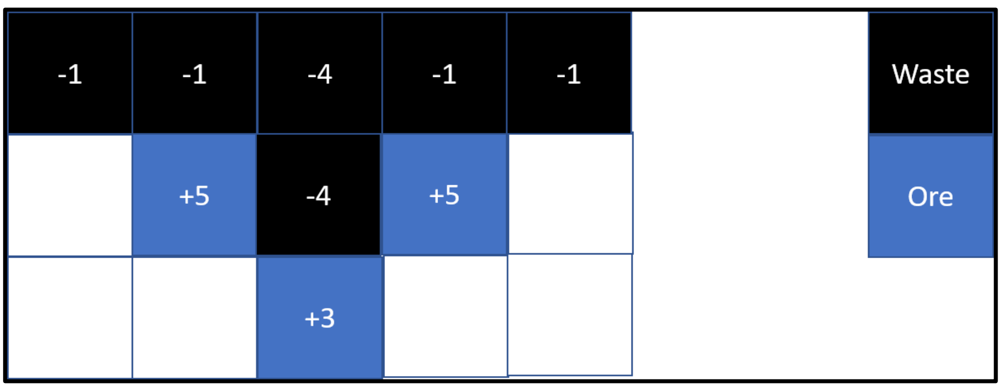

Figure 3.9 Example of a block model

The cone is floated from left to right along the top row of blocks in the segment. Where there is a positive square it is eliminated/mined. Seeing that the top row has no positive block, the negative blocks are then be removed to get access to the second row as shown in Figure 3.10.

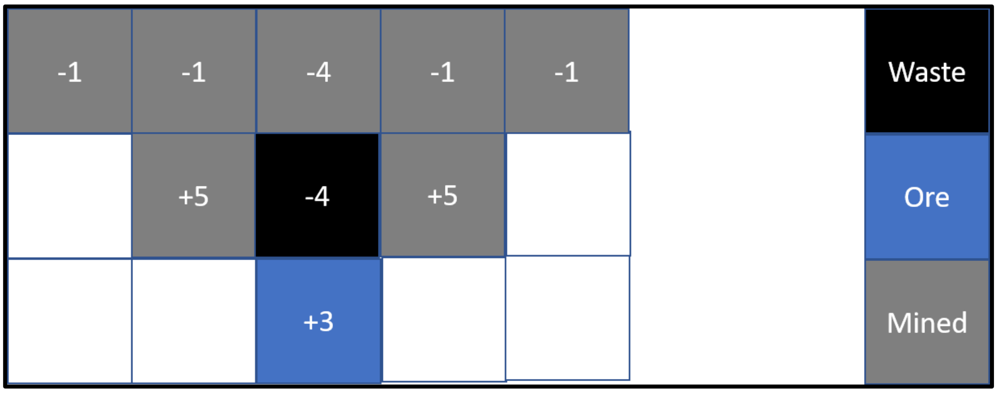

Figure 3.10 Mined blocks

Figure 3.10 shows the mined blocks from the first row to the second. To get the positive block, one starts from the left and searches for the positive block. If the total number of blocks falling inside the cone is positive, the blocks are mined. Note that the floating cone has a slope of 45° which implies that access to a block in the second row requires the removal of 3 blocks in the first row. In a three-dimensional system, the concepts can be extended to the removal of five blocks above the targeted block below. These geotechnical constraints are handled by SimSched DBS with ease.

From Figure 3.10, the floating cone process is followed until the bottom row and no more blocks can be removed; this is shown in Figure 3.11.

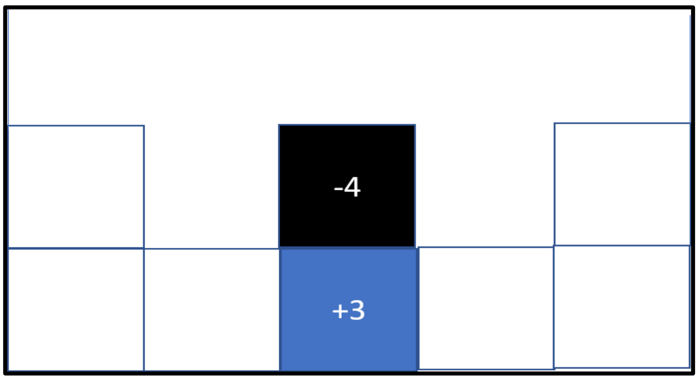

Figure 3.11 Final pit

At this point, the last positive block cannot be mined which leads to the final pit design. The profitability of the mined area can be found by adding the values of the blocks have been removed. The values of the blocks are determined as follows:

Block Value for both the Ore

$$BV = (P - s) * g_B * y - c - m \quad (3.10)$$

Block Value for both waste

$$BV = -m \quad (3.11)$$

Where P = Price; s = Sales cost; c = Processing cost; y = Recovery; m = Mining cost; g_B = Block grade; and BV = Block Value.

From the final pit design, the overall stripping ratio can also be determined as the number ratio of positive blocks to negative blocks.

Finally, the value of a systematic approach to the planning and design of pits using sound geotechnical engineering methods cannot be over-emphasized. Indeed, open pit mines need to be operated in an integrated manner, safely and economically. Pit construction should be done at a minimum unit cost and under acceptable social and legal constraints (DME, 1999). This is to ensure that the operation is maintained for the longest time at a profit. Known as the life of mine, the time duration of the operation is mostly influenced by the mining rates, the production processes, the production dump as well as the size, shape, and orientation of the excavation. The effects of mining and processing costs on the life of mine are reported upon later in Chapter 4.

3.4.2 Assumptions for pit optimization

Open-pit mining can be argued to be superior to underground mining in terms of ore recovery, production power, grade control, and dilution losses. With that in mind, the following heuristic assumptions were made:

- Dimensions of the blocks are 30 x 30 m with a bench slope of 45°
- Density is 2.75 t/m³ which is fairly used in this field.
- Discount rate 10 %.
- Recovery for Cu and Au process are 0.88 and 0.60 respectively.
- Process 1 is 30 000 000 tons and Dump 1 is 50 000 000 tons with a total of 80 000 000 tons each period.
- All other limitations and constraints are inherently taken into account when executing the algorithm for pit limit optimization.

3.4.3 Mining and processing costs

In order to explore the influence of processing costs and specifically comminution costs on the anticipated net profit value of the ore body, the following scenarios were considered based on existing literature (Božić, 1998; Gokhal, 2010; Palangio et al., 2005):

- Mining costs are allowed to fluctuate from 70 % to 140 % of the baseline mining costs so that the NPV is evaluated.
- Similarly, processing costs are varied from 70 % to 140 % while the strategy on cut-off grade is appraised.

SimSched DBS is used to simulate the proportional influence of mining costs as well as processing costs on the production plan of the typical open pit mine built around the *Newman1* deposit. The simulation scenarios are optimised in terms of the NPV of the mining projects generated.

3.4.4 Simulations scenarios considered

As the deposit selected for the purpose of this study, the *Newman1* block model has 1 060 blocks, 3 922 rules of precedence, and 6 time periods.

When loading the block model information into SimSched DBS, one can view the 3D rendering of the model as shown in Figure 3.12.

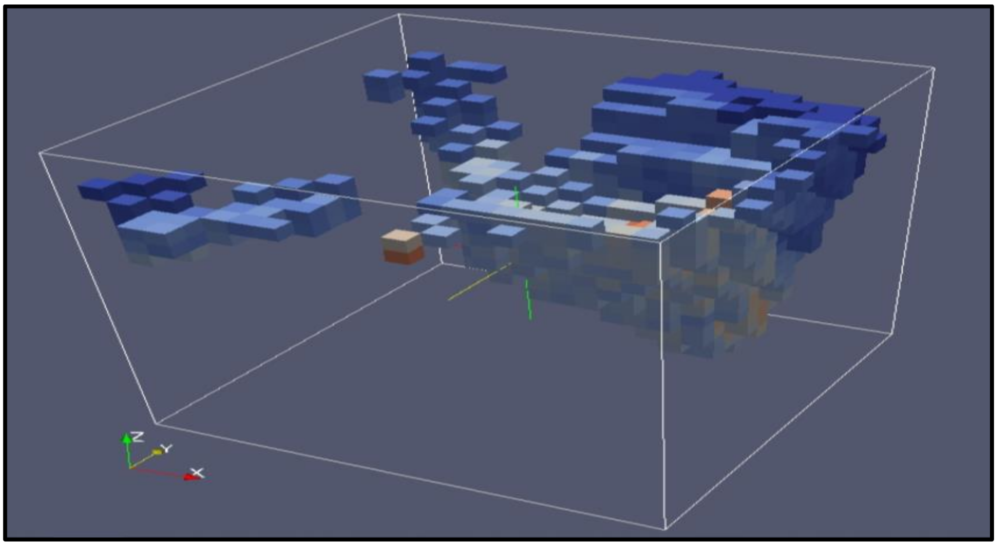

Figure 3.12 Block Model of the Newman1 ore body extracted from MineLib

Login parameters that SimSched DBS requires as input data are a valued block model that considers at least two economic destinations for process and disassembles, using Equations (3.4) and (3.5).

Calculations are performed for each block considering assumptions on input data as well as on mining and processing costs presented in Sections 3.3 and 3.4.3. In addition to this, the economic and geometric parameters that remain fixed are listed below. They were included in setting up the simulation models:

- A fixed cost of mining is considered 1.5 USD/t.
- A rehandling cost of 0.8 USD/t.
- The discount rate is considered 10 % annual.
- A minimum mined width of 50 m and bottom width of 100 m.
- A maximum vertical rate of advance of 150 m.
- Slope angles, metallurgical recovery, as well as the laws (Au / ppm) are coded in each block.

SimSched DBS was used to obtain the production plan of various scenarios for later analysis. For this research study, a matrix was made with different periods to get the optimal production rate. Mining and processing costs are systematically varied from 70 % to 140 % while the NPV was estimated in each case. Below are two examples of ultimate pits produced under two different scenarios. They are interpreted in detail later in Chapter 5.

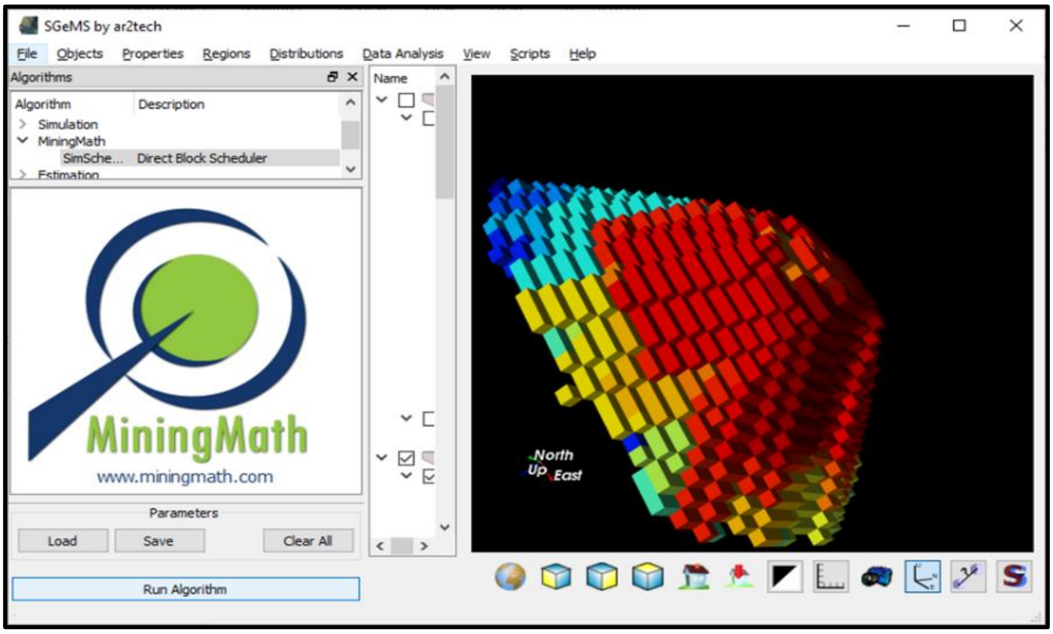

Figure 3.13 Block model of fixed data and assumptions made

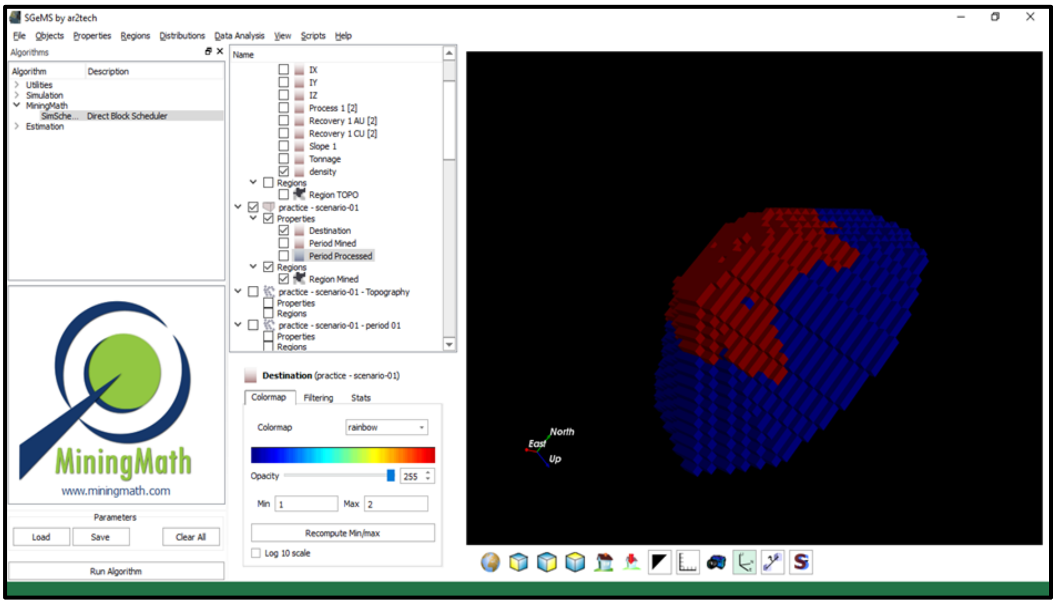

Figure 3.14 Block model of assumptions made with variation of data

3.5 Limitations of the work and challenges encountered

This research work is limited to the simulation of the *Newman1* model. The Direct Block Scheduling engine of the SimSched freeware was used for the purpose. The engine relied on the Linear Programming engine that works only on linearized versions of all mining and processing parameters of the optimization model. This slightly erodes the value of the final solution of the pit optimization problem.

Notwithstanding the above, the chapter described the techniques employed to collect the data for analysis in line with the objectives set out for the research dissertation. SimSched DBS as the simulation tool considered in this study was described in detail while all assumptions made were explained. The next chapter presents the results of the simulation work; then, the relevant findings are made sense of in Chapter 5.

Chapter 4 Strategic mine planning under comminution constraints

4.1 Introduction

In this chapter, output data collected from the simulation work covered in Chapter 3 are analysed. The key objective is to evaluate the influence of mining costs as well as processing costs on the production plan of a typical open pit mine. The motivation for the scenario analysis is to whether there is a benefit in implementing cost savings on comminution operations from the point of view of strategic mine planning. Subsequent to this, the implications of cost savings on the cut-off grade are also explored. All the above is tested the *Newman1* block model, a copper-gold deposit hosted on the MineLib electronic database.

4.2 Determination of the break-even cut-off grade

The break-even cut-off grade was previously defined in Section 3.3.3. Based on the supporting Equation (3.9), it is possible to obtain an estimate of the break-even cut-off grade applicable to the *Newman1* block model. However, it is imperative to clarify the costs used in the calculations of the break-even cut-off (Hall, 2014). This is summarized in Table 4.1.

Table 4.1 Assumptions on prices and costs used in estimating the break-even cut-off grade

Description	Symbol	Units	Value
Dilution	D	%	
Processing cost	H	USD/t milled	43.87
Unit time costs	F	USD/t milled	37.53
Copper recovery	Y	%	95
Cu metal exchange	Pcu	USD/t metal	9 000
Selling cost per tonne	Kcu	USD/t metal sold	495.88

Element processing cost – Copper	Hcu	USD/t Copper contained in feed	671.35

The break-even cut-off grade, COG, applicable to both copper and gold present in the *Newman1* deposit can be expressed as follows:

$$COG = \frac{d*(h+f)}{(y*(pcu-kcu)-hcu)} \tag{4.1}$$

Therefore, $COG = \frac{43.87+37.53}{(\,95*(9000-495.88)-671.35} = 1\ \%$

The break-even cut-off grade in this production process is 1 % which includes the contribution of gold to the revenue. Anything beneath the cut-off grade is regarded as waste. Figure 4.1 illustrates the discarded material below the cut-off grade.

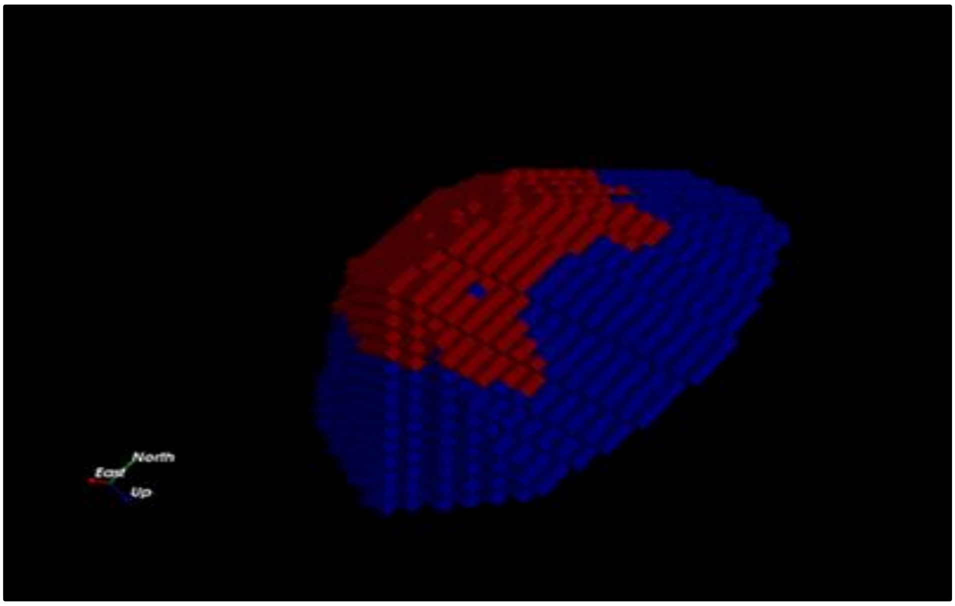

Figure 4.1 Correlation between mine production and cut-off grade (van Daalen, 2012)

Based on the data generated with the SimSched DBS software, it was estimated that 24 494,758 tons of the material is of grade below 1.0 %. This represents 41% of the mineralized material going to the dump site.

4.3 NPV-based optimization

Having defined the cut-off grade and tonnage of valuable material to be mined in Section 4.2, the next course of action was to determine the best mining sequence that will incur maximum NPV.

The simulation data resulting from the pit optimization is listed in Table 4.2 with period represents the full year of exploitation from the commencement of the project.

Grades associated with copper are noted to be higher relative to the gold ones. This is mostly because of the way that NPV considers the influence of the two metals in the optimization cycle (Mugwagwa, 2017).

Table 4.2 Profile of the production schedule over the life of mine

PERIOD	TONNAGE_kt	Output_Metal_CU_kt	Input_Grade_CU_%	Output_Metal_AU_kg	Input_Grade_AU_ppm
1	58147,32	173,56	0,298	49502,96	0,851
2	57788,8	222,05	0,384	39767,12	0,688
3	59519,44	338,64	0,569	40498,36	0,68
4	59781,33	301,68	0,505	32731,96	0,548
5	59829,88	424,94	0,71	35917,75	0,6
6	14567,42	131,2	0,901	7830,27	0,538
7	59998,93	395,97	0,66	29409,84	0,49
8	22476,99	148,48	0,661	10981,51	0,489
9	40555,45	266,61	0,657	21375,17	0,527
10	1706,95	3,54	0,208	559,85	0,328
11	21137,09	118,68	0,561	10333,57	0,489
12	14380,49	84,98	0,591	8809,82	0,613
13	20264,3	113,79	0,562	9800,95	0,484
14	13753,34	80,38	0,584	6251,32	0,455
15	336,4	0,96	0,286	79,17	0,235
16	52030,55	337,27	0,648	25102,36	0,482
17	1871,48	6,33	0,338	455,81	0,244
18	58520,53	329,65	0,563	22229,19	0,38
19	59041,8	331,7	0,562	22785,13	0,386
20	43237,95	216,76	0,501	15815,75	0,366
21	14563,98	64,22	0,441	6451,9	0,443
AVERAGE	**34929,07**	**194,83**	**0,53**	**18889,99**	**0,49**

The average copper grade is low in the initial two years in light of the limited access to enough uncovered high-grade blocks. This indeed can be attributed to the need for the initial stripping of the overburden material. Once, the first layer of mineralized ore is uncovered, copper grade ascends between Year 3 and Year 10. Afterwards, a drop in average grade is recorded; then, an erratic fluctuation around an average grade of 0.45 % is observed until the exhaustion of the deposit.

Another note is that the stripping ratio is high when exploitation starts because of quickened squander stripping to uncover high value ore domains. This results in reduced cash outflow that steadily expands the NPV of the project thereafter.

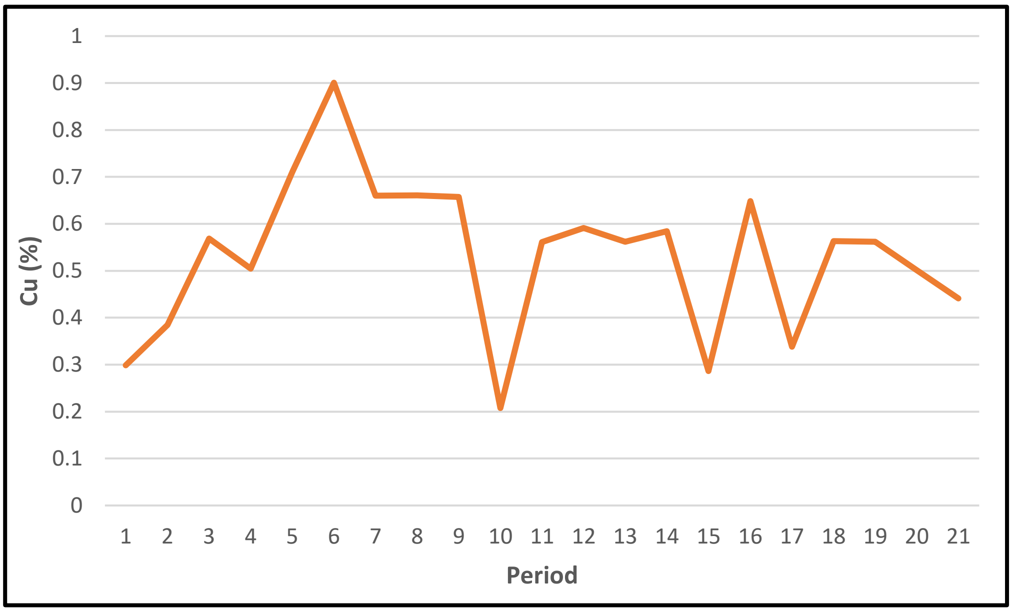

Figure 4.2 Variation of copper grades over the life of the mine

A look at the gold grade in Figure 4.3 shows a slowly declining trend throughout the life of a mine. This may be attributed to the fact that gold is more of a by-product of copper than anything. Its inclusion is therefore a source of additional revenue.

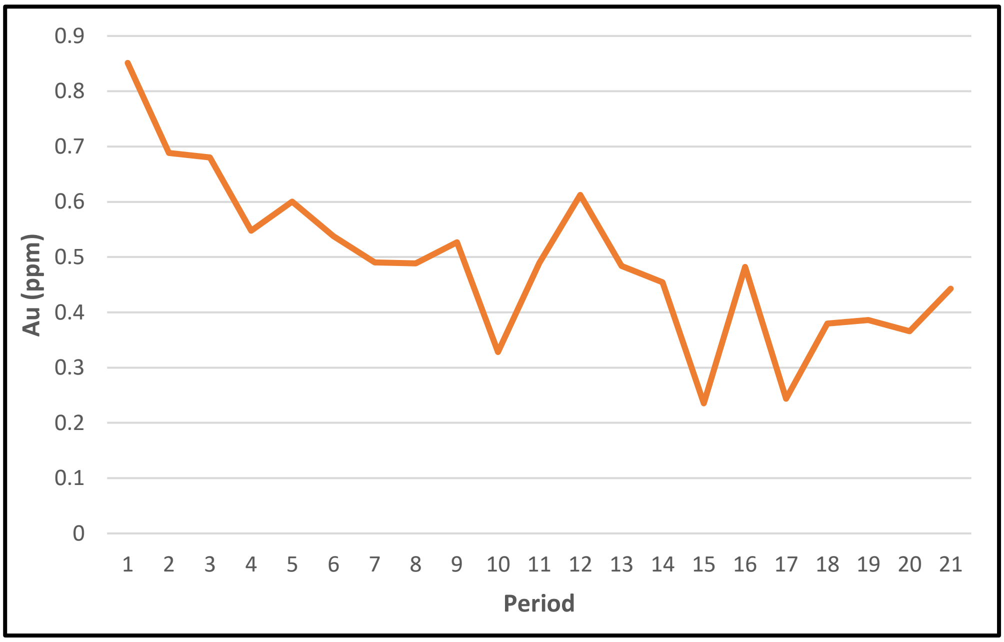

Figure 4.3 Variation of gold grades over the life of the mine

The declining trend of gold grade observed in Figure 4.3 is also indicative of the intrinsic nature of the mineralization of the *Newman1* deposit. By analysing individual blocks, it seems that high-gold-grade blocks tend to contain less or no copper. However, the goal is to improve copper yield since copper-bearing rock is regarded as the primary source of revenue. So, any attempt to increase the grade of gold in the mix has a weakening impact on the copper grade. Scheduling would therefore target copper-bearing blocks regardless of whether gold is present even at a lower grade or not. Metal yield follows similar trends and distributions as grade over the life of mine as shown in Figure 4.4.

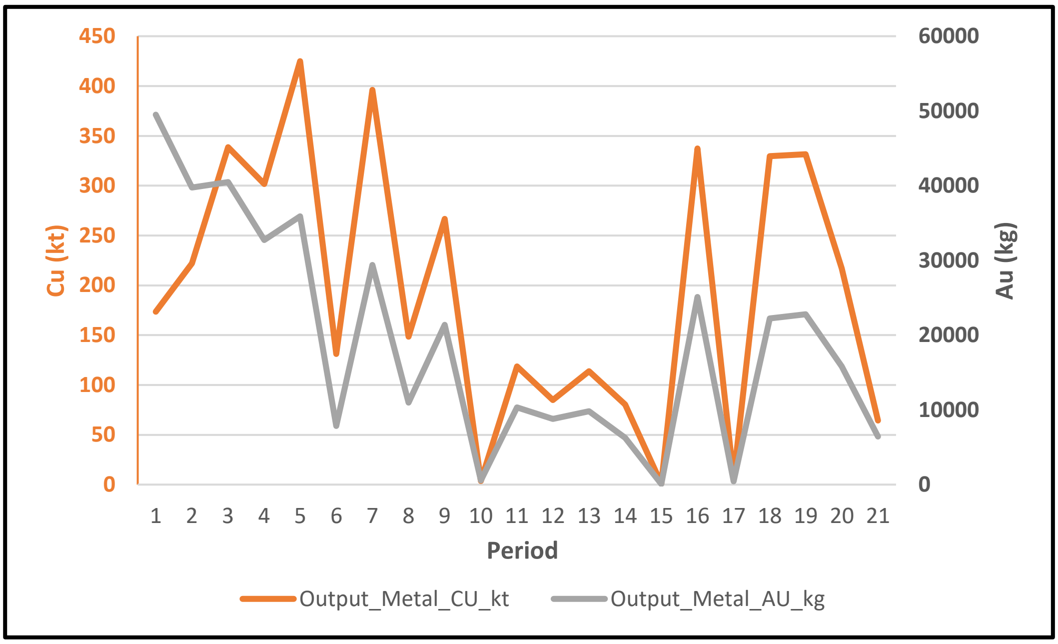

Figure 4.4 Cu and Au metal outputs over the life of the mine

The steady increase in copper metal yield in the first five years is evident. As explained earlier, this is brought about by waste stripping. Gold yield tends to follow a similar pattern to copper yield from Year 5.

It should be noted that the mine production process and the production dump process run concurrently. The first chain accommodates the valuable fraction of the deposit to be processed while the second takes care of the valueless fraction of grades below cut-off. The outcome of the production dump is shown in Table 4.3. This material cannot pay for itself.

Table 4.3 Production destined for the dump site

PERIOD	TONNAGE_kt	Output_Metal_CU_kt	Input_Grade_CU_%	Output_Metal_AU_kg	Input_Grade_AU_ppm
1	26824,98	0	0,038	0	0,067
2	25932,52	0	0,056	0	0,098
3	25472,53	0	0,05	0	0,08
4	25469,7	0	0,061	0	0,08
5	25614,69	0	0,033	0	0,035
6	25819,67	0	0,027	0	0,03
7	22243	0	0,055	0	0,058
8	23018,81	0	0,03	0	0,029
9	27700,58	0	0,046	0	0,037
10	25447,56	0	0,009	0	0,015
11	25339,1	0	0,014	0	0,012
12	25257,73	0	0,025	0	0,024
13	25189,67	0	0,028	0	0,02
14	25376,02	0	0,023	0	0,021
15	26206,99	0	0,005	0	0,003
16	24235,83	0	0,025	0	0,02
17	25364,73	0	0,015	0	0,012
18	24991,43	0	0,027	0	0,018
19	25256,79	0	0,059	0	0,039
20	24242,25	0	0,056	0	0,032
21	9385,34	0	0,09	0	0,072
AVERAGE	**24494,7581**	**0**	**0,036761905**	**0**	**0,038190476**

Both Cu and Au grades going to the production dump are significantly small compared to the mine production (see Figure 4.5). They is not expected to produce any metal at a profit; that is why, metal outputs are at zero.

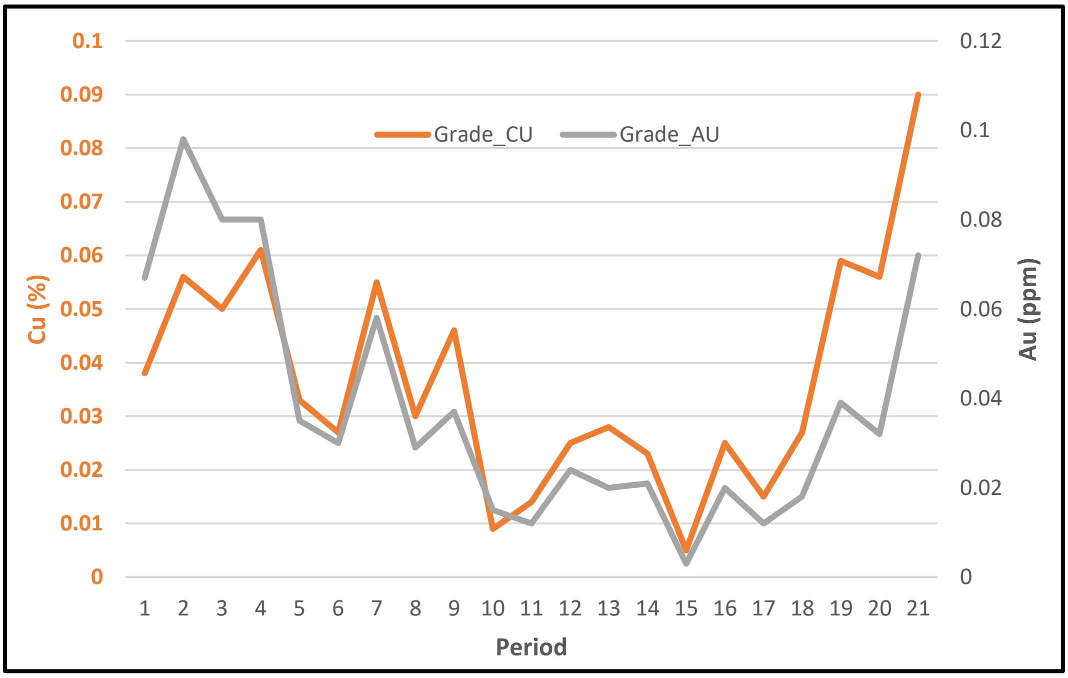

Figure 4.5 Cu and Au grades sent to the dump site

In terms of NPV, it can be seen that Figure 4.6 tends to closely follow the trend of gold in Figure 4.4. Although gold is regarded as a by-product, its contribution to the revenues may be non-negligible probably because it is a highly priced precious metal.

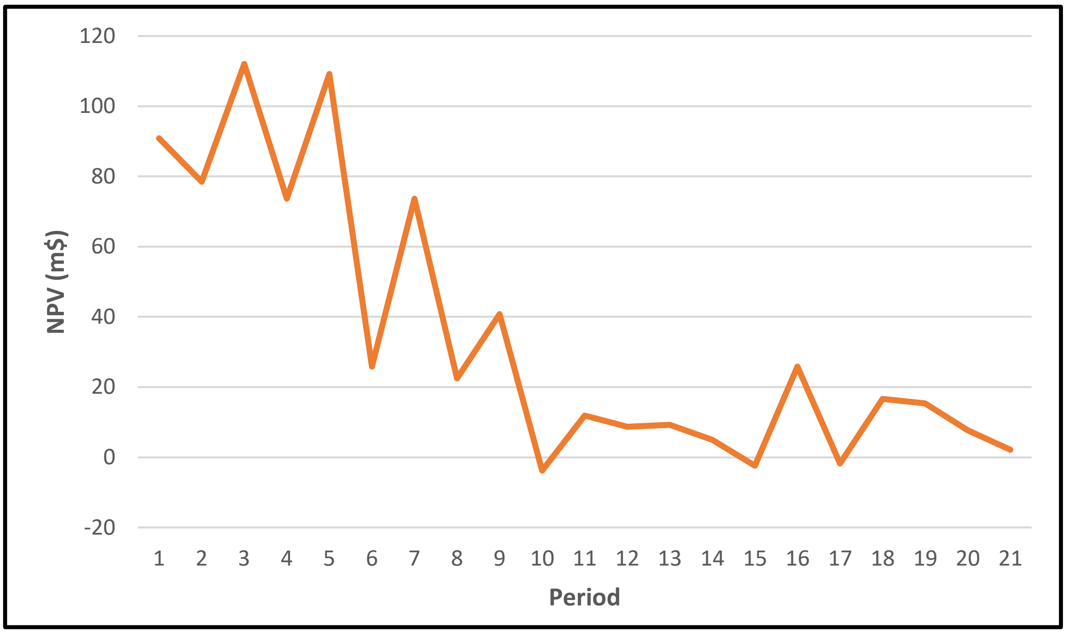

Figure 4.6 Net present value over the life of the mine

Low NPVs and losses are recorded from the tenth year onward while early years are profitable with the highest net present value being 112 000 000 USD. Figure 4.7 also shows the cumulative NPV for the life of the mine.

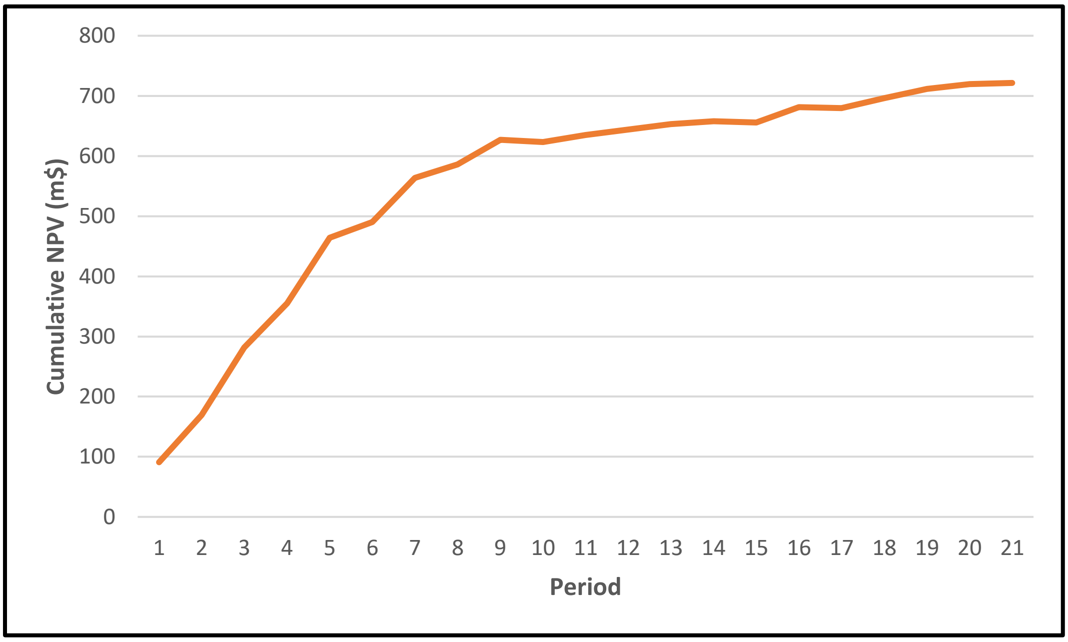

Figure 4.7 Cumulative NPV over the life of the mine

The cumulative NPV accelerate in the early years and slows down from Year 10. This flattens towards the end of life of the mine to a total NPV of slightly above 700 000 000 USD.

4.4 Optimization based on SimSched DBS

The total NPV of 700 000 000 USD in Figure 4.7 was accumulated over a period of 21 years. It would be argued that the NPV was produced following the “optimized” mine production schedule generated with the help of the SimSched DBS algorithm. The implications of this NPV-based optimization plan are analysed in this section.

4.4.1 Parameters and assumptions

Initial input parameters and supporting assumptions made for simulation were covered in Chapter 3. However, the following should be recalled from Figure 4.8: density = 2.75 t/m³; discount rate = 10 %; slope angle = 45°; fixed mining and rehandling costs are at 1.5 USD and 0.8 USD respectively.

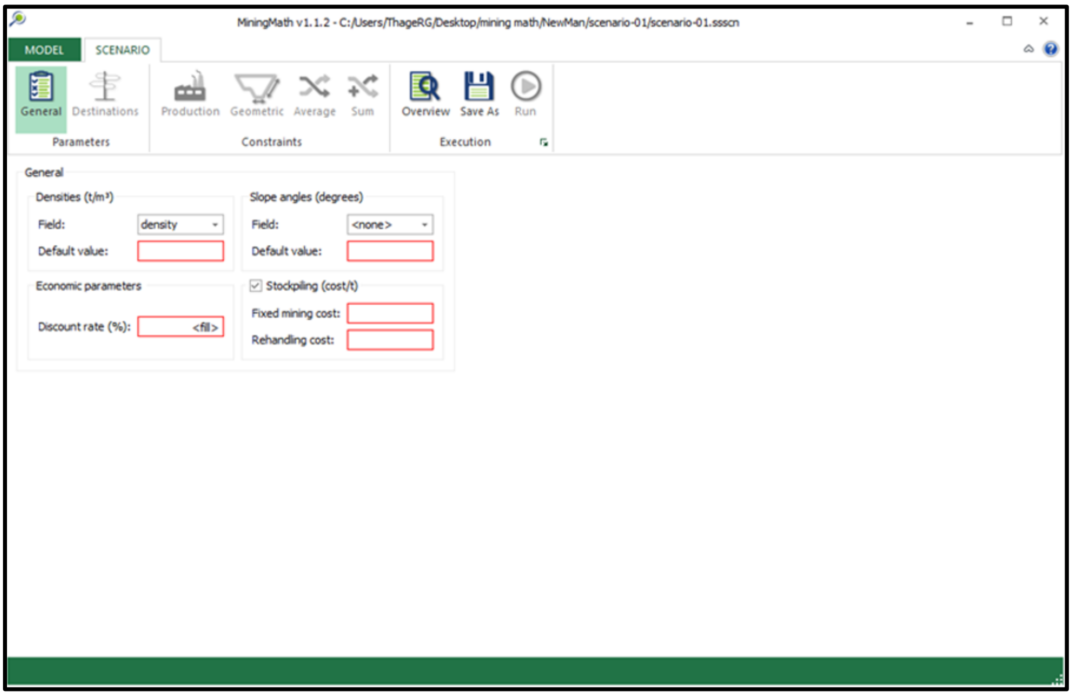

Figure 4.8 SimSched DBS setup

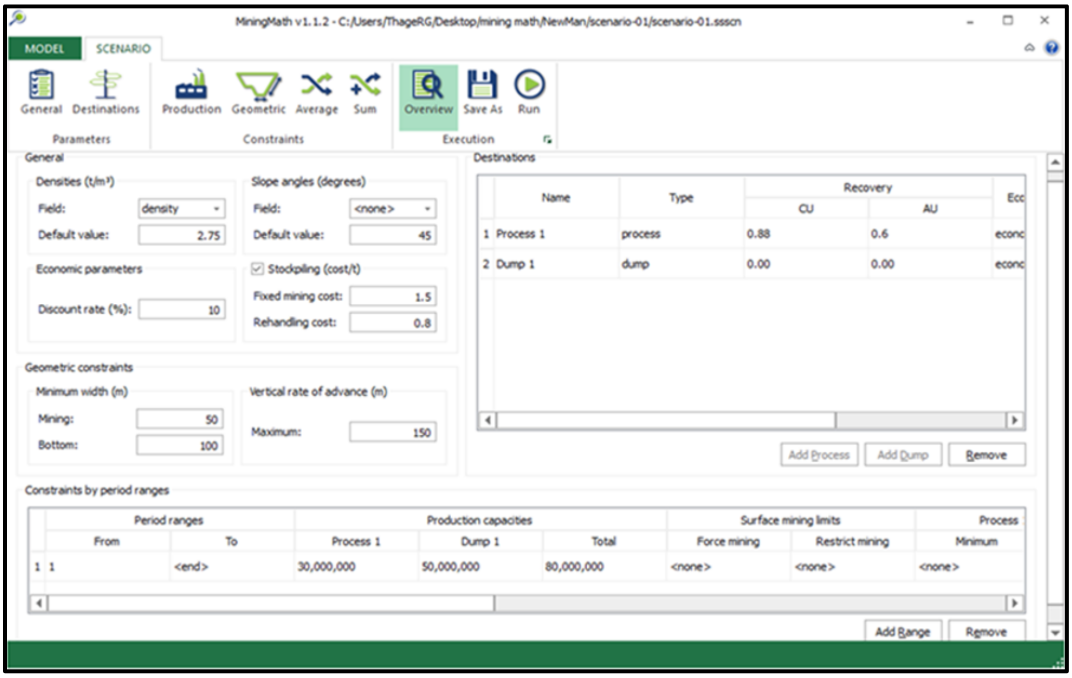

Figure 4.9 Synopsis of parameter setup in SimSched DBS

4.4.2 Production rates from optimized SimSched DBS plan

It is necessary to do an inventory of the volume of material to be excavated over time in line with the 21-year optimized mine plan. Based on the data generated in SimSched DBS, Table 4.4 was produced to that effect.

Table 4.4 SimSched total production over the life of a mine

Period	Production (mt)	Period	Production (mt)
1	84,97229932	**12**	39,63822484
2	83,72131937	**13**	45,45397088
3	84,99196960	**14**	39,12935921
4	85,25102855	**15**	26,54338944
5	85,44456346	**16**	76,26637722
6	40,38709120	**17**	27,23620416
7	82,24192678	**18**	83,51195584
8	45,49580435	**19**	84,29858816
9	68,25602656	**20**	67,48019520

10	27,15451471	**21**	23,94932352
11	46,47618977		

The tonnage of exploited high-grade ore from the beginning to Year 5 is between 83 Mt and 86 Mt. Metal yield is also high during this period. However, between Year 10 and Year 15, SimSched DBS estimates a drop in production as reported in Figure 4.10; this is partly due to the production capacity being fully utilized with a mix of high- and low-grade ore.

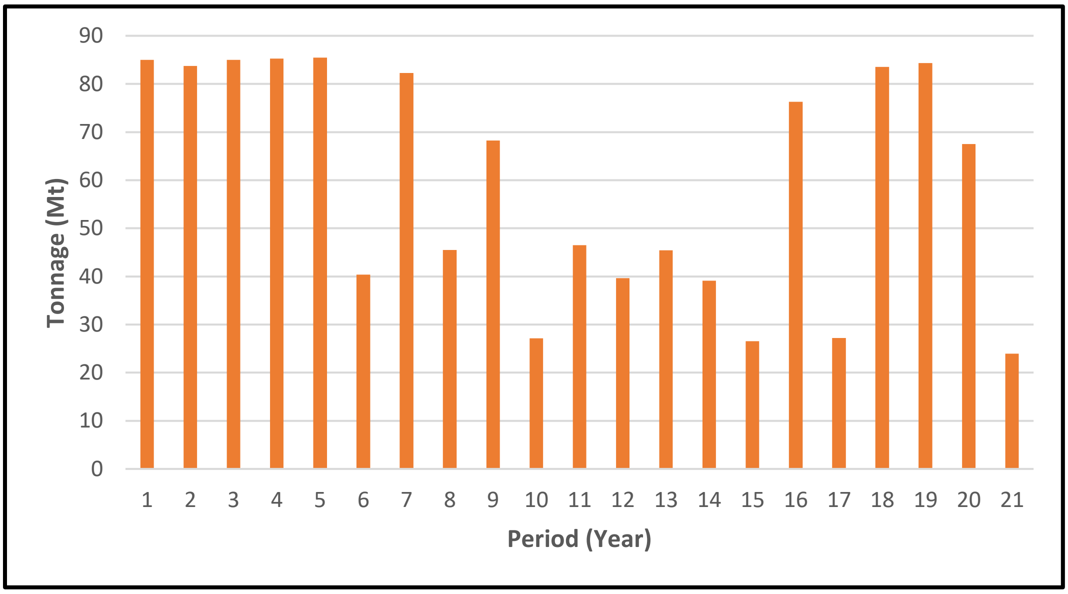

Figure 4.10 Total tonnage of ore produced over the life of the mine

The recuperation of metals at the low tonnages recorded between Year 10 and Year 15 may probably not encourage the selective mining of high-grade ore. Further research should be initiated to investigate the effect of low tonnages on metal recuperation. Grades over the life of mine are generally high while economic block are still available. Once the pit nears exhaustion, stockpile recovery begins with a steady drop in grades until closure.

It is important to state that SimSched DBS also optimizes the net return of the smelter. This is seen in the monetary value of individual blocks in the orebody model. In the first quarter of the life of the mine, the revenue is optimized by booking the ideal grade of copper. Midway through the life of the mine, low-grade blocks become more accessible which bring about a decrease in net smelter return. Conversely, the decrease is compensated by the excavation of gold-rich blocks planned for the later years. So, even though SimSched DBS optimizes the NPV by globally considering copper and gold content, block values are biased towards gold due to its higher selling price. As a result of this, the “competing” commodities accrue a high net smelter return than initially anticipated. One thing is sure, the pursuit of high-value blocks in the early years of the mining project necessitates high stockpiling capacity. The recovery of the extracted ore in the stockpiles may only begin once the deposit is exhausted. At this point in time, the head grade to the plant starts to drop and leads to the subsequent decline in metal production. When this strategy is compared to one driven by the net return of the smelter, it appears that stockpiling incurs an initial margin on the NPV until Year 9 (see Figure 4.7). Ultimately, the overall cumulative NPV is the same as illustrated in Figure 4.11. Also note the slight discrepancies not exceeding 10 % in NPV over time between Figure 4.7 and Figure 4.11 but the same lifespan of the mine (i.e. 21 years).

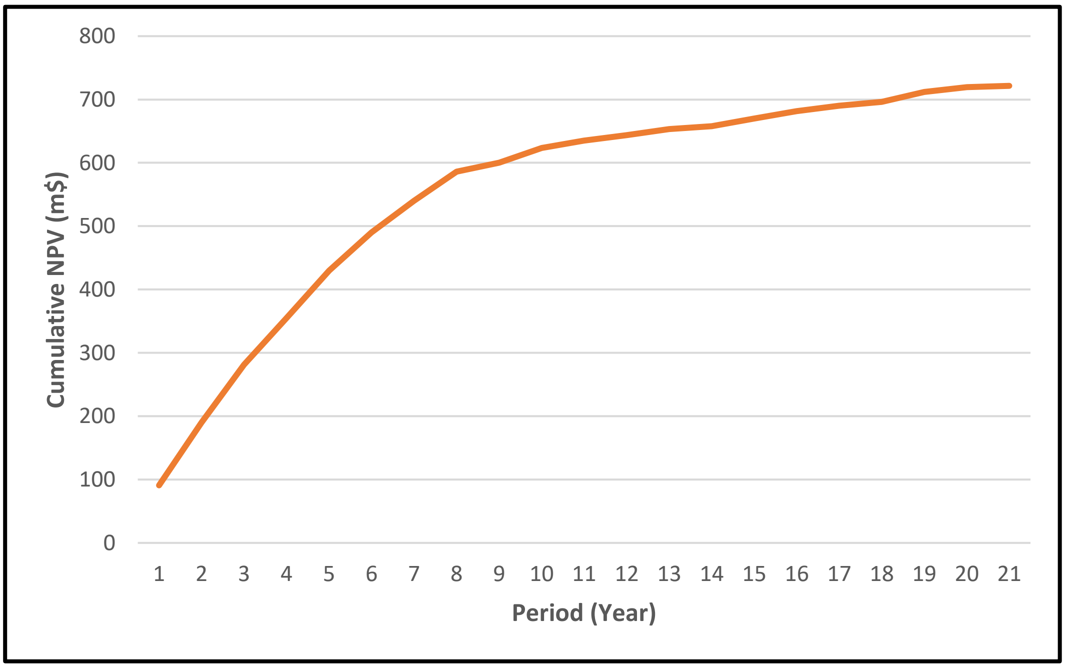

Figure 4.11 Cumulative NPV

It is clear from comparing Figures 4.7 and 4.11 that different strategies may actually affect the distribution of NPV over the life of the mine with varying benefits.

When imposing the stockpiling strategy in Figure 4.11, SimSched DBs generates the optimized production plan reported in Table 4.5.

Table 4.5 Life of mine schedule obtained from SimSched DBS

PERIOD	Waste Mined (kt)	Processed tons (kt)	TCu (%)	TAu (ppm)	Cu Produced (kt)	Au Produced (Kg)
1	26824,98	58147,32	0,298	0,851	173,56	49502,96
2	25932,52	57788,8	0,384	0,688	222,05	39767,12
3	25472,53	59519,44	0,569	0,68	338,64	40498,36
4	25469,7	59781,33	0,505	0,548	301,68	32731,96
5	25614,69	59829,88	0,71	0,6	424,94	35917,75
6	25819,67	14567,42	0,901	0,538	131,2	7830,27
7	22243	59998,93	0,66	0,49	395,97	29409,84
8	23018,81	22476,99	0,661	0,489	148,48	10981,51
9	27700,58	40555,45	0,657	0,527	266,61	21375,17
10	25447,56	1706,95	0,208	0,328	3,54	559,85
11	25339,1	21137,09	0,561	0,489	118,68	10333,57
12	25257,73	14380,49	0,591	0,613	84,98	8809,82
13	25189,67	20264,3	0,562	0,484	113,79	9800,95
14	25376,02	13753,34	0,584	0,455	80,38	6251,32
15	26206,99	336,4	0,286	0,235	0,96	79,17
16	24235,83	52030,55	0,648	0,482	337,27	25102,36
17	25364,73	1871,48	0,338	0,244	6,33	455,81
18	24991,43	58520,53	0,563	0,38	329,65	22229,19
19	25256,79	59041,8	0,562	0,386	331,7	22785,13
20	24242,25	43237,95	0,501	0,366	216,76	15815,75
21	9385,34	14563,98	0,441	0,443	64,22	6451,9
AVERAGE	**24494,7581**	**34929,07**	**0,53**	**0,49**	**194,83**	**18889,99**

It can be seen from Table 4.5 that the mine has great metal yield for quite a long while. Production optimized on cut-off grade is at its greatest during the first 5 years with plant throughput around 60 000 kt. This performance becomes difficult to maintain thereafter declining to below 30 000 kt in several time periods. The reason for this behaviour is that SimSched DBS does not complete optimization based on the grade of a block but on its recoverable value. That is why the final product follows a declining cut-off grade strategy. So, even though the value of recoverable copper is the parameter guiding the optimization, the grades are eventually streamlined.

4.5 Significance of the findings

Cut-off grade optimization involves the excavation of blocks with the highest recoverable values at the beginning of the mine life. This method is accomplished by rapidly mining and stockpiling low-value blocks for later processing. This enables the unearthing of high-value blocks that are then fed to the processing plant. Plant throughput is maintained at a high level so that the initial loan on the investment is repaid earlier. The consequence of this strategy is that metal yield diminishes to uneconomic levels with time until the deposit is exhausted.

The alternative to optimization centred on cut-off grade is to maximize the NPV of the deposit. In the case of this dissertation, the *Newman1* block model was used for the purpose. The simulation results covered in this chapter showed that there is a prospect for improved NPV when cut-off grade is optimized. Indeed, it appears that the two optimization schemes correlate as evidenced by concordant trends in Figures 4.7 and 4.11. A detailed study is needed in order to gain a better understanding of this correlation. Suffice it to say that there seems to be a benefit in stockpiling low-grade blocks to be later fed to the plant instead of relying on blending.

In summary, of the optimization scenarios simulated, it has been shown that the break-even cut-off grade is key to determining the life of a mine. However, considerations of the smelter may have a negative impact on the overall cumulative NPV. That is the reason why SimSched DBS tends to favour the production of more metal in the early stages of mine life. This can potential lead to a build-up of waste material as a result of stripping.

Having established the above as the baseline, the next chapter explores the effects of mining and processing costs on the mine plan.

Chapter 5 Effects of comminution costs on mine planning

5.1 Introduction

In the previous chapter, a baseline was established for the *Newman1* block model in terms of achievable NPV and throughput. A mine plan stemming from the effort was generated for predefined break-even cut-off grade. This made it possible to estimate for example the tonnages to be mined and associated grades of copper and gold over the life of mine.

The optimized mine schedule produced in Chapter 4 is now revisited in this chapter. The idea is to test scenarios whereby the proportion of processing costs relative to mining costs is varied. In doing so, insights on the contribution of comminution costs to mine planning is gained. Ultimately, the findings can be used to guide comminution practice in the context of mine planning in line with the objectives set out in Section 1.3 for this research.

5.2 Comparison of scenarios

A detailed description of the varied comparison scenarios is outlined in this Section. The discussion is documented in chronological order as stated in the introduction of the chapter.

5.2.1 Comparison of cut-off grade policy

A comprehensive comparison on the average and the input grade of both Au (Gold) and Cu (Copper) is shown in Figure 5.1. The maximum break-even cut-off grade of the given metals (Au and Cu) is about 1.0 % over a period of 21 years. It should also be indicated that the duration of the distribution curves is mostly controlled by the added stockpiles which in turn increases the metal availability. The break-even cut-off grade of 1.0 % shows that the input grade for Cu and Au are beneath the cut-off grade. This means that metals with grade beneath the cut-off grade go to the production dump and considered to be waste rock. The average grade shows the Copper and Gold is above the cut-off grade. In this regard, the two ore grades allow mining activities to progress profitably. However, in case the average grades are lower than the required cut-off grade, such ore is not economical to be mined forcing the mine to run at a loss. It makes sense to rather operate at a positive cash-flow to justify the mining business.

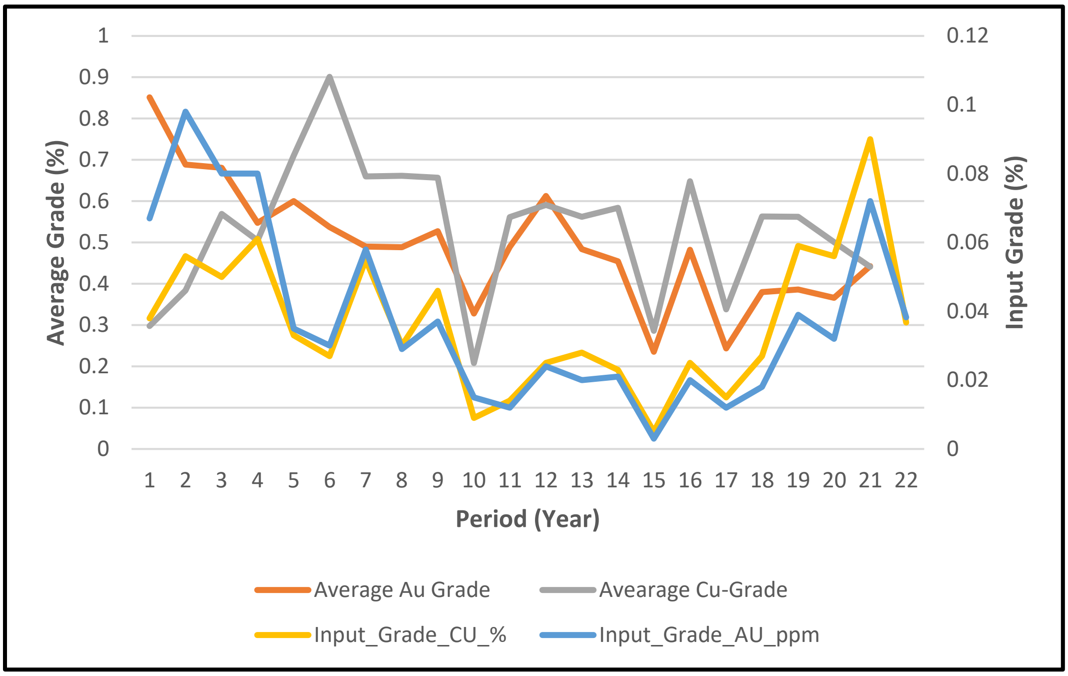

Figure 5.1 Cut-off grade policy

Also note that both SimSched DBS and NPV give a twenty-one-year life of mine at a break-even cut-off grade of 1 %. The current existence of mine cut-off grade policy gives a clear existence of mine of around 21 years.

5.2.2 Comparison of mined tonnages

Figure 5.2 shows that the NPV is most notable during the start of mine life. The NPV is also in line with the target of the cut-off grade optimization since it is above zero tonne. It decreases to a value below zero on the tenth year and gradually decreases towards the end of life of mine. Scheduled optimization in NPV does not lead to high grading; hence, the decrease in the NPV throughout the life of the mine.

Production tonnage follows the same trend as that of the SimSched production total. However, production tonnage is low while the SimSched production is high because of optimization. The treatment of higher-grade ore from the beginning to year five is high and as a result metal yield becomes high. SimSched DBS shows an optimistic production however, during the years ten to fifteen the metal output is low. The production capacity is fully utilized with a mixture of high- and low-grade ore.

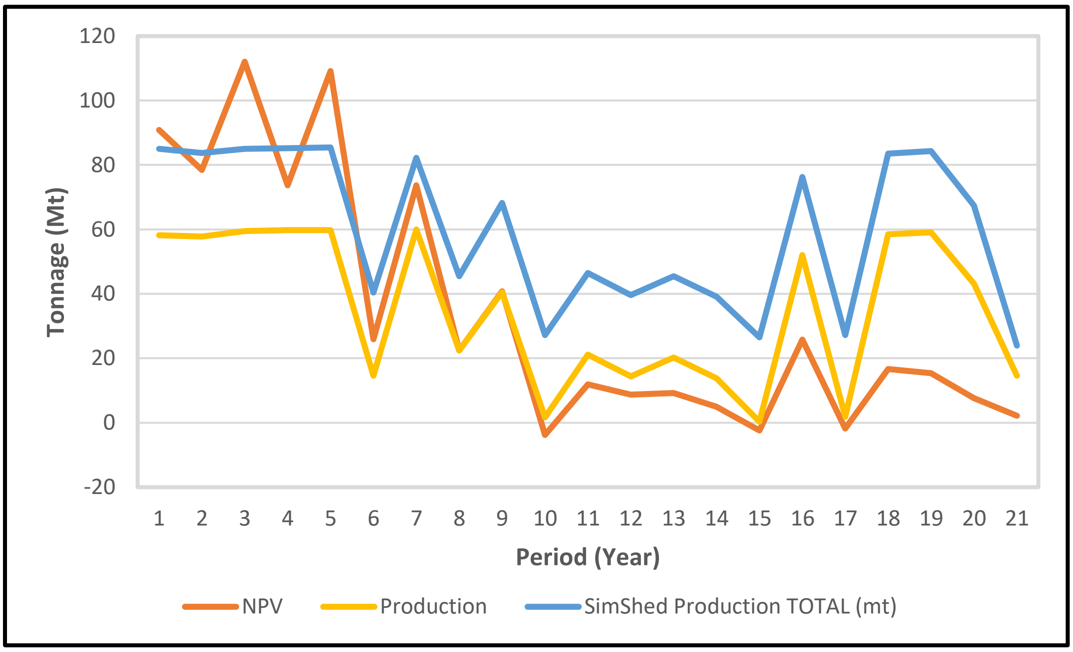

Figure 5.2 Comparison of mined tonnages

The stripping backlog is clear in all the situations for the first quarter (5 years) of the life of mine except for the NPV which decreases (lowest being below 0) and increases throughout the life of mine. A small change in cut-off grade results would result in a huge tonnage loss. SimSched production is higher, and that shows that optimization is at maximum. The current existence of mine and the break-even cut-off grade policy both allow a five-year mining (stripping) life.

5.2.3 Life of a mine for copper and gold production

Copper production in Figure 5.3 shows that the output metal is generally high at the beginning of the life of mine due to SimSched optimization. The copper production gradually decreases until the end of the life of mine. NPVs have demonstrated a negative effect whereby a portion of the metal in the current pit is not financial to mine. The break-even cut-off grade produces copper comparable metal which is a blend of copper and gold. Break-even cut-off policy gives a high copper output which is relative to the NPV and a lower gold output because it is a by-product in the process.

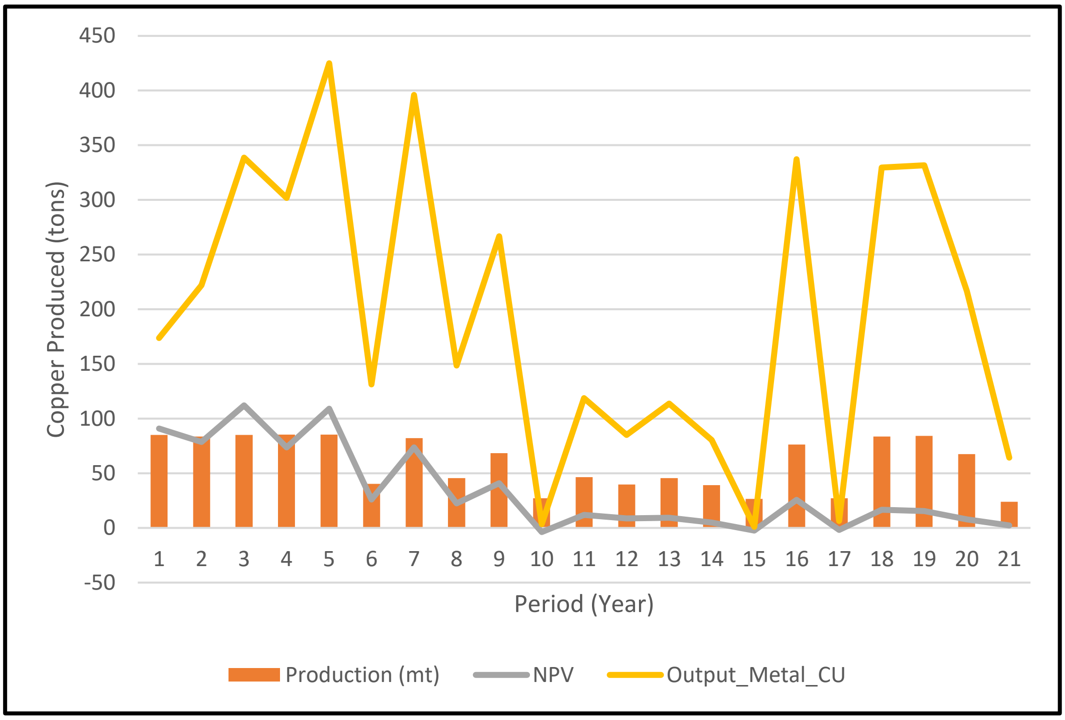

Figure 5.3 Life of a mine copper production, the NPV and metal output

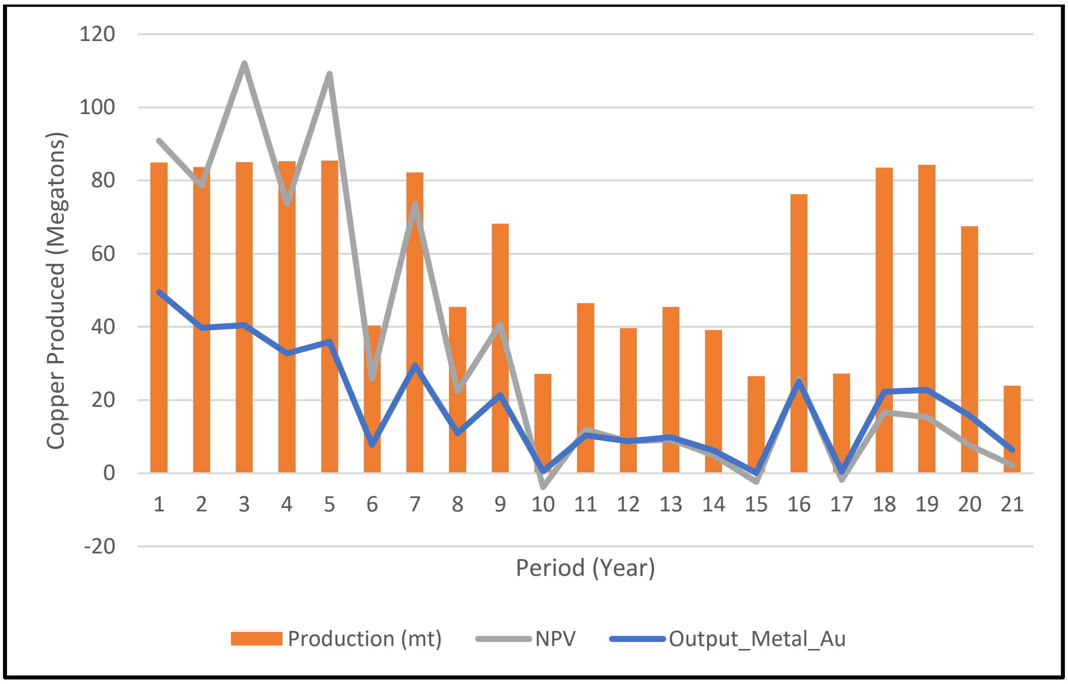

Figure 5.4 Life of a mine gold production, the NPV and metal output

With reference to Figures 5.3 and 5.4, it has been observed that the NPV was ranging from 90.00 to 120.00 Million USD during the first five years. It is anticipated that the NPV distribution in the first five years was mainly affected by break-even cut-off grade policy, these results also correlate very well with previous studies. Mugwagwa (2017) for instance pointed out that the market price affects the NPV of copper since the break-even cut-off policy gives a high copper output than that of gold as gold is the by-product.

5.2.4 NPV variation for the life of mine

All the scenarios investigated show a decrease in NPV throughout the life of the mine. The decrease in NPV is brought about by the declining metal yield. Figure 5.5 shows that the diagram for SimSched gives a lopsided decrease in NPV contrasted with the current existence of mine. The current existence of mine gives the most elevated NPV in the principal quarter before bringing down the remainder of the diagrams. This is brought about by the imperfect stripping to clear the waste mining accumulation. There is hence a need to smoothen this by the cut-off grade strategy from SimSched DBS and that plainly shows in the initial 5 years of the life of mine.

A correlation of the NPV from the researched scenarios gives the outcomes that appeared in Figure 5.5. The current existence of mine arrangement was required to give the most elevated NPV since it consolidates extra metal from the stockpiles. Nonetheless, this is not the situation because of deferred money inflows and higher money outflows brought about by increased stripping in the prior years of the mine life.

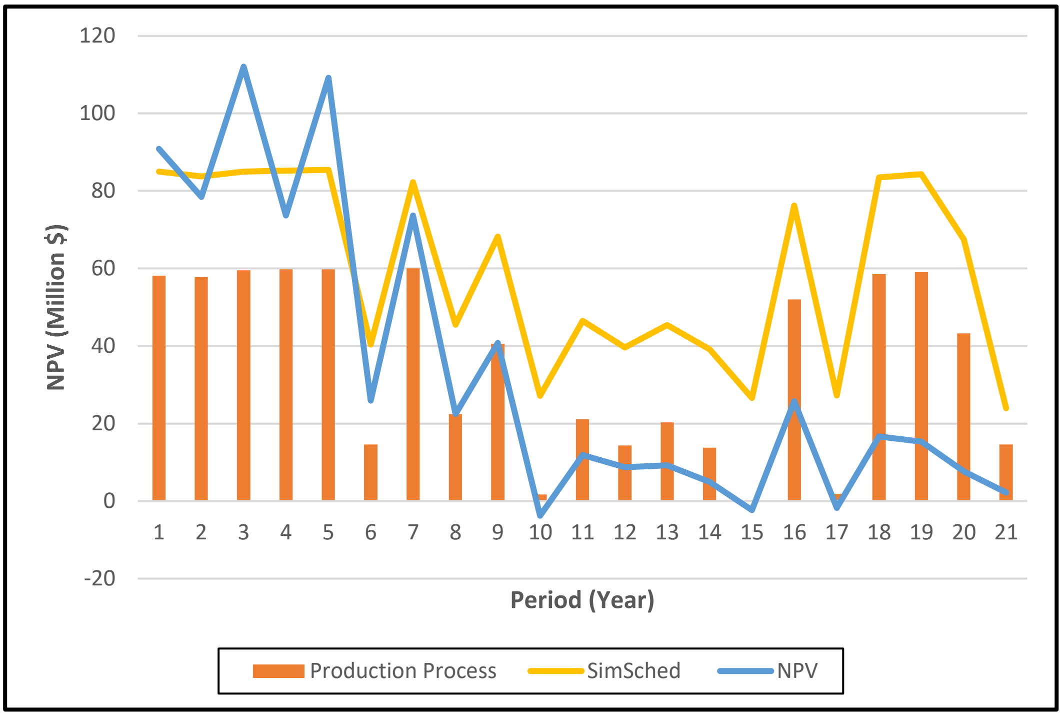

Figure 5.5 Net present value

The NPVs give the most noteworthy NPV because of deferred money outflows by diminished mining toward the start of the life of the mine as appeared in Figure 5.5. SimSched NPV is the highest because it has been based on optimization principles. The cut-off grade optimization based on recoverable value tends to high-grade the block model. This then sterilizes a portion of the lower-grade metal. The monstrous stripping and stockpiling towards the start of the life mine negatively affect the mine life because of higher money surges brought about by early stripping and re-handle from the reserves.

5.2.5 Overall tonnes mined

Figure 5.6 shows that optimization in SimSched yielded great results in the first five years (highest tonnage being over 60 000 kt). The equivalent optimized grade follows the decreasing cut-off grade, which results in higher metal yield in the first five years. This is also in line with the optimization of the cut-off grade and value of the mine. The policy to accelerate the mining rate to get the higher-grade ore for NPV optimization also influences the decision. There is a massive drop in tonnage mined in the sixth year which then went up quickly (From 60 000 kt to 14 000 kt and 60 000 kt). During the tenth year to year fifteen, there is more waste mined than the actual payable ore and the increase in payable ore increased until the end of the life of the mine. This also marks the beginning of the decline in grade and hence the subsequent decline in metal production.

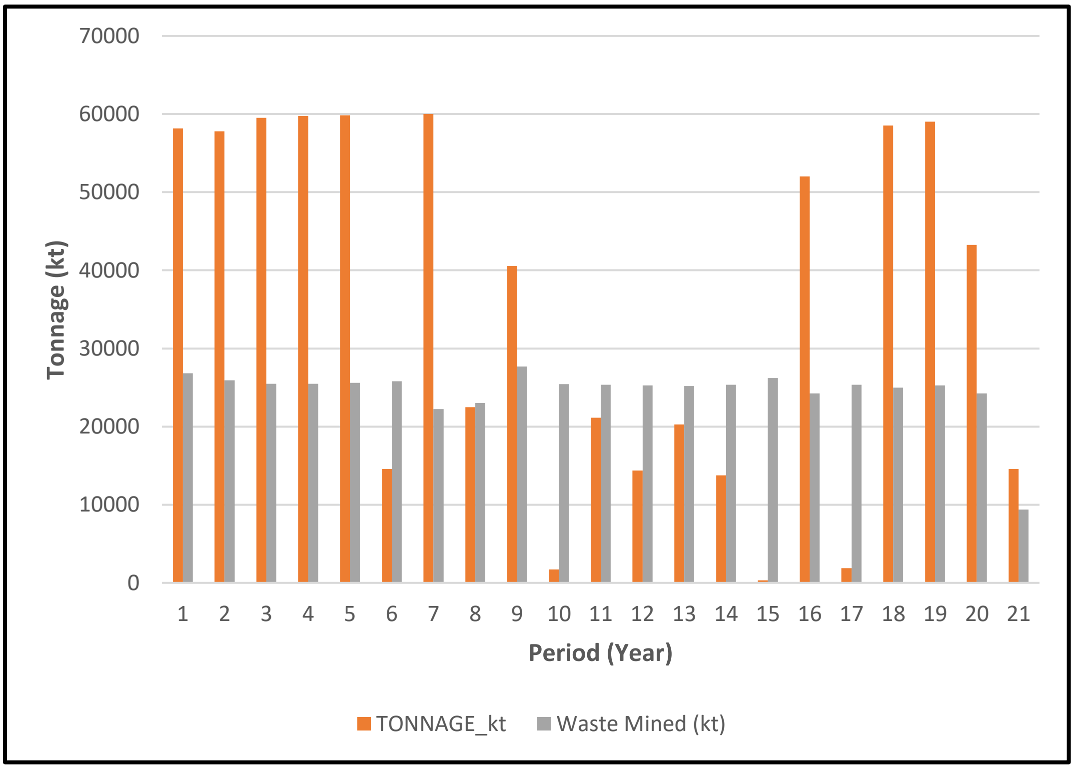

Figure 5.6 Overall tonnes mined

Figure 5.7 shows the tonnes mined as shown on SimSched. Red blocks represent high-grade ore, while blue represents waste mined.

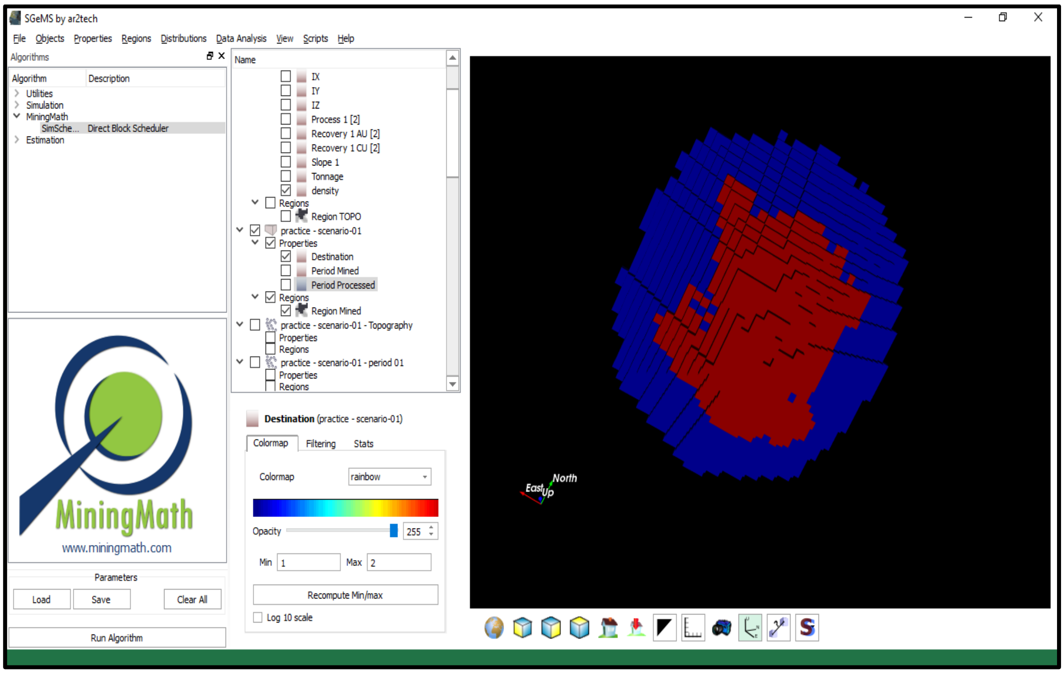

Figure 5.7 Grade variation of tonnes mined rendered in SimSched DBS

5.3 Analysis of the effects of processing and mining costs

5.3.1 Effect of processing costs on the cut-off grade

With reference to Figure 5.8, it should be stated that the processing costs are varied between 70 % and 140 % from the baseline. The effects are then recorded in terms of the cut-off grade. There is a small increase between 70 % and 75 % of the processing costs. However, the pattern of the graph shown in Figure 5.8 shows that the increase in processing costs results in the increases of the cut-off grade.

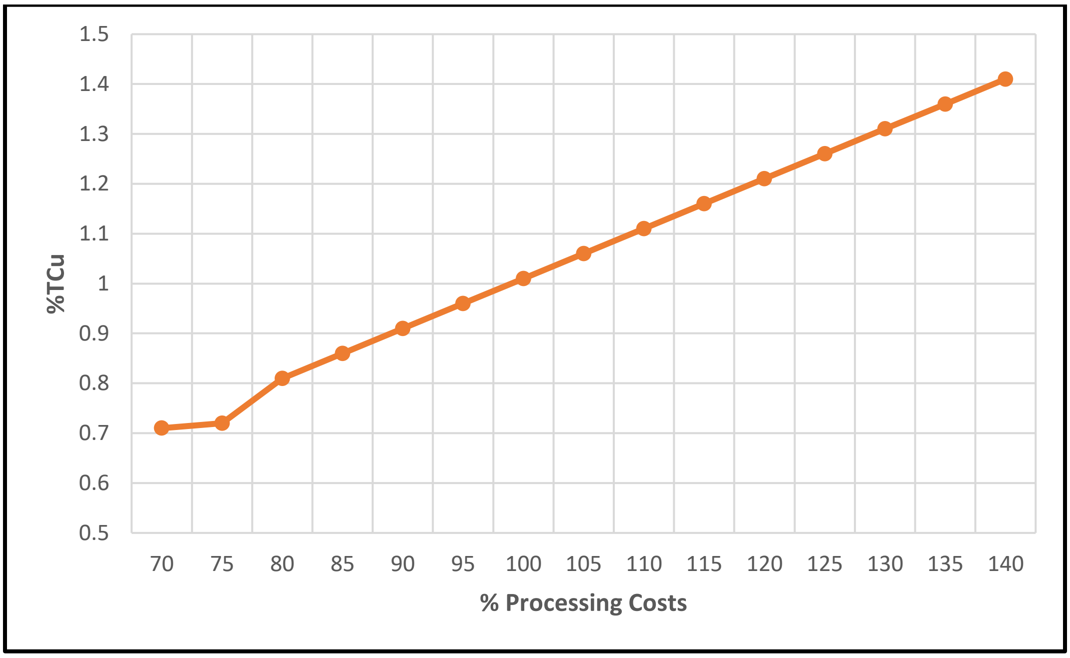

Figure 5.8 Copper grade of mined ore as a function of processing costs

The increase in cut-off grade reduces the metal output of the orebody as illustrated in Figure 5.9. Figure 5.9 shows a decreasing pattern in the metal output as the processing costs increase. The increase is very minimal between 80 % and 95 % of the processing costs and shows a significant increase from 100 % to 140 % of the processing costs.

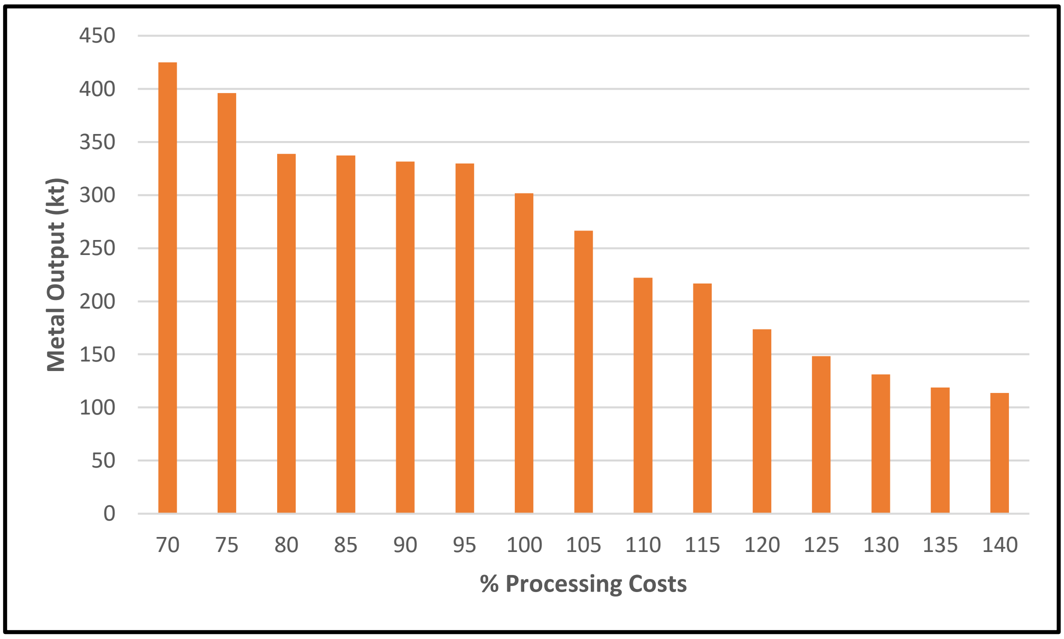

Figure 5.9 Metal output from mined ore as a function of processing costs

The decrease in metal output proves that there is a decrease in ore reserves presented for treatment as there is also an increase in the cut-off grade as shown in Figure 5.9 above.

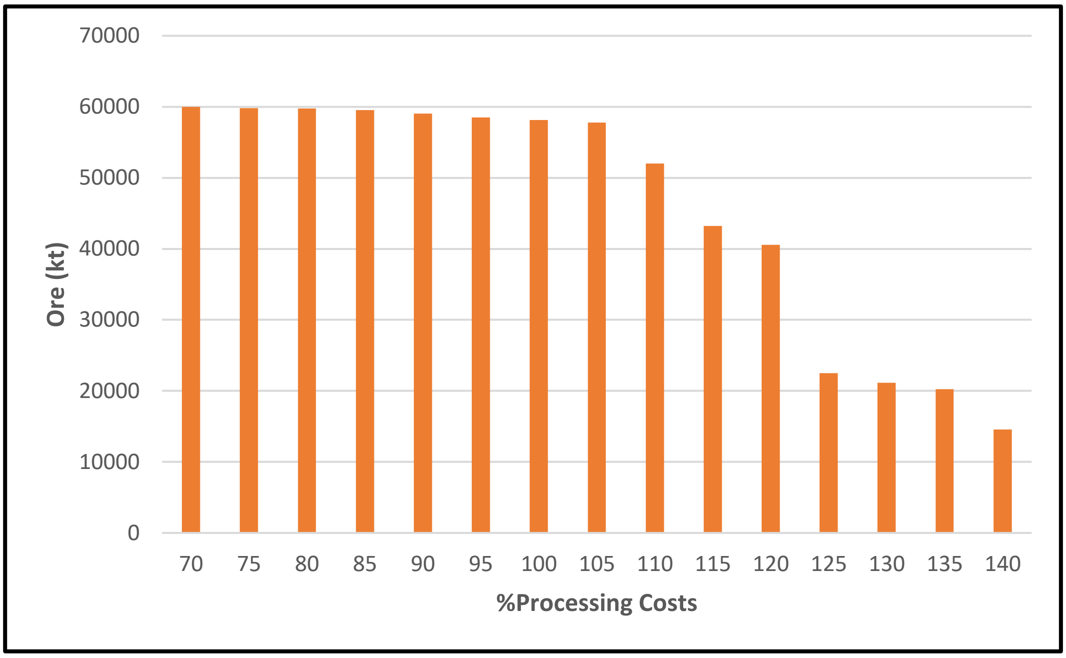

Figure 5.10 Ore reserves as a function of processing costs

5.3.2 Effect of mining costs on the NPV

Figure 5.11 denotes the distribution of mining cots with NPV in percentage. The general distribution of the curves revealed that NPV values ranging between 72.1 to 112.1 Million USD are estimated, with the mining cost ranging from 70 % to 140 %. It is also observed that the majority of the NPV are scattered across 79 % to 80 %, this is influenced by cut-off grade optimization.

Based on Figure 5.11, it is observed that from 70 % to 85 % of the mining cost the NPV shows a gradual decrease, it is anticipated that the gradual decrease might have been influenced by the extraction ratio. Furthermore, the distribution has shown some rapid increase in the NPV as the distribution approaches a mining cost of approximately 91.2 %. For the remaining part, the distribution shows a variation that alternates between 70 % and 90 %.

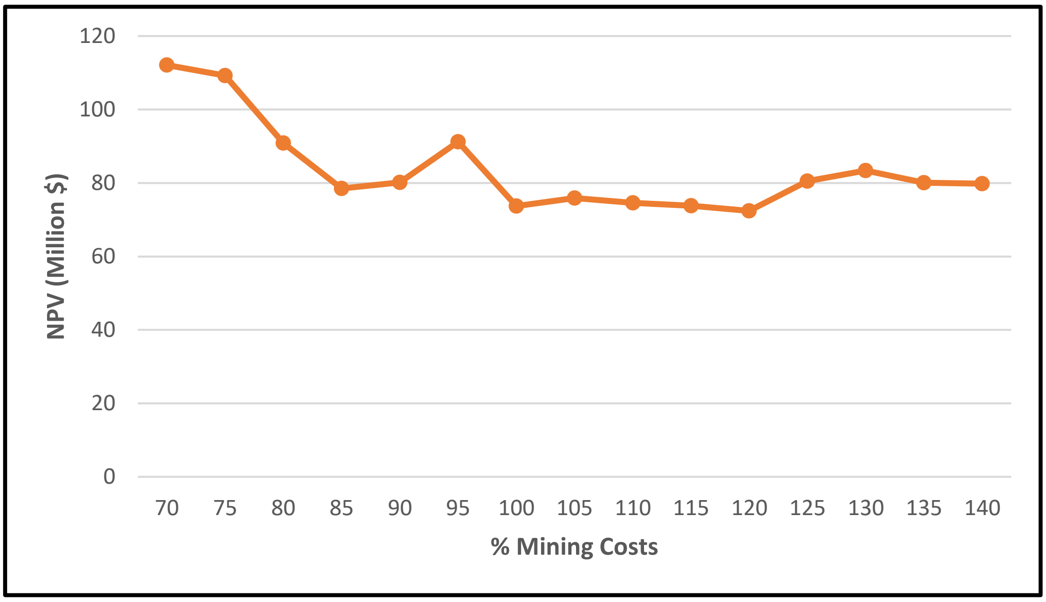

Figure 5.11 Variation of the NPV ore with mining costs

There is an inverse proportion between the NPV and the mining costs during the 70 % and 85 % variations. Results moreover show a dominant pattern of stability in the NPV from 100 % to 140 % of the mining costs variation.

5.3.3 Effect of the discount rate on the NPV

Shifting the discount rate has no impact on the cut-off grade and that is the explanation it has not differed because it follows the same trend. The discount rate is utilized on the benefit after all the allowances are made. For this situation, the limiting rate is not being treated as an expense and it has no part in cut-off grade optimization. However, the NPV decreases with an increase in the discount rate.

Figure 5.12 shows that all graphs for various scenarios with discount rate values above 10 % are below the base value graph as selected in SimSched. This analysis shows that trying to recover capital by increasing the discount rate does not work for the Newman1 block model. The total

NPV additionally diminishes when the discount rate is increased. High discount rates negatively affect the estimation of the block model.

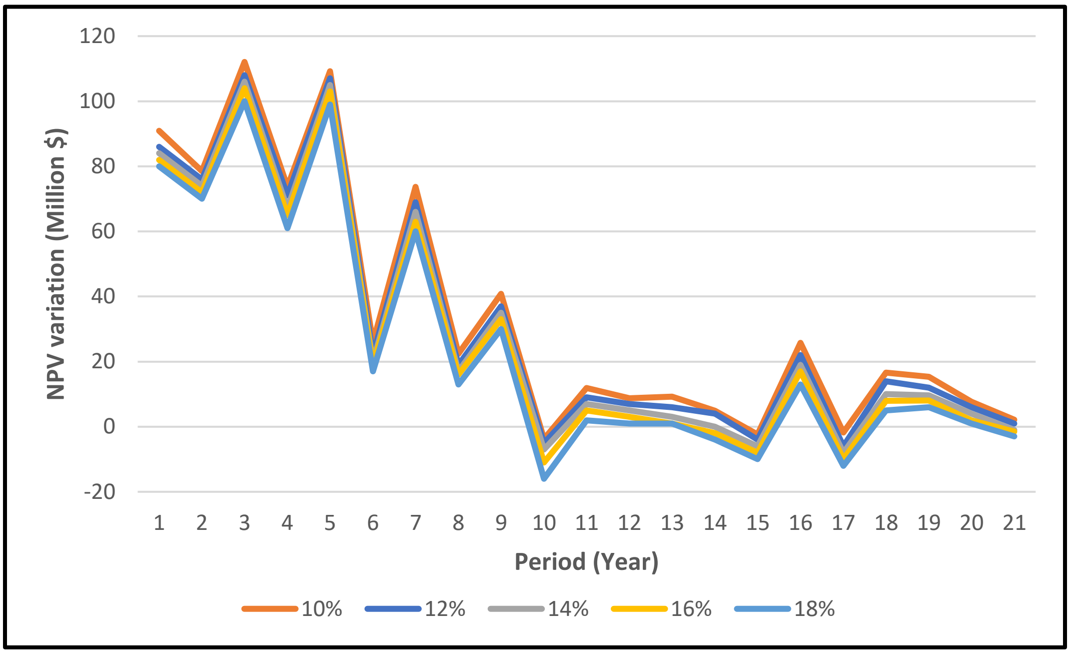

Figure 5.12 Effects of the discount rate on the NPV over the mine life

Indeed, the distribution of NPV variation with several discount rates has shown that the distribution is not neutral throughout, there is an increase, decrease distribution observed along the curves (see Figure 5.12). All the curves follow the same pattern as that of the 10 % discount rate (Base curve). The more the discount rate, the lower the NPV variation.

5.3.4 Effect of variation on the copper price

The dissemination of the minable grade in the orebody is such that individual patterns are not noticeable. The pattern is visible on the minable grade at every rate change in cost. As the copper cost is increased the grade diminishes. Thus, a portion of the lower grade material that is uneconomic to extract at the lower cost becomes economic and brings down the grade of the mineral accessible for mining which increases the reserves. An increase in price advantages more ore for processing. The increased copper output with a price increase indicates increased cash inflows and hence an increase in NPV.

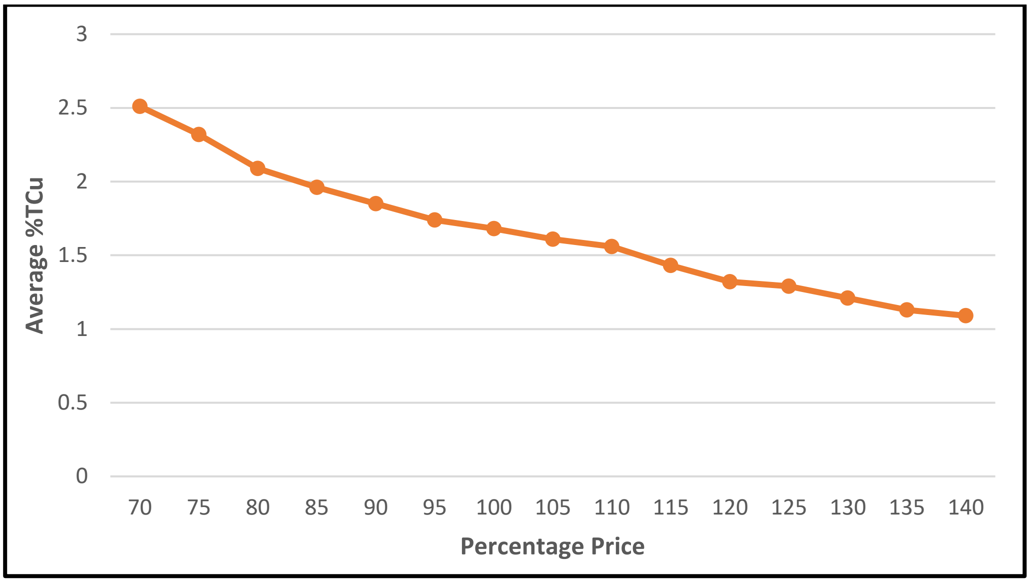

Figure 5.13 Effects of variations on commodity price on the average grade of the optimally mined ore

5.4 Concluding remarks

Of the explored scenarios, it has been demonstrated that the break-even cut-off grade gives a long-term life of mine with high metal yield. SimSched DBS advances the production of more metal in the first five years of the life of mine and that shows that more profit can be made during the life of a mine. The outcomes have demonstrated that mining costs have no impact on the preparing of cut-off grade. However, the increase in metal value brings down the cut-off grade and vice versa.

Chapter 6 Conclusions and recommendations

6.1 Introduction

The integration and impact of comminution costs on open pit mine planning has been investigated in this dissertation. Based on the literature review covered, it was established that comminution alone accounts for 30 – 70% of the overall operational costs of a mining project. This has subsequently drawn research into ways of reducing associated processing costs. On one front, energy-efficient technologies are being developed while on another innovative processing circuits are seeing the light. The present dissertation attempted to approach the problem differently. Indeed, it explored the effects of processing and mining costs on an open pit mine plan. Simulations were employed to generate production schedules from the block model of a copper-gold deposit. Mine planning was done with the help of SimSched Direct Block Scheduler (DBS), a freely download software for pit optimisation. The optimisation algorithm behind SimSched DBS is essentially based on the seminal work of Lerchs and Grossman (1965).

Various scenarios were considered as part of the simulation work.

The orebody modelling theory is used to determine the value of the mineral deposit and the potential of making a return by a prospective venture by analysing the values of grade, tonnage.

Simulation results of various scenarios with assumed constraints are executed in Chapter 4. Pit optimization was carried out using SimSched DBS and satisfactory results in line with the objectives of this study are acquired. The analysis presented is predicated on the assumption that is relevant to the current mining laws.

6.2 Concluding remarks

The mine life schedule created with the help of SimSched DBS for the *Newman1* block model provided an optimized cut-off grade. Planned mining times are shown in different colours in Figure 3.13.

Waste stripping schedules by cut-off grade optimization showed that high-grade ore will be usable from the beginning of the first year of the mine existence. All the methods used to produce the cut-off grade policy in this study have shown that high-grade minerals can be used in the early years of mine existence.

There is a great opportunity to boost the NPV of the *Newman1* block model. This can be achieved by improving the cut-off grade. This is demonstrated by the comparison of the NPV from the current life of mine provided by the various scenarios in Figure 5.5.

High impacts of copper and gold prices on block values can cause high grading. This is demonstrated by a lower grade material that is no longer economical to process because it makes maximum use of plant capacity with less metal production. NPVs are not as aggressive as SimSched DBS in optimizing metal production, thus giving a smoother schedule.

Copper clarifies the optimization process; however, it may not be suitable for more economic study.

The break-even cut-off grade has the lowest NPV rating. This means that operating a mine at break-even cut-off grade does not completely maximize the value of the operation. As such, this cut-off grade policy can only be used as a quick estimate.

The possibilities of improving the NPV from that based on SimSched DBS is very high. This can be achieved by concentrating on important input parameters, such as lowering production costs, increasing throughput, and improving the policy of cut-off grades.

6.3 Recommendations for future work

Controlling the mining rate in the early years of mine life in SimSched DBS is very important as it avoids large cash outflows at the beginning of the life of mine. In the first five years, the scheduled mining rate is extremely high. As a result, this generates a lot of capital in the extracted ore, which is not in line with the time value of the money. Maximizing cash inflows and suspending cash outflows as often as possible without losing the credibility of the business by optimizing the NPV is essential. The mining rate can be reduced at the commencement of the life of the mine can be achieved by suspending cash outflows. The NPV from SimSched DBS could have been much higher with a reduced mining rate.

There are several calculations beyond the SimSched DBS program. SimSched DBS would automatically create economic block values to make the software more user-friendly. This would make the optimization runs even faster from a human point of view. There is a need for SimSched DBS to update the coordinates of the block model to the indices without the operator having to do it manually. The goal is to eliminate human error as much as possible.

Optimization of block economic value appeared to have a high-grade effect on the *Newman1* orebody. The by-product, which is gold, has a higher unit price relative to the main product, which is copper. This led to the addition of copper grade blocks below the break-even cut-off grade. This leads to the scheduling of very low grades which may not be feasible for metal recovery in the factory. Integrating a way to avoid uneconomically low-grade mining can help to avoid this.

REFERENCES

Abouzeid, A.Z.M., Fuerstenau, D.W., 2009. Grinding of mineral mixtures in high-pressure grinding rolls. International Journal of Mineral Processing, vol. 93, no. 1, pp. 59 – 65

Achireko, P.K., 1998. Application of modified conditional simulation and artificial neural networks to open pit optimization. PhD thesis, Dalhousie University

Agoston, M.K., 2005. Computer graphics and geometric modelling: Implementation and algorithms. Springer, London

Asad, M.W.A., 2005. Cutoff grade optimization algorithm with stockpiling option for open pit mining operations of two economic minerals. International Journal of Surface Mining, Reclamation and Environment, vol. 19, no. 3, pp. 176 – 187

Asad, M.W.A., Topal, E., 2011. Net present value maximization model for optimum cut-off grade policy of open pit mining operations. Journal of the Southern African Institute of Mining and Metallurgy, vol. 111, no. 11, pp. 741 – 750

Ataei, M., Jamshidi, M., Sereshki, F., Jalali, S.M.E., 2008. Mining method selection by AHP approach. Journal of the Southern African Institute of Mining and Metallurgy, vol. 108, no. 12, pp. 741 – 749

Austin, L.G., 1984. Concepts in process design of mills. Mining Engineering, vol. 36, no. 6, pp. 628 – 635

Baird, B.K., Satchwell, P.C., 2001. Application of economic parameters and cut-offs during and after pit optimisation. Mining Engineering Journal. vol. 53, no. 2, pp. 33 – 40

Ballantyne, G.R., Powell, M.S., 2014. Benchmarking comminution energy consumption for the processing of copper and gold ores. Minerals Engineering, vol. 65, pp. 109 – 114

Barratt, D., Sherman, M., 2002. Factors which influence the selection of comminution circuits. Mineral Processing Plant Design, Practice and Control, vol. 1, pp. 539 – 565

Božić, B., 1998. Control of fragmentation by blasting. Rudarsko-Geološko-Naftni Zbornik, vol. 10, no. 1, pp. 49 – 57

Caccetta, L., Hill, S.P., 2003. An application of branch and cut to open pit mine scheduling. Journal of Global Optimization, vol. 27, no. 2 – 3, pp. 349 – 365

Casteel, K., 2006. HPGR: Making its case for commercialization. Engineering and Mining Journal, vol. 207, no. 1, pp. 60 – 64

Chandran, B.G., Hochbaum, D.S., 2009. A computational study of the pseudoflow and push-relabel algorithms for the maximum flow problem. Operations Research, vol. 57, no. 2, pp. 358 – 376

Charles, W.D., Gallagher, A.E.J., 1982. Comminution energy usage and material wear. Design and Installation of Comminution Circuits, Society for Mining, Metallurgy and Exploration (SME – AIME), pp. 248 – 274

Chaves, L.S., Carvalho, L.A., Souza, F.R., Nader, B., Ortiz, C.E.A., Torres, V.F.N., Câmara, T.R., Napa-García, G.F., Valadão, G.E.S., 2020. Analysis of the impacts of slope angle variation on slope stability and NPV via two different final pit definition techniques. REM – International Engineering Journal, vol. 73, no. 1, pp. 119 – 126

Dagdelen, K., Kawahata, K., 2007. Cut-off grade optimization for large scale multi-mine, multi-process mining operations. Proceedings of International

Symposium on Mine Planning and Equipment Selection (MPES), pp. 226 – 233

Dagdelen, K., 1992. Cutoff grade optimization. Proceedings of 23rd Application of Computers and Operations Research in the Mineral Industry (APCOM), University of Arizona, Tucson, Arizona, pp. 157 – 165

Dagdelen, K., 2001. Open pit optimization – strategies for improving economics of mining projects through mine planning. 17th International Mining Congress and Exhibition of Turkey, pp. 117 – 121

Dagdelen, K., Francois-Bongarcon, D., Johnson, T.B., 1986. Optimum open pit mine production scheduling by Lagrangian parameterization. Proceedings of the 19th International Symposium on the Application of Computers and Operations Research in the Mineral Industry (APCOM), pp. 127 – 142

Daniel, M.J., Lewis-Gray, E., 2011. Comminution efficiency attracts attention. AusIMM Bulletin, vol. 5, pp. 20 – 30

Daniel, M.J., Morrell, S., 2004. HPGR model verification and scale-up. Minerals Engineering, vol. 17, no. 11 – 12, pp. 1149 – 1161

DME, 1999. Geotechnical Considerations in Open Pit Mines – Guideline, Version 1.0. Department of Minerals and Energy, Western Australia, https://www.dmp.wa.gov.au/Documents/Safety/MSH_G_GeotechnicalConsiderationsOpenPitMinespdf.pdf

Dowd, P., 1994. Risk assessment in reserve estimation and open pit planning. Transactions of the Institution of Mining and Metallurgy, Section A: Mining Industry, vol. 103, pp. A148 – A154

Drozdiak, J.A., 2011. A pilot-scale examination of a novel high pressure grinding roll/stirred mill comminution circuit for hard-rock mining applications. Doctoral Dissertation, University of British Columbia

Eloranta, J., 1995. The selection of powder factor in large diameter blast holes. Proceedings of the 21st Annual Conference on Explosives and Blasting Research, vol. 1, Nashville, TN, pp. 68 – 77

Erarslan, K., Celebi, N., 2001. A simulative model for optimum open pit design. Canadian Institute of Mining (CIM) Bulletin, vol. 94, no. 1055, pp. 59 – 68

Erarslan, K., 2012. Computer aided ore body modelling and mine valuation. In Earth Sciences, Imran, A.D. (Ed.), InTech Europe, Rijeka, Croatia, pp. 345 – 372

Espinoza, D., Goycoolea, M., Moreno, E., Newman, A., 2013. MineLib: a library of open pit mining problems. Annals of Operations Research, vol. 206, no. 1, pp. 93 – 114

Fuerstenau, D.W., Abouzeid, A.Z., 2002. The energy efficiency of ball milling in comminution. International Journal of Mineral Processing, vol. 67, no. 1 – 4, pp. 161 – 185

Goddard, W., Melville, S., 2001. Research methodology: An introduction, Second Edition. Juta Press, ISBN 0 7021 5660 4

Gokhal, V.B., 2010. Cost analysis of rotary blasthole drilling. CRC Press, ISBN 9781138496255

Halatchev, R., Moustakerov, I. 1994. Optimum scheduling of waste and ore production in open pit mines. Mining Technology Journal, vol. 78, no. 894, pp. 61 – 64

Hall, B., 2014. Cut-off grades and optimising the strategic mine plan. Spectrum Series 20, The Australasian Institute of Mining and Metallurgy (AusIMM), Carlton Victoria, Australia

Hartman, H.L., 1992. Mining engineering handbook, 2nd Edition. Society for Mining, Metallurgy and Exploration (SME – AIME), Littleton

Hlabangana, N., Danha, G., Hildebrandt, D., Glasser, D., 2016. Use of the attainable region approach to determine major trends and optimize particle breakage in a laboratory mill. Powder Technology, vol. 291, pp. 414 – 419

Hochbaum, D.S., 2001. An efficient algorithm for image segmentation, Markov random fields and related problems. Journal of the Associate for Computing Machinery (JACM), vol. 48, no. 4, pp. 686 – 701

Hoek, E., Bray, J.D., 1981. Rock slope engineering. Third Edition, CRC Press, London

Hukki, R.T., 1975. The principles of comminution: An analytical summary. Engineering and Mining Journal, vol. 176, pp. 106 – 110

Hummel, D., McCann, M.J., 2011. Wireless detonator assemblies, and corresponding networks. U.S. Patent 7,929,270. Orica Explosives Technology Pty Ltd

Hustrulid, W., Kuchta, M., 2006. Open pit mine planning and design – vol. 1 Fundamentals. Taylor & Francis

Hustrulid, W.A., Kuchta, M., Martin, R.K., 2013. Open pit mine planning and design. Two volume set & CD-ROM pack, CRC Press

Jackson, J.A., 1997. Glossary of geology. American Geological Institute, Alexandria, Virginia

Jimeno, E.L., Jimino, C.L., Carcedo, A., 1995. Drilling and blasting of rocks. CRC Press

Kawatra, S.K., Eisele, T.C., Weldum, T., Larsen, D., Mariani, R., Pletka, J., 2005. Optimization of comminution circuit throughput and product size distribution by simulation and control. Technical Report, Michigan Technological University, United States of America, doi:10.2172/837186

Kennedy, B.A., 1990. Surface mining – Second Edition. Society for Mining, Metallurgy and Exploration (SME – AIME), Littleton

Khumalo, N., 2007. The application of the attainable region analysis in comminution. PhD thesis, University of the Witwatersrand, Johannesburg

King, B., 2011. Optimal mining practice in strategic planning. Journal of Mining Science, vol. 47, no. 2, pp. 247 – 253

Labys, W.C., Thomas, H.C., 1975. Speculation, hedging and commodity price behaviour: an international comparison. Applied Economics, vol. 7, no. 4, pp. 287 – 301

Lane, K.F., 1988. The economic definition of ore: Cut-off grades in theory and practice. London: Mining Journal Books Limited

Lane, K.F., 1964. Choosing the optimum cutoff Grade. Colorado School of Mines Quarterly, vol. 59, pp. 811 – 829

Lerchs, H., Grossman, L., 1965. Optimum design of open-pit mines. Transaction CIM, vol. 58, no. 1, pp. 47 – 54

Moema, J.S., Papo, M.J., Slabbert, G.A., Zimba, J., 2009. Grinding media quality assurance for the comminution of gold ores. World Gold Conference, The Southern African Institute of Mining and Metallurgy, pp. 27 – 34

Morales, N., Jélvez, E., Nancel-Penard, P., Marinho, A., Guimarães, O., 2015. A comparison of conventional and direct block scheduling methods for open pit mine production scheduling. Application of Computers and Operations Research in the Mineral Industry (APCOM), pp. 1040 – 1051

Morley, C., 2010. HPGR – FAQ. Journal of the Southern African Institute of Mining and Metallurgy, vol. 110, no. 3, pp. 107 – 115

Mugwagwa, D., 2017. Cut-off grade optimisation for a bimetallic deposit: A case study of the Ruashi Mine Copper-Cobalt deposit. Master of Science dissertation, University of the Witwatersrand, Johannesburg

Mulenga, F.K., Mwashi, A.M.M., 2018. Use of the attainable region approach for the determination of the run-of-mine size distribution for optimal downstream processing. Society of Mining Professors 6th Regional Conference 2018, Birchwood Hotel and Conference Centre, Johannesburg, 12 – 13 March 2018, The Southern African Institute of Mining and Metallurgy, pp. 141 – 154

Musingwini, C., 2016. Optimization in underground mine planning-developments and opportunities. Journal of the Southern African Institute of Mining and Metallurgy, vol. 116, no. 9, pp. 809 – 820

Nadolski, S., Klein, B., Kumar, A., Davaanyam, Z., 2014. An energy benchmarking model for mineral comminution. Minerals Engineering, vol. 65, pp. 178 – 186

Napier-Munn, T., 2015. Is progress in energy-efficient comminution doomed?. Minerals Engineering, vol. 73, pp. 1 – 6

Nieuwland, D.A. (Eds.), 2003. New insights into structural interpretation and modelling. Geological Society of London

Olofsson, S.O., 1988. Applied explosives technology for construction and mining. Applex Publisher, Arla, Sweden

Onur, A., Dowd, P., 1993. Open pit optimisation – Part 2: Production scheduling and inclusion of roadways. Transactions of the Institution of Mining and Metallurgy, Section A: Mining Industry, vol. 102, pp. A105 – A113

Osanloo, M., Gholamnejad, J., Karimi, B., 2008. Long-term open pit mine production planning: a review of models and algorithms. International Journal of Mining Reclamation Environment, vol. 22, no. 1, pp. 3 – 35

Palangio, T.W., Palangio, T.C., Maerz, N., 2005. Advanced automatic optical blast fragmentation sizing and tracking. Brighton Conference Proceedings, European Federation of Explosives Engineers, pp. 259 – 267

Paley, N., Kojovic, T., 2001. Adjusting blasting to increase SAG mill throughput at the Red Dog Mine. Proceedings of the 27th Annual Conference on Explosives and Blasting Research, Orlando, Florida

Pande, J., Kumar, V., Singh, A., 2015 Electronic SCB delay detonator system for mines. International Journal of Electrical, Electronics and Data Communication (IJEEDC), no. 1 (Special Issue), pp. 131 – 135

Pasieka, A.R., Sotirow, G.V., 1985. Planning and operational cut-off grades based on computerised net present value and net cash flow. CIM Bulletin, vol. 78, no. 878, pp. 47 – 54

Persson, P.A., Holmberg, R., Lee, J., 1993. Rock blasting and explosives engineering. CRC press

Powell, M.S., Weerasekara, N.S., Cole, S., LaRoche, R.D., Favier, J., 2011. DEM modelling of liner evolution and its influence on grinding rate in ball mills. Minerals Engineering, vol. 24, no. 3 – 4, pp. 341 – 351

Radziszewski, P., 2013. Energy recovery potential in comminution processes. Minerals Engineering, vol. 46, pp. 83 – 88

Ramazan, S., Dimitrakopoulos, R., 2013. Production scheduling with uncertain supply: a new solution to the open pit mining problem. Optimization and Engineering, vol. 14, no. 2, pp. 361 – 380

Ravenscroft, P.J., 1992. Risk analysis for mine scheduling by conditional simulation. Transactions of the Institution of Mining and Metallurgy, Section A: Mining Industry, vol. 101, pp. A104 – A108

Refsgaard, J.C., van der Sluijs, J.P., Højberg, A.L., Vanrolleghem, P.A., 2007. Uncertainty in the environmental modelling process – A framework and guidance. Environmental Modelling and Software, vol. 22, no. 11, pp. 1543 – 1556

Rendu, J.-M., 2008. An introduction to cut-off grade estimation. Society for Mining, Metallurgy and Exploration, Littleton, Colorado, USA

Rosario, P.P., 2010. Comminution circuit design and simulation for the development of a novel high-pressure grinding roll circuit. Doctoral Dissertation, University of British Columbia

Rosario, P.P., Boyd, K., Grundy, M., 2009. Recent trends in the design of comminution circuits for high tonnage hard rock mining. Recent Advances in Mineral Processing Plant Design, pp. 347 – 355

Rose, D., Meadows, D.G., Westendorf, M., 2015. Increasing SAG mill capacity at the Copper Mountain mine through the addition of a pre-crushing circuit. SME Annual Conference, Denver, USA, 15 – 18 February 2015

Rossi, M., Van Brunt, B. 1997. Optimizing conditionally simulated orebodies with Whittle 4.0. Optimising with Whittle, pp. 119 – 128, Whittle Programming Pty Ltd, Melbourne

Roth, I.F., Ambs, L.L., 2004. Incorporating externalities into a full cost approach to electric power generation life-cycle costing. Energy, vol. 29, no. 12 – 15, pp. 2125 – 2144

Saramak, D., 2011. The influence of chosen ore properties on efficiency of HPGR-based grinding circuits. Gospodarka Surowcami Mineralnymi, vol. 27, pp. 33 – 44

SimSched, 2014. SimSched Pit Optimizer Instruction Manual. MiningMath

Singer, D., Menzie, W.D., 2010. Quantitative mineral resource assessments: An integrated approach. Oxford University Press

SME, 2005. Exploring opportunities: Careers in the Mineral Industry. Society for Mining, Metallurgy and Exploration

Soanes, C., Hawker, S. (Eds.), 2005. Compact Oxford English Dictionary. Oxford University Press, United Kingdom

Stewart, R.M., Kennedy, B.A., 1971. The role of slope stability in the economics, design, and operation of open pit mines. Proceedings of the 1st Symposium on Stability in Open pit Mining, Vancouver 1970, pp. 5 – 21, AIME, New York

Torries, T.F., 1998. Evaluating mineral projects. Society for Mining, Metallurgy and Exploration, Littleton, USA

Tromans, D., 2008. Mineral comminution: energy efficiency considerations. Minerals Engineering, vol. 21, no. 8, pp. 613 – 620

Tromans, D., Meech, J.A., 2010. A fundamental analysis of fracture mechanics of minerals during comminution. Research Report, The Centre for Environmental Research in Minerals, Metals and Materials (CERM3), University of British Columbia, Canada

Valery, W., Jankovic, A., 2002. The future of comminution. Proceedings of the 34th International Conference on Mining and Metallurgy, pp. 287 – 298, Bor Lake, Yugoslavia, October 2002

von Michaelis, H., 2005. Real and potential metallurgical benefits of HPGR in hardrock ore processing. Proceedings of the Randol Innovative Metallurgy Forum, pp. 31–39, Perth, W.A., Australia, 21 – 24 August 2005

Wang, C., Nadolski, S., Mejia, O., Drozdiak, J., Klein, B., 2013. Energy and cost comparisons of HPGR based circuits with the SABC circuit installed at the huckleberry mine. 45th Annual Canadian Mineral Processors Operators Conference, Ottawa, Ontario

Whittle, J., Rozman, L., 1991. Open pit design in 90s. Proceedings of the Mining Industry Optimisation Conference, pp. 13 – 19, Sydney

Whittle, J., 1988. Beyond optimisation in open pit design. Canadian Conference on Computer Applications in the Mineral Industries, pp. 331 – 337, Balkema, Rotterdam

Whittle, J., 1997. Optimization in mine design. WH Bryan Mining Geology Research Centre, Brisbane, Australia

Whittle, J., 1999. A decade of open pit mine planning and optimization – The craft of turning algorithms into packages. 28th Proceedings of the Application of Computers and Operations Research in the Mineral Industry (APCOM), pp. 15 – 23, Colorado School of Mines, Golden, Colorado, USA

Wills, B.A., Finch, J.A., 2015. Wills' mineral processing technology: An introduction to the practical aspects of ore treatment and mineral recovery, 8th Edition. Butterworth-Heinemann, United Kingdom, ISBN 978-0-08-097053-0

Workman, L., Eloranta, J., 2003. The effects of blasting on crushing and grinding efficiency and energy consumption. Proceedings of the 29th Conference on Explosives and Blasting Techniques, pp. 1 – 5, International Society of Explosive Engineers, Cleveland, Ohio, USA

Zhao, Y., Kim, Y.C, 1992. A new optimum pit limit design algorithm. Proceedings of the 23rd Application of Computers and Operations Research in the Mineral Industry (APCOM), pp. 423 – 434