Comparison of the Business Case between Explosives and Non-Explosives Mining Methods for Underground Platinum Mining Projects

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Declaration

I declare that this research report is my own work, except in those areas where it is clearly stated. It is being submitted for the degree of master’s in engineering at the University of Witwatersrand. It was not submitted before for any degree or examination at any other University

Signed……Raju…………….

Date……10 April 2021……..

Abstract

The scope of this study is to detail a methodology that could be adopted to determine the economic value of introducing non-explosive mining technology in deep level platinum underground mines. In so doing this study also illustrates a new and novel approach to investment risk analysis and reduction when considering investment in new technology mining projects. The study will highlight the complexity of building financial models for mining projects and the strong link between financial modelling and mine design, scheduling and planning. It also highlights the value that simulation can add to the process of risk reduction early in the mine planning process ultimately leading to improved investment decision making when considering new technology for rapid access and rapid mining systems.

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List of Terms

Build-up head grade – The total 4E grams produced from the concentrating process from concentrated metallics (where applicable) and tailings divided by total tons milled

Capital Expenditure – Total capital expenditure on mining and non-mining property, plant and equipment and capital work in progress

Chair-Lift Decline- An excavation parallel to the reef plane, equipped with a chairlift for conveying people (endless rope with chairs attached)

Conventional Mining – The use of drill and blasting methods to dislodge the rock from the face

Conveyor Belt Decline – Declined excavation normally parallel to the reef plane, equipped with a conveyor belt for rock handling

Development – Any tunnelling operation that has for its objective either exploration or exploitation or both

Dilution – The dilution refers to the lowering of the reef grade due to the addition of rock with a lower grade or loss of content through various possible mechanisms

Equipping - The preparation of a ledged out area to allow the extraction of the remaining reef area through a process called stoping

Hard Rock Cutting – Use of mechanical methods to dislodge the rock from the mining face

In-situ reserves – refers to the total amount of ore occurring in the natural state, while in situ minable reserves is the amount of ore that is economically minable and is controlled by factors such as the grade, thickness and accessibility of the ordebody

Ledging – The widening of tunnels (normally in the reef plane) to install equipment necessary for reef extraction (stoping)

Millhead grade – is the grade of the ore which, has arrived at the mill from the mine, and is generally lower than the sampled grade due to dilution by waste rock

Mine call factor – is the ratio between the mineral recovered, plus residue, and the anticipated recovery and residue arrived at by the mines measuring method

Off-Reef – All development that is either done in the footwall waste or hanging wall waste does not have direct access to paying ore

On-reef- This is development that is done on the reef plane thereby ensuring immediate access to the reef or paying ore

Ore- A natural mineral bearing substance of economic interest

Ore-reserve- Is an estimate of the in-situ tonnage and specific mineral content of all ore adequately exposed for mining and valued as at the date of computation

Ore Stoping - is the process of extracting the desired ore or other mineral from an underground mine, leaving behind an open space known as a stope

Recoverable grade – is the millhead grade less metallurgical losses

Reef Width – The sum of the widths of the individual reef bands within the reef body

Sampled grade – is the grade of the ore in place as determined by underground, surface or drill hole sampling

Secondary Development – Development carried out within the area delineated by primary development

Sinking Shaft- A vertical access from surface to below the reef plane in order to allow the entry of men, material and ventilation to the underground workings. The process when this access is developed is a sinking shaft

Stoping – Operations directly associated with the extraction of reef

Vertical Shaft Boring – The use of mechanical methods to drill big diameter vertical holes into the earth's surface in order to access the ore body

List of Acronyms

CFM – Commodities Factor Model - comprises annual forecasts leading to the long-term price (LTP) assumptions as appropriate for Ore Reserve declaration

PGM - Platinum group metal, six element metals of the platinum group nearly always found in association with one another. These metals are platinum, palladium, rhodium, ruthenium, iridium and osmium

SIB – Stay-in-Business – Ongoing capital investments required to sustain existing operations and infrastructure. SIB capital is applicable for all capital equipment replacement.

UG2 – A chrome reef in the Bushveld sequence often containing economic value of PGM's

4E – Four elements. The grade that is always measured as the combined content of the four most valuable precious metals, platinum, palladium, rhodium and gold.

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Chapter 1 - Introduction

1.1 Background

The commodity landscape is in constant flux with, price volatility, geopolitical turmoil, rising costs, declining ore grades and a general lack of financing (Bliss,2018). The transformative potential of technology in mining production systems is widely recognized, even while the precise configuration and extent of the possible transformation remain largely untested (Martin et al,2017). Trends towards higher levels of mechanisation promise greater speed and precision of production as well as reduced exposure to dangerous tasks for employees. New production technologies could help overcome the stagnant productivity of recent decades and make way for more value-added activity (Martin et al, 2017). Development and introduction of new and innovative mining techniques and equipment will improve efficiencies and help mining companies remain competitive in these tough economic conditions (Elliott,2015).

In hard rock, metalliferous underground mining, explosive or drill and blast is the most commonly used mining and development method (Voigt,2016). The drill and blast mining cycle consist essentially of three sets of series-linked processes as depicted in figure 1 below: (i) drilling, charging, blasting, ventilating, (ii) mucking (iii) scaling and installation of preliminary ground support (Fernberg,2007). The broken rock is then transported from the face to the main ore handling system via locomotives or Load Haul and Dump trucks (Schmidt,1976). Mining in this manner is part of a batch process on the other hand, non-explosive or rock cutting for underground development and mining purposes comprises the excavation of rock using various forms of cutting technology such as picks, disc cutters, toothed roller cutters or button cutters. These types of mining methods are continuous, leading to process and efficiency improvements (Rupprecht,2006). The explosive type of mining method is well entrenched in the South African underground hard rock mines, while the non-explosive type of mining was mostly being used in the soft rock mines such as coal mining (Van den Berg, 2014).

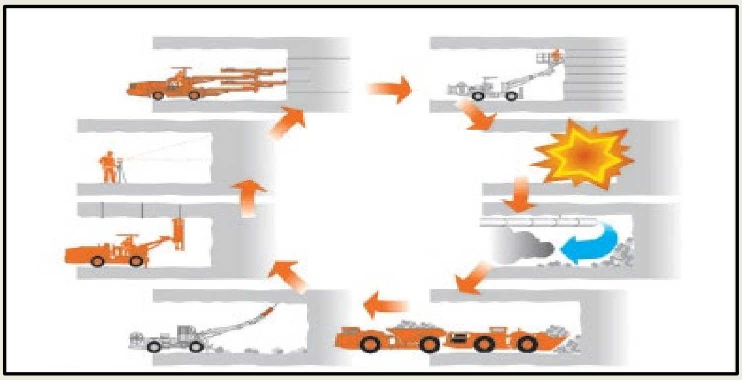

Figure 1 - Drill and Blast Cycle (Rock Excavation Handbook, pg 216)

This research report details a proposed methodology, that can be adopted in determining the economic value of introducing non-explosive mining technologies in deep level platinum underground mines.

Figure 2 - Hard Rock Cutting Machine (Sandvik, 2016)

The approach adopted in this research report, was to simulate a fully conventional drill and blast mine and develop a business case which will be used as the basis of the study. Multiple scenarios were then run in SimMine to verify if there is any value add of hard rock cutting technology as illustrated in figure 2, for underground hard rock narrow tabular mining. The scenarios were modelled so as to incrementally introduce cutting technology to develop and mine the orebody, (footwall developed mine infrastructure, the mine infrastructure such as haulages, ventilation etc is developed in the footwall at approximates 50m below the stoping levels) this included the following areas of the mine: Shaft sinking, Stations and project level development (before declines), Decline barrels, Working cost development (haulages), Stoping (reef removal) using (Conventional drill and blast, Ultra Low Profile and slot boring using a slot boring machine).

On-reef layouts were also designed, with the infrastructure, such as the declines, haulages and ventilation tunnels all developed on the reef plane. There was very limited waste development before stoping started. An on reef pre-developed infrastructure and logistical system based on the Ultra-Low-Profile mining layout was also modelled in this study. In this layout, scenarios were run to show the value of using a mix of cutting technology suites in different areas of the mine and to consider the two dominant types of infrastructure layouts used in narrow tabular mining operations.

Strategic long term mine planning deals with those components and decisions that largely impact the value of the business over the life of the mine (LoM). The core of this process is the development of a business model and plan to maximize the NPV of the company by the extraction of the entire mineral resource available to the organization (Newman et al,2010). This can be achieved through consideration and optimization of the key elements of the mining value chain: exploration strategy, extraction method, mining sequence, cut-off grade/pay limit, scale of operations, metallurgical processing route, social and labour plans, environmental and sustainable development philosophy, and marketing/sales strategy (Hall,2015). The effectiveness of the selected strategy and business model, which is defined at the beginning of any business cycle, should be periodically reviewed as internal and external aspects of the business environment are continuously changing (Hall,2015). In operating mines, the scope of strategic mine planning is related to the continuous revision of medium- and long-term business plans (replacement and expansion projects) to ensure that short term operations are consistent with long term strategic objectives (Smith,2011). The strategic mine plan is thus necessary to provide consistent direction but also to provide a base against which to assess new extraction options and projects (Smith,2011).

1.2 Purpose of the Study

The purpose of this study was to demonstrate that although non-explosive mining methods have not been fully implemented on mining operations in South Africa, resulting in the lack of credible operational benchmarks, investors who wish to invest capital in future new technology, in PGM mining projects can develop financial models to show the positive change in financial value add of using rock cutting technology for mining hard rock ore-bodies.

1.3 Research Background/Context

In response to the global drive by major mining companies to achieve a more sustainable approach to mining, economically, socially and environmentally friendly (Nelson,2013), these companies embarked on programs that would enable them to be prepared for the fourth industrial revolution (4IR). These programs were envisaged to bring together several value creating initiatives that use innovation and/or technology as an enabler to deliver greater value from a company's broader mining activity (Kenyon,2014). It would also position mining companies to be at the leading edge of technology development and innovation in mining and create a holistic approach on how technology can benefit entire mining systems, not just parts of them (Rafie,2012). These would provide an open and collaborative approach to bring benefits to the mining industry. A key aspect of these initiatives was the development of continuous mining machines that are devoid of blasting cycles, the so-called rock factories (Robbins,1997).

1.4 Research Motivation

The motivation for this research is based on research that 60% of the current world PGM production originates from conventional explosives mines, with conventional layouts and labour intensive mining methods which gives investors and shareholders alike, the assurance that the marginal financial returns are creditably more bankable, than unproven non-explosive mining methods, (Van den Berg,2014).

However, conventional platinum mining is labour-intensive, unsafe and uneconomical due to low productivity levels, increasing labour costs and current weak metal prices (Rabie,2012). Alternative mining methods and mine designs are being sought in order to extract more value from the PGM Orebody. Investors are reluctant to invest future funds in PGM mines that employ conventional mine designs and practices but on the other hand are also cautious about investing in new mining technology. If this persists the PGM, chrome and even gold mining industry in South Africa will slowly die (Voigt,2016).

By conducting this study, the researcher intends to give present and future mining project investors the ability to further quantify the risks associated with the implementation of modernized non-explosive mining methods and to also optimize mining layouts to set up these operations for future success. In so doing this would assist the underground mining sector move away from this costly, and unsafe conventional handheld drill and blast mining methods, and to increase Foreign Direct Investment into these South African mining sectors.

1.5 Problem Statement

A mining project will not get investment capital if it does not show a positive return from the funds invested in the development of the mining operation (Smith,2011). One of the most common methods to evaluate a mining project is the DCF method (Mirakovski, et al,2009). However, in the case of new mining technology the requirement for actual production input data, into the financial models are problematic to acquire as these mining systems have not been tried and tested over prolonged periods of time at any given underground mining operation (Rupprecht,2006).

Major mining companies are still adopting there internally developed techno-economic methodologies to quantify key investment indicators, such as NPV and IRR. This investment modelling methodologies were developed to only consider conventional drill and blast mining methods, which leads to those project investments that apply modernized mining methods being totally non-existent or very limited (Kenyon,2014). Rapid advances in technological innovation, including through automation, digitization and electrification, are having a fundamental impact on the mining sector, but these are still not considered as options in bankable feasibility studies (Bliss,2018).

1.6 Research Question

What is the financial benefit for a Platinum Group Metal mining company in South Africa to implement non-explosive (hard-rock cutting) mining methods compared to an explosive (conventional handheld drill and blast) mining method?

1.7 Research Objectives

The objectives of this research report are to:

1. Compare through simulation the financial benefits that non-explosive mining methods have over explosive mining methods:
2. Determine the key value drivers for PGM Mining Projects in South Africa, and the influence these have on the techno-economic modelling for non- explosive vs explosive mining methods:
3. Analyze various financial evaluation techniques used to develop underground mining project business cases to demonstrate the complexity in developing these models.
4. Develop a methodology that can be used to compile bankable feasibility studies that use new and innovative mining methods for underground mining endeavors.

1.8 Assumptions used for the Study

It is assumed that the experts are familiar with the basic decision-making process for investment appraisals. While most of the proposed respondents have some understanding of the concepts, the quality of their feedback depends on their level of understanding of the statistical and financial concepts used to calculate the key investment appraisal indicators such as NPV,IRR, ARR. Expert opinion is used to determine the probability distribution of the input parameters. Where no proxy or experimental means exist, experts may have to rely on judgement, hunch or intuition. It is assumed that expert opinion is correct and reliable. Certain experts may not take the review of the outcomes seriously and the results from there feedback may not reflect the correct trends, as experts may prefer established conventional mining methods and not be open to new ideas in the mining sector.

Some of the other assumptions made during this study also included the nature of the orebody studied and the extraction methodologies adopted. It is assumed that the configuration of the underground PGM orebody referred to in this study is consistent throughout this mining sector in South Africa and that the geological and other losses are consistent with other locations and mining methods. Although there are many different mining methods that could be adopted to exploit this type of orebody this study was limited to conventional handheld drill and blast, mechanized, XLP/ULP and hard rock cutting.

1.9 Ethics Clearance

It is important that research which involves gathering data from people (human participants) treats them fairly and meets ethical standards. Broadly speaking this involves full disclosure (telling them about the research), non-coercion, and privacy issues. This research report entailed accessing various company information and the use of expert's opinions for verification of the methodology outcomes.

1.10 Outline of Chapters

Chapter 1 sets the scene with introductory notes and assessments of non- explosive mining methods and to explain conventional explosive mining methods.

In Chapter 2 of this research report a Literature review of the impact of mechanization and automation in the mining industry, particularly in the gold and platinum sector is undertaken. The review also gives a view of the different financial decision-making indicators that is generally used in the mining industry. The review also included the various hard rock cutting machines developed and deployed in the hard rock mining sector.

In Chapter 3 the author covers the methodology and research strategy adopted in this report. The research design that is used to highlight the need for this type of risk evaluation and mitigation process in the mining industry. This chapter also covers the setup and execution of the simulation instrument to verify and validate the need for this type of thinking around new technology projects in mining.

Chapter 4 outlines the research simulation findings and analysis of these findings. It also gives the reader a better understanding of the cost and other inputs into the various simulation models that was developed.

In Chapter 5 the author discusses the results from the various options studied in order to demonstrate the complexity of developing business cases for new technology mining projects. The various models run during the study showed that there is a strong business case to implement non-explosive mining methods as compared to drill and blast. A detail analysis of these outcomes is discussed in this chapter.

In chapter 6 the author focused on concluding the study work by focusing on answering the research question. The various objectives of the study were also concluded to demonstrate the validity of the original research question and objectives. Various recommendations were also made regarding the outcomes from this study.

Chapter 2 - Literature Review

2.1. Preamble

The literature review of this study gives an account of the various aspects that need to be considered for the financial evaluation of underground mining projects. It begins with an overview of the geological, grade profile, resource to reserve calculations and finally the requirements to build up the production profile of an underground mine. The literature review also considers rock cutting technology and the equipment used for rapid vertical shaft sinking, rapid horizontal development and stoping operations. These provide a basis for the considerations taken in this study.

2.2 Introduction

Blasting is the main method of rock breaking for the purposes of mining. When used in an underground narrow tabular operation, blasting is a batch process (Voigt, 2016). If rock can be excavated by cutting it, rather than by blasting, underground mining can become continuous, leading to process and efficiency improvements which ultimately leads to a positive financial return on investments (Voigt, 2016).

Platinum Group Metals (PGMs) are recovered through both underground and open pit mining from sulphide ores containing PGMs, Nickel, Copper and other metals. Platinum (Pt), Palladium (Pd), Rhodium (Rh) and Ruthenium (Ru) are the most prevalent of the six PGMs, with Iridium (Ir), and Osmium (Os) found in much smaller quantities and referred to as minor PGM's (Rudenno,2012). Gold is often associated with PGM deposits and treated as part of a family -with Platinum, Palladium and Rhodium -collectively known as "4E". Nickel and Copper on the other hand are the most prevalent base metals, with Chromite (used to derive Chrome) being another significant byproduct. Silver and Cobalt are also found in trace quantities (Lurie,2004). World PGM reserves total about 2.122 M oz (160 years at the current rate of production) and are found in relatively few areas of the world; Southern Africa (95%), Russia (1.7%), North America (1.8%), and Rest of World (1.5%) (Lurie,2004).

South Africa (SA) is the source of over 40% of newly mined Palladium and 75% for Platinum (Johnson Matthey,2008). PGM mining in South Africa is predominantly executed in the Bushveld Complex in three regions commonly referred to as the Western, Eastern, and Northern Limbs. In Zimbabwe PGM mining occurs in the Great Dyke and the reef is referred to as MSZ (Main sulphide zone). The Bushveld Complex includes three distinct mineral-bearing strata known as reefs. The Merensky Reef, the source of most current SA production, contains relatively higher PGM grades and ratios of Platinum (vs. Palladium) (Lurie,2004)

The UG2 Reef is a consistent chromite-rich reef with higher Rhodium content but lacks Merensky's high yield of Gold, Copper and Nickel byproducts. UG2 ores may contain as much as twice the PGM reserves as Merensky and Platreefs. Platreef is a wider reef with lower PGM values, but higher base metal content (Rudenno,2012).

2.3 Underground Mining

In underground hard rock mines, orebodies may take the form of strata-bound layers, cross-cutting veins and stockworks. Orebodies are commonly displaced by faults and shear zones, dykes, potholes, replacement pegmatoid zones and sills. The type of orebody and the nature of the host rock are among the factors that determine the method of mining to be adopted (Lurie,2004, pg. 80). The selection of the most appropriate mining method for underground mining operations is reliant on many variables such as the shape of the orebody (that is tabular, cylindrical, spherical), orientation of the orebody (sub-horizontal, sub-vertical), the continuity of the orebody, size of the orebody, ore grade (high grade, low-grade), distribution of the ore-bearing minerals within the orebody (massive or disseminated with a cut-off grade), depth of the overburden, strength of the orebody and overburden or host rocks, capital costs (rate of financial recovery, cashflows), area or land availability for waste disposal, safety concerns, impacts on the environment and rehabilitation concerns (Harraz,2016).

2.3.1 Evaluation of mineral Deposits

The realistic evaluation of mineral occurrences demands expertise in many fields since its value is dependent on the cost of mining, treating the ore and marketing the final product (Lurie,2004). The study area chosen was situated approximately 30km north west of the town of Rustenburg in the North West Province of the Republic of South Africa and occurs within the 2060-million-year-old Bushveld Complex, which covers over 65 000 km2 of the four most northern provinces of the Republic of South Africa. The Bushveld Complex is divided into five discrete compartments termed "limbs". It comprises of the intrusive Rustenburg Layered Suite (RLS) and a series of granites (Bushveld Granites) and extrusive rocks. The RLS is the world's largest known mafic igneous layered intrusion containing 90% of the world's known PGE's.

Only the RLS occurs and within this area and specifically underlies the study investment area. Figure 3 depicts the target area in relation to the five limbs of the RLS in a regional context. The Study area lies within the western limb which stretches in an arc westward from the vicinity of Pretoria (Tshwane) through Rustenburg and then northwards in the direction of Northam. The Pilanesberg Complex, the remnant of an alkaline volcanic plug, which intruded into the Bushveld Complex about 1,250 million years ago, splits the Western Limb into two limbs (north-western and south-western). The target area is located just south of the Pilanesberg Complex (Figure 3).

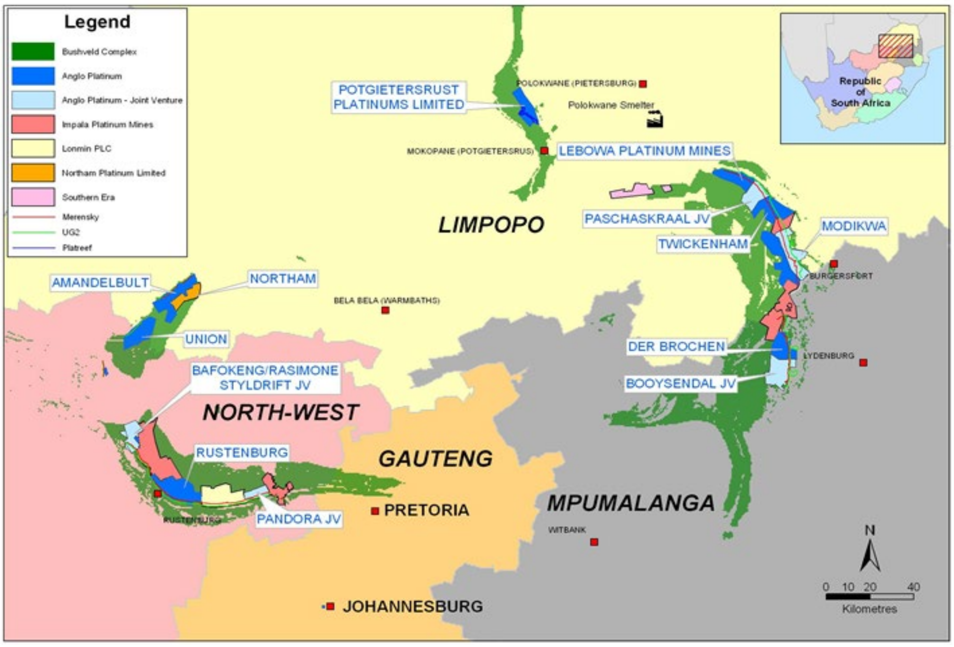

Figure 3- Location of Study Target Area

For this study a conditional simulation was completed for the accumulation variable (content) of all the facies areas. The percentage error calculations were at 90 % confidence limits per estimated block and are used to guide the resource classification. Geological loss estimates have been provided by the competent engineer and are based on a comprehensive set of database information including bench marking with the nearby mines. The 3D seismic survey has had an important impact on the resource estimate, by assisting with the delineation of some of the facies areas and their structures.

In summary, this study resource estimate has included as many of the new drill holes and assays that were available at the time of this estimate. This has resulted in a significant portion of Measured Resources which decreases the previous Indicated and Inferred Resources. The new drill holes have been instrumental in highlighting areas of risk associated with the rolling reef and the embayment of Thick Reef into the Central Reef area. Measured and Indicated Resources may be upgraded to the Reserve category with consideration of all modifying factors.

2.3.1.2 Ore Grade and Reserves

When considering the grade of an ore it must be unambiguously understood what is being referred to, In the narrow sense ore grade refers to the metal content or tenor, but in the case of an industrial mineral it may refer to the content of the mineral itself, while in the case of a commodity like coal the quality may be expressed in the term of the proximately analysis and the calorific value (Lurie,2004). An important concept about most mineral deposits is the cut-off grade and its impact on the economics of a project (Rudenno,2012, pg.329). A higher cut-off grade increases the average grade of the ore entering the plant and thus results in the higher net present value of the project (Dagdelen,2001). In determining the optimal cut-off grade, minerals must be extracted so that they maximize the net present value of the operation (Tatiya,1996).

2.3.1.3 Grade

The actual process of identifying and quantifying geological losses involves consideration of historic exposure (benchmarking with old workings as well as recent drill hole or geophysical information (Lurie,2004). Geological losses are based on historic data (actual measured potholes, faults, IRUPS and dyke losses in the mined-out areas) as well as information obtained from drill holes, mapping, seismic and aeromagnetic surveys (Dhakol,2012). Planning of the geological loss factors are carried out using the mid-year mining face positions. The results of the planning are compared to the results obtained the previous year. The various geological loss domains are then reviewed, and changes made where necessary (Earle,2015).

2.4 Mining Project Context

Mining projects are known to be complex business investment opportunities (Smith,2011). Mining projects tend to be capital intensive and usually require many years of successful operation to break even on costs incurred (Iloiu & Iloiu,2009). As an industry, mining is regarded as relatively risky compared to other industries (Martinez et el, 2012). This is primarily since value of a mining project or business case is influenced by many underlying natural, economic and physical uncertainties (Samis et al,2010). Mining projects are often long-term, and the business case attractiveness depends upon the performance of the project over the life of mine (LOM) (Bhappu et al,1995). Besides these factors' other factors such as political, environmental, social and technology also effect the decision-making process (Samis et al, 2010). When an opportunity exists to either acquire an existing mining operation or to invest capital in developing a new mine, the shareholders or the owners of the mining company want to know the value of the mine (Minnitt et al,2005). In a capital constrained environment and increasing uncertainties around commodity prices, the investment decisions become far trickier and vital as companies try to balance between long-term sustainability and short-term financial goals (Martinez et al,2012).

2.5 Mining Project Evaluation Techniques

The use of a robust business case evaluation technique becomes extremely important when evaluating mining investments due to the inherently high risks and uncertainties involved (Botin et al,2011). There are many business case evaluation models that are available and are used at different levels in various mining companies (Runge,2011). An ideal mining project business case evaluation method should be able to provide decision makers with answers relating to timing of the capital investment and the capacity of the mining operation that is its optimum production profile for the LOM (Mirakovski et al, 2009).

Iloiu & Iloiu, (2009) reviewed four evaluation techniques that are popular in the mining industry i.e. Net Present Value (NPV), Internal Rate of Return (IRR) under the Discounted Cashflow Method (DCF); Decision Tree Method (DTM); and Real Option Valuation (ROV). The DCF methods do not allow for flexibility in management decision making as all the input parameters are decided for the rest of the LOM while modelling the business case. This in turn excludes managerial decision making in the future relating to changing environments that may change the input parameters (Iloiu & Iloiu,2009). The DTM scores above DCF in terms of allowing flexibility to include different managerial strategies and calculate all the expected NPV (Iloiu & Iloiu,2009). This provides the whole picture of the project; however, this method becomes cumbersome when the number of variables increases.

On the other hand, simulation methods are not standalone methods and it is used along with DCF or DTM. It tries to model the uncertainties by assigning statistical distributions to the variables or the project parameters. This provides the decision makers with many possible outcomes of the project (Iloiu & Iloiu,2009). The ROV method's strength is mainly in its flexibility to accommodate managerial decisions at any given point in time. This allows decision makers to respond to future events in a way that increases return. However, the need for and the value of flexibility depends on the uncertainty of a company about future reinvestment needs (Damodaran,2005). Firms with predictable reinvestment needs value flexibility less than firms whose investment needs are volatile.

There are many other valuation methods e.g. differential discounting, tail margin, Black- Scholes option pricing, Farm-in analysis and others (Lilford et al,2005). Because the focus of the literature review was on evaluation methods specifically for new mining projects, discussions on the above-mentioned methods have been excluded as they are primarily used for stock valuation or running asset valuation (Arnold,2002). Lilford and Minnitt (2005) conclude that selection of the valuation method depends on the project teams' ability to correctly interpret all the available information on the input variables e.g. commodity price, exchange rate, technical information, economic information and other risks, and from these select the appropriate valuation method (Nelson,2011).

2.6 DCF with NPV and IRR as a Basis for Valuation

To date the most commonly used business case evaluation technique is DCF method with NPV and IRR measures for mining related investments (Mirakovski et al,2014). Bhappu et al, (1995) did a survey and found that some 95% of the surveyed mining companies use some sort of DCF technique in their decision making with NPV and IRR as the key criteria. A parameter such as payback period was also used as secondary criteria in decision making. DCF evaluation models has specific strengths over other methods used in the industry that is its ability to consider royalties, leases, taxation and financial gearing on the resulting cashflow (Lilford et al,2005).

DCF, in addition, can model unredeemed capital balances, losses, deprecation etc. on the free cashflow. DCF is suitable to be used to evaluate a mining project in a stable macroeconomic environment (Lilford et al,2005). NPV is a standard method that calculates the net income flow of a mining asset over its entire economic life (Domingo et al,2007). It involves forecasting the net cashflow the mining project would generate if exploited optimally, then discounting it to the current date with appropriate cost of capital or discount rate (Mirakovski et al,2014). The NPV methodology is aligned to the concept of maximizing the value of a firm by choosing the right project with maximum NPV. Domingo & Lopez-Dee,(2007) describe the four key elements of NPV as, Time value of money, NPV recognizes the concept that a rand earned today is worth more than a rand earned a year from now; income flows, NPV calculates the projects expected income flows and eliminates accounting inconsistencies as the income inflows consists of all the cash flows and not just profits; risks, NPV can incorporate the risks associated with the project in terms of the expected income flow and discount rate (Arnold,2002); flexibility, NPV method provides flexibility as it can adjust for inflation and be used with other analytical tools such as scenario analysis.

The Internal Rate of return method determines the interest rate (the average rate of return) at which the present value of future cash flows is equal to the cost of the investment now (Arnold,2002). This basically means the higher a project's IRR, the more attractive the project is. Along with NPV, IRR is also widely used to rank several prospective project investments that a firm is considering making an investment in (Bhappu et al,1995). Assuming all other factors are equal among the various projects, the project with the highest IRR would probably be considered the best.

Despite many benefits of the NPV like its ability to consider absolute amounts of wealth change and the handling of non-conventional cash-flows (Arnold,2002), it has its own limitations. The two main inputs that NPV analysis has, are net income flow and discount rate. Determining the net income flow is not easy for mining projects as it involves many assumptions and expectations underpinning the income flow (Domingo et al,2007). Guj and Garzon, (2007) argue that for resource projects with low cashflow and healthy net present value, the NPV method will continue to be an appropriate method for project investment decisions. However, the NPV/DCF method can be biased in evaluating or comparing marginal investment cases since it often applies a single risk-adjusted discount rate to compare the value of the project with different risk factors (Samuels et al,1995).

Choosing an appropriate discount rate is crucial for assessing the value of the mining project, when using the NPV method (Botin et al,2011). At the same time an NPV obtained using suitable variable discount rates within the lifetime of the asset is more realistic than the NPV calculated using one constant risk-based discount rate (Domingo et al,2007). DCF/NPV techniques do not account for the ability of the managers to take advantage of changing situations in the future of the project (Bhappu et al,1995). Despite the above-mentioned drawbacks in the NPV method, it is still a powerful tool and is still widely used in evaluation and comparing of mining projects. There are many other tools e.g. Modern Asset Pricing (MAP) which in certain cases provide a better evaluation but they still use DCF analysis as the basis and only compliment it further (Guj et al,2007). Ellis, (1995) states that despite the exact answer to the real value of a project or mineral property not being fully determinable from the DCF/NPV method, it is still recommended to use NPV as the primary method of valuing an operating mine, or mineral assets, that has indicated resources and a degree of engineering, production, Capex and Opex data.

Major mining companies usually adopt a project lifecycle with go/no-go stage gates for project approvals. During the project development process the project usually progresses from Concept /Scoping phase to feasibility and then into execution, in so doing the level of accuracy of the estimate increases as depicted in figure 4 below.

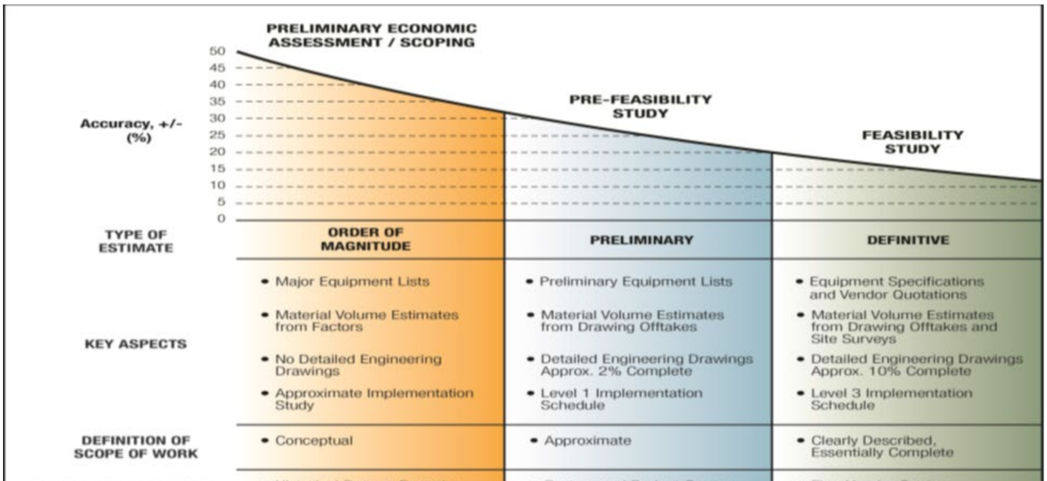

Figure 4: Summary of Level of Accuracy Required per Study Level (Rustenburg Platinum Mines

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From the Concept stage until the end of Feasibility when the final decision is made for capital approval for execution, there is a varying degree of information relating to engineering, Capex, Opex and production (Figure 4). Many major mining houses use NPV as the key criteria along with IRR as the secondary criteria in comparing and deciding on project capital investments (Park et al,2004). The Capex estimate accuracy level depends on the degree of engineering progress at various stages (Table 1).

Table 1: Capex Accuracy in Mining Projects (Nixon and Hampton, 2016)

Phase of the Project	Class	Classification of Accuracy
Concept Phase	0	Accuracy range of +40% to -25% before contingency is considered. Capital cost estimates need to support the business case and a decision to proceed to a pre-feasibility stage.
Pre-Feasibility Phase	1	Accuracy range of +25% to -15% before contingency is considered. Capital cost estimetes of accuracy needed to support the selection of a single option and a decision to proceed to feasibility stage.
Feasibility Phase	2	Accuracy range of +15% to -5% before contingency is considered. Capital cost estimates sufficiently detailed and accurate to support project funding and control.

Mining projects are inherently risky (Martinez et al,2012) as they are influenced by several underlying natural, economic and operational/technical uncertainties over the Life of Mine (Figure 5). These are not known with much certainty during the project evaluation stage.

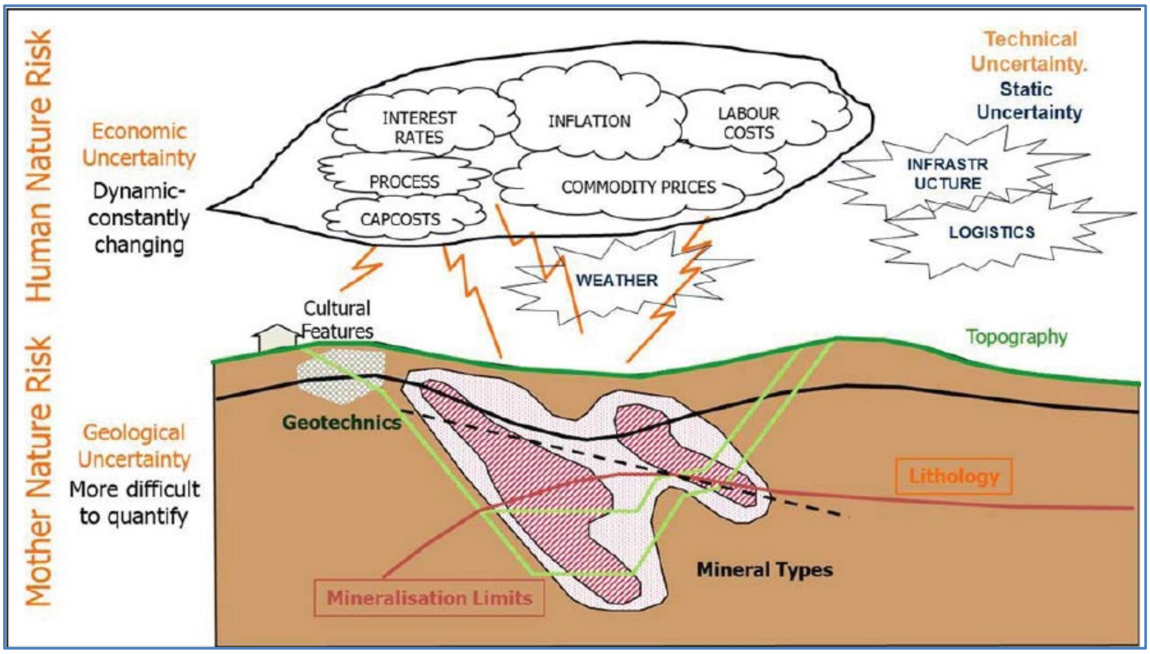

Figure 5: Source of Uncertainties in Mining Projects (Martinez et al,2012)

The natural risks come from geology that is distribution of ore underground as well as the ore grade, (Jarman,2012). This is because during the investment decision making process, the sampling obtained is not fully representative of the full orebody. Martinez et al, (2012) argued that, economic uncertainties come from a variety of factors that is, capital expenditure, operational expenditure and commodity prices. The evaluation of potential mining systems requires the calculation of several technical inputs, such as mineable reserves, production rates, recoveries, costs and revenues (Allen,1980). Analysts do not know with absolute certainty how much it is going to cost to produce a unit of the commodity. In addition, future metal price and exchange rate are always uncertain (Rudenno,2012). Metal price and exchange rate depends on many factors including demand and supply forces and speculative pricing (Martinez et al,2012).

Steady state mine production in an underground mine, will typically commence with a ramp-up period, with production starting at a lower level and building up to the planned full capacity or steady state production (Extractives hub, 2020).

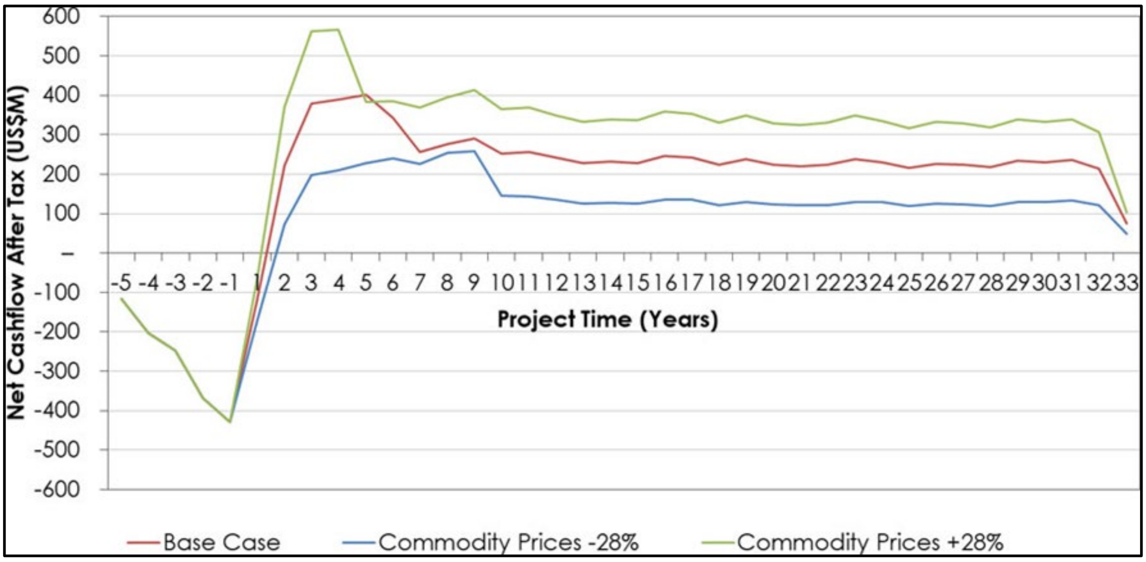

Figure 6- Ivanhoe Project Cashflow Profile

Faster development rates (figure 6) in underground mines would generate value by reducing the time from capital expenditure (development) to revenue generation (Steward et al,2006). In 1997, Macfarlane documented the relationship between higher advance rates and improved NPV for a project. Van Rooyen (1998) conducted similar exercises and found similar results.

2.6.1 Key Requirements for Mining Project Investment Decision

In general there are about eleven categories of information that must be addressed in any bankable feasibility study for mining project investments these include, project background and ownership, geology, ore reserves, reconciliation that is provision made for metallurgical accounting, mining, processing and metallurgy, infrastructure and support services, environmental baselines, permitting, development schedule, operational costs and also management and personnel issues (White,1997). For instance, (Erdem et al,2012) provide a comprehensive list of input parameters that influenced the value of a copper mine project which also included related parameters namely ore grade, copper price, process recovery, Capex, Opex and interest rates. With respect to PGM mines, the key input parameters influencing the valuation of the project in terms of NPV and IRR are most likely to be like the ones identified in the studies discussed above. However, various factors need to be determined from the existing projects data; and the associated distributions need to be established if it is to be applied into a simulation model.

2.7 Rock Cutting Technology

Mechanical excavation is an alternative method of mining as opposed to the conventional drilling and blasting method (Voigt,2016). The mechanical excavators can extract large amount of ore/rock by continuous cutting. The activity cycle for a mechanical excavator is described by (Persson and Schmidt, 1976) as follows: First gripping where the machine stabilizes itself from the impact due to the boring activity. The next process depends on the type of the mechanical excavators being used. The boring commences where extracted rock is mucked via for example a mucking apron. The apron is the part of the equipment, its purpose, is to collect the rock from the front of the machine and then to be transferred by for example a conveyor belt installed on the machine. The process may include the re-tract activity or not. Re-tract must be done for some of the excavators after every stroke/thrust of the machine by moving back the machine (here the changing of the cutter discs and inspection may be performed). The activities are repeated in the cycle until the required amount of rock mass is extracted (Persson and Schmidt 1976 and Maidl et.al,2008).

2.7.1 Different Types of Equipment Used for Mechanical Excavations

Mechanical excavators can be divided into part face and full-face rock underground equipment. Part face equipment are for instance road-headers and continuous miners and full face excavators are hard rock tunneling boring machines that can be further divided into three groups (Muirhead et al,1968): Tunnel Boring Machine (TBM) - full face shielded and shield machines (that are not falling into the TBM group but are also shielded), Raise Boring Machine (RBM) and third continuously developing group which is the mixture of both, the so-called Mobile Tunnel Miner (MTM), Modular Mining Machine (MMM) or Mobile Miner (MM) (Maidl et.al,2008). Figure 7 shows examples of the equipment used in mechanical excavation, the slot borer, Rapid Mine Development System (RMDS) and Tunnel Boring Machine.

Figure 7- Mechanical Excavation Equipment (Skawina,2013)

Depending on the geology and geo-mechanics of the ground, different types of machines are chosen. An influencing parameter is Uniaxial Compressive Stress (UCS) (Perritt,2007). For low UCS values but quite competent ground the road-headers are still the leaders in the excavation of the rock (Rexborough,1973). The mobile miner is good for hard rock conditions and is being used for narrow veins. It combines the rock cutting techniques (to some extent) from a TBM and the flexibility from the road-header (Kenyon,2014). The examples as shown in Figure 7 of the different types of mechanical excavators (Sandvik, Atlas Copco MRE Presentation 2012 and Chadwick,2010) are typical TBM's which are very good for any type of soil but very expensive and not as flexible as the other systems. Moreover, there is only one shape available (circular) excluding the special TBMs such as the Mobile Tunnel Miner (Aker Wirth GmbH), which is a special type of the TBM that can create more shapes than just circular ones (Luxner,2012).

2.7.2 Mechanical Blind Shaft Boring

Mechanical blind shaft boring machines will be used in the simulation modelling to develop both the main and service vertical shafts. Pre- Sinking of both shafts will be to a depth of 60 meters using conventional drill and blast sinking methods. Figure 8 below show the different types of blind boring machines available in the market to sink vertical shafts in the hard rock environment.

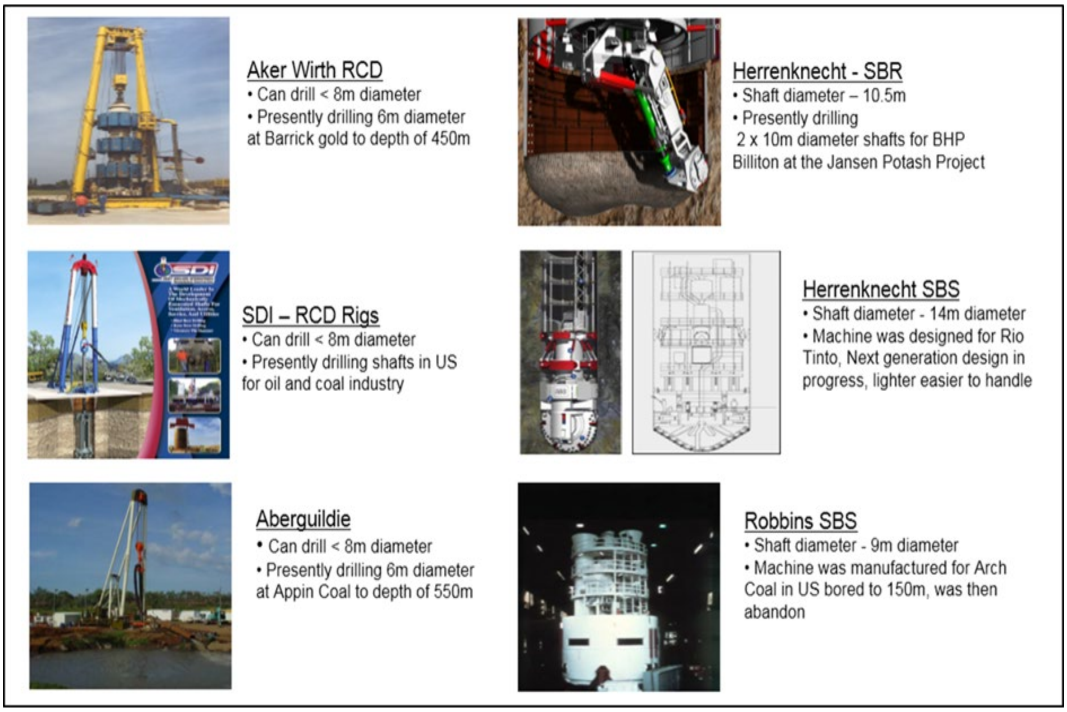

Figure 8- Vertical Blind Boring Machines (Pickering,2016)

2.7.3 Mobile Tunnel Miner (MTM)

The Mobile Tunnel Miner (MTM) is designed to cut profiles customized for underground mine layouts and mine logistics (Ramezanzadeh et al,2010). The machine was designed and manufactured to adopt the undercutting disc concept (Ramezanzadel et el,2010). The MTM is designed to cut the horizontal ends of the stations, project development, declines and working cost flat end development depending on the scenario. The various machine specifications and cut dimensions are shown in Appendix 2.

2.7.4 Rapid Mine Development System (RMDS)

The Rapid Mine Development System (RMDS) is used in the technical simulation models to cut the horizontal ends of the stations, project area, declines and working cost development ends depending on the scenario. There are two types of RMDS machines, the RMDS 4.5m x3m and the 4.5m x 2.2m. Appendix 2 show the machine specifications and cut dimensions. The RMDS is classified as a “Part Face Cutting Machine” (Lyly,2018). The cutting mechanism used in the machines allows the cutters to roll on the rock surface creating fracture points that then allow the rock to break away from the face (Lyly,2018). The force from the cutter drum is transmitted directly to the face in a perpendicular direction, (Lyly,2018). This is the difference between the RMDS and the MN220 cutting mechanisms.

2.7.5 Tunnel Boring Machine

The very first Tunnel Boring Machine (TBM) was constructed and used to excavate a tunnel below the Thames in London 1825-1842 (Backbolm et al,2004). The present generation TBM constructed as a mechanical rotating machine was developed in the 1950's (Cigla et al,2001). From the time of its construction thousands of kilometres of tunnels have been excavated worldwide in a wide range of ground conditions and in diameters ranging from around 1.6m up to 12m (Cigla et al,2001). The Tunnel Boring Machine (TBM) will be used in the technical simulation models to develop the horizontal ends of the declines depending on the cutting scenario. The parameters used for scheduling as well as the machine specification are listed and depicted in Appendix 2.

2.7.6 Slot Boring Machine

The Slot Boring Machine is used in the technical simulation models for mechanical stoping operations slotting profile is as shown in Appendix 2. The machine will only be used in on-reef development layouts. The Slot Boring Machine allows the drilling of slot holes from horizontal to +/- 20 degrees incline the maximum slot length is 20m and the holes are bored with an overlap to each other. A suction/vacuum system is used in the technical simulation models to transport the ore that has been bored. The vacuum suction unit is included with the slot borer.

2.7.7 Ultra-Low-Profile Equipment

The Ultra-Low Profile (ULP) mining method is designed for stoping widths of 80 -120 cm. The method is an on-reef mining method which requires more uniform ground conditions and limited reef rolling conditions. On steeper dipping reefs dip tramming roadways are required. The ULP mine design is based on a pre-developed panel strike section, which is mined from either side of each raise-line. The method initially begins with the panels first being ledged and then stoped from the ledging roadway, as has been done in traditional Extra Low-Profile Mining (Fourie,2014).

During which time a pre-development team is extending the ASD & pre-developing the section's raise lines and winzes. The ULP suite of equipment shown in Appendix 2 and figure 8 below, mainly includes a drill rig, dozer, roof bolter, utility vehicle, charger and sweeper. The ULP mining cycle as depicted in figure 9 below starts with dozer cleaning, roof bolting, face drilling, charge up and then blasting.

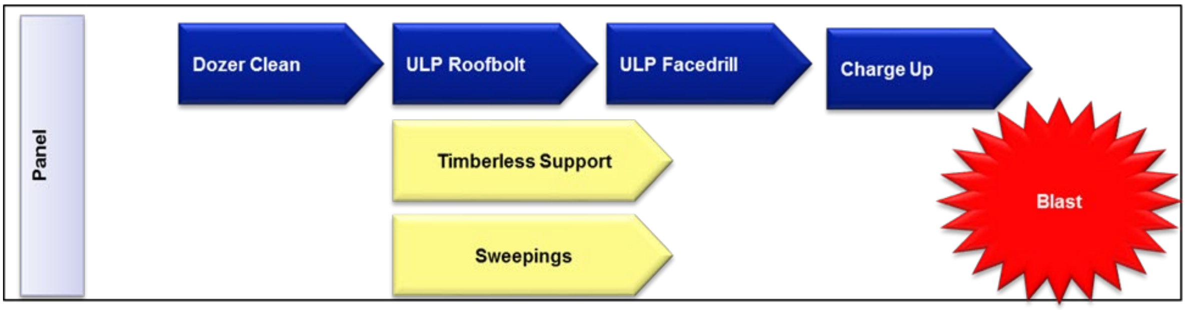

Figure 9- Ultra Low-Profile Panel Sequence (Fourie, 2014)

2.7.8 MN220 Machine

The Sandvik Reef Miner, MN220 with USU250 shown in Appendix 2, is a disc cutting mining machine with an integrated pneumatic ore removal system. The MN220 is designed for mining narrow reefs of hard and abrasive rock (Pickering et al,2006). Therefore, this machine can exceed limits of the operational range as given for other mining machines, but it is nevertheless also faced with the ultimate limits set by the machine rock interaction under extreme conditions. The Sandvik Reef Miner is electrically powered. Optimum comfort and safety for operators are provided due to fully remote control. The MN220 was designed as a combination of the principles of Sandvik roadheaders and tunnelborers, resulting in a machine with high performance rates in hard and abrasive rock conditions. The integrated pneumatic ore removal system USU250 is developed to remove the cut material directly at the face behind the cutter wheel of MN220. The machine specifications are as stated in Appendix 2.

2.9 Conclusion

As rapid advances in technological innovation, including through rapid mining systems, automation, digitization, and electrification, are having a fundamental impact on the mining sectors (Bliss,2018), mining companies, investors and governments must prepare and respond to these technological shifts in the mining sector. There is a need to ensure these key stakeholders and communities continue to achieve shared value from mining, such as financial benefits of increased efficiencies and productivity (Fraenkel,2019).

When evaluating mining investment opportunities, one should consider the risks associated with mineral exploration and development (Park et al,2004). These are commonly classified as technical, economic, and political risks, and are accounted for in the investment decision by changing the discount rate (Park et al,2004). In the next few chapters the focus will be on how to minimize the risks associated with these mining investments and demonstrate the advantages of rapid continuous mining as opposed to drill and blasting operation.

2.10 The Proposed Framework for Bankable Business Cases for Mining Projects that adopt New Mining Technology

Mining companies usually adopt a tried and test decision-making process for investment appraisal that is internal to the organization. On the other hand, outdated appraisal techniques that are set within the appraisal methodology that do not ask the right questions and which provides erroneous conclusions will destroy the wealth of shareholders (Arnold,2002). This study proposes a new framework as depicted in figure 10 below for the compilation of a bankable business cases when new technology is included in the mine design.

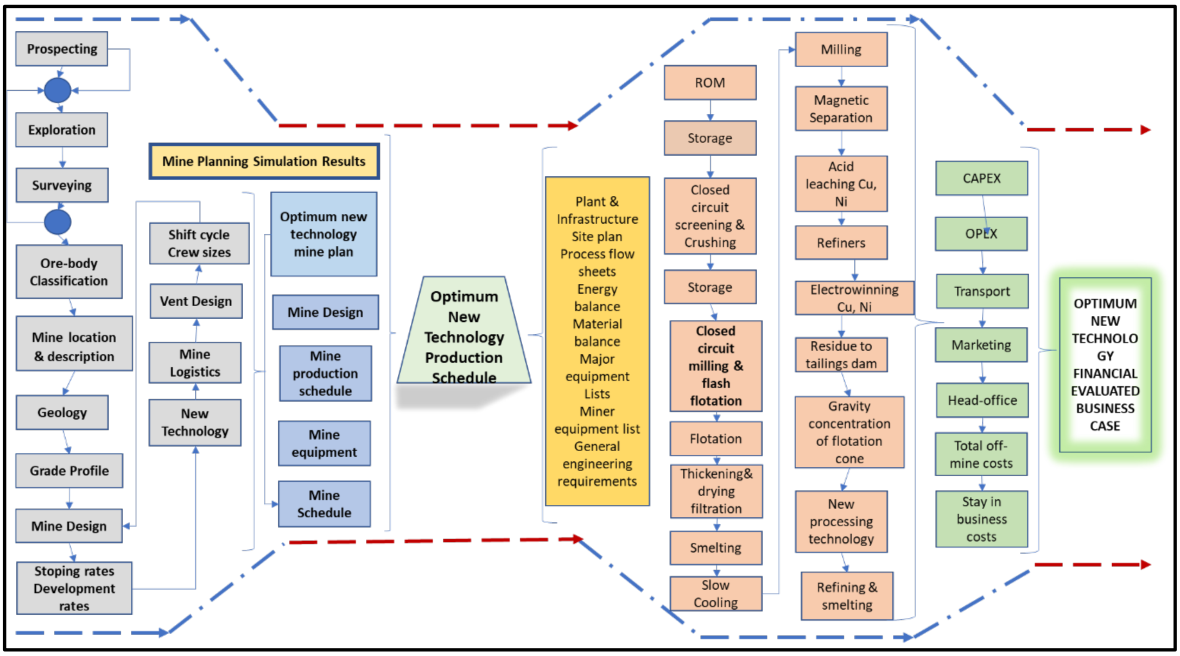

Figure 10 - New Technology Bankable Business Case Process Description

This proposed framework for developing business cases brings in the use of simulation techniques to arrive at an optimum new technology financially evaluated business case. Mining companies in general have tended to finance mining projects in-house. Banks lend funds which are repaid from project cashflows. Their recourse is limited to ensuring that the loan capital is repaid before any distributions to the owners' shareholders, and the potential for large profit margins is limited unless there is a high degree of risk associated with the project (Sullivan, et al,2015). The framework as proposed is anticipated to give the banks the added benefit of financial model verification and validation steps, to ensure risk mitigation in the outcome.

Chapter 3 – Methodology and Research Strategy

3.1 Preamble

Research design is referred to by (Mouton and Marais,1990) as the plan or exposition of how the investigator has concluded to implement the research to answer the articulated research questions. This chapter covers the details of the methodology and research strategy adopted in this study.

3.1.1 Research Methodology

The need for this study was to give mining companies seeking to secure investment when wanting to utilize new underground mining technology, an opportunity to identify, cost and design out key risks early, to setup operations for success. In order to answer the research question, a mixed-method research methodology was used, adopting qualitative and expert opinion knowledge base techniques. The purpose of this form of research is that it provides a better understanding of a research problem or issue than either research approach alone (Creswell,2008). Driscoll et al, (2007) state that mixed-method methodology provides practical advantages in exploring complex research question. Mixed-method research design is logical and intuitive providing a bridge between the qualitative and quantitative paradigms (Onwuegbuzie et al,2006). This method has the flexibility to meet the needs of this study. It provides opportunities for triangulation of data and ensuring its validity by expert opinion.

Qualitative and quantitative data complimenting each other provides robustness to the research process, one clarifying the other throughout the study. To answer this specific research question, the methodology was to breakdown the problem into various steps. The initial part of the design was to develop the technical tools for the study, this entailed compiling different mine designs for each type of PGM mining method, in order to analyse the differences in non-explosive mining compared to explosive mining methods. The PGM infrastructure could be developed either in an off-reef or on-reef configuration (Smith,2011). The mining layouts were thus divided into both options, off-reef and on-reef and further split into the various mining areas as shown in table 2 for the off-reef options and table 3 the on-reef options.

A base case mining layout was first developed from first principles and a financial analysis was conducted on this base case. All options were then compared against this base case. Each option followed a methodological process as per figure 10 below, in order to obtain the key financial indicators which were then used for comparison of non-explosive mining methods with explosive mining methods.

Table 2- Footwall Developed Drill& Blast/Cutting Simulation Scenarios (Cyest Scope of Work 2014)

Footwall Developed Conventional/Cutting Scenarios (Study Layout)					
Scenario	Mine Area				
	Shaft	Stations and Project Development	Barrels	Development	Stoping
Revised conventional Base Case	Conventional	Conventional	Conventional	Conventional	Conventional
C02	Boring	Conventional	Conventional	Conventional	Conventional
C03	Boring	Cutting (MTM)	Cutting (TBM)	Conventional	Conventional
C04	Boring	Cutting (MTM)	Cutting (TBM)	Cutting (RMDS)	Conventional

Table 3 - On Reef Developed Cutting Scenarios Studied (Cyest Scope of Work 2014)

On Reef Developed Cutting Scenarios (based on ULP layout)					
Scenario	Mine Area				
	Shaft	Stations and Project Development	Barrels	Development	Stoping
Revised on-reef base case	Conventional	Conventional	Conventional	Conventional	Conventional
M01a	Boring	Cutting (MTM)	Cutting (TBM)	Cutting (RMDS)	Slotting
M01b	Boring	Cutting (RMDS)	Cutting (TBM)	Cutting (RMDS)	Slotting
M03	Boring	Cutting (RMDS)	Cutting (RMDS)	Cutting (RMDS)	Slotting
M04	Boring	Cutting (RMDS)	Cutting (RMDS)	Cutting (RMDS)	ULP
M05	Boring	Cutting (RMDS)	Cutting (RMDS)	Cutting (RMDS)	Drill and Blast Conventional
M06	Boring	Cutting (RMDS)	Cutting (RMDS)	Cutting (RMDS)	Sandvik MN 220

Further, using the mixed-method processes can lead to discovery of emergent themes or information that would not have been otherwise possible with either qualitative or quantitative methods alone (Driscoll et al,2007). Hence, to answer this research question it will be required, firstly to find the key value driving parameters through quantitative analysis, as per the various mining options listed in table 2 for the footwall mine design and table 3 for the on-reef mine designs. Then it will be necessary to verify the outcomes of the financial analysis from each of these options analyzed, namely the NPV and IRR results by discussing with various key industry experts to assess their validity.

3.1.2 *Research Design*

The process that was followed to answer the research question from a research design perspective, 'what is the financial benefit to a PGM Mining Company in South Africa if it implemented non-explosive compared to explosives mining method?', is demonstrated in figure 11 below. The figure depicts the methodological process and interaction between, Mine 2-4D which is a mine computer aided designing software, Cyest Mine Scenario Planning solution which is a rules based mine design and activity scheduling solution, SimMine which is a mine simulation software, Cyest LoM Economics solution which is an operational modelling solution for forecasting, planning and optimization of the economics for a life of mine production schedule.

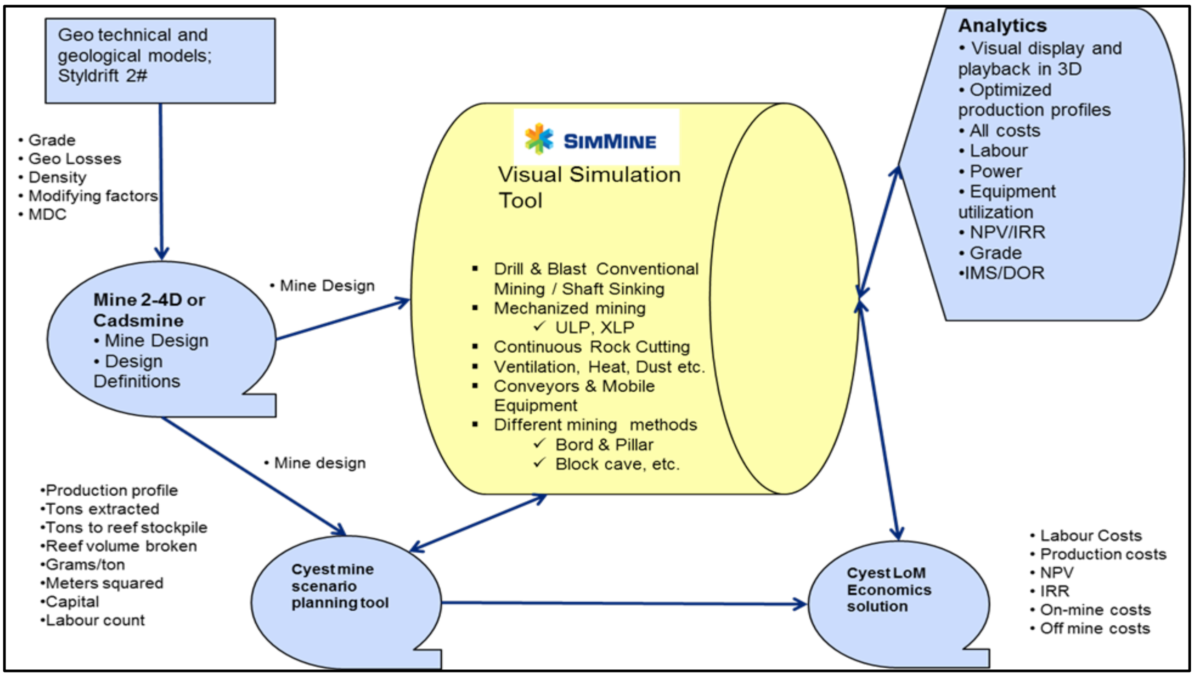

Figure 11: Central Question Modelling Process Flow (Raju, 2012)

The business cases for mining operations are developed using Oracle Hyperion Strategic Finance (HSF) tool, Excel spread sheets or Life of Mine Economic Solutions (Anderson et al,2005). HSF is a financial forecasting and modelling tool, Oracle Hyperion Strategic Finance, that has many features which can help the user generate NPV's and IRR's in the shortest timeframe possible. However, it does not have inbuilt features for the Simulation of various technical scenarios (Ballington,2002).

Using HSF various technical scenarios were run to show the impact of rock cutting technology. The successive scenarios incrementally adopted cutting technology, and was simulated using a Software called SimMine to study mine layouts on the off-reef pre-developed infrastructure of the following areas of the mine: shaft sinking; stations and project level development (before declines); decline barrels;

working cost development (haulages); stopes (Conventional drill and blast, ULP and slotting).

Thereafter, an on reef pre-developed infrastructure and logistical system based on the mechanised mine design was modelled. In this layout technical scenarios were run to show the value of using a mix of cutting technology in different areas of the mine. Figure 12 below, illustrates the study approach and scenarios for both the off-reef and on-reef evaluations respectively.

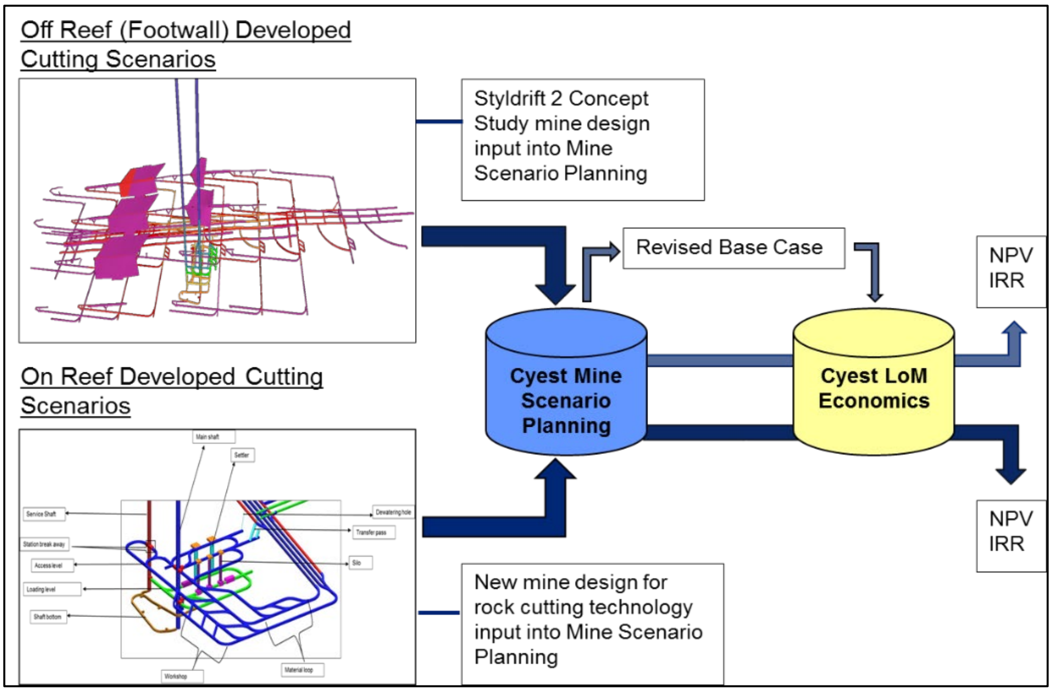

Figure 12: Research Approach and Scenario Evaluation (Raju, 2012)

Due to the size of these simulation models and the limited processing capacity of the IT technology presently. The options were only run three times per mining method, based on a triangular statistical distribution, using the minimum, maximum and mean input values. There were four different scenarios for the off- reef options thus 12 runs were executed. The on-reef scenarios had 6 different mining options, this equated to 18 runs. In total 30 runs were executed for the entire study in order to arrive at the final production and grade profiles as presented in chapter 4.

3.1.3 Result Analysis and Presentation

The results where to be analyzed and presented to various professionals from different disciplines and expert opinion was collected through unstructured discussions and the responses were analyzed and presented. The inputs into the financial models were verified against industry benchmarks firstly and followed a verification process via the outcomes of the simulation models run for the various scenarios.

3.1.4 Selection of Mine to Study

The statistical information that is required for this section of the research methodology adopted was obtained from completed platinum mining projects. The mine selected was an underground mine with a vertical shaft. In South Africa platinum mines exploit four types of orebody namely, Merensky, UG2, Platreef and Main Sulphide Zone (MSZ) (Lurie,2002). The vertical and inclined shafts are mainly used to mine Merensky, UG2 and MSZ orebody, while the opencast is used to mine the Platreef orebody. As non-explosive mining methods are primarily used to mine UG2 in underground platinum mines, (Van den berg,2014) the focus of this study was on this configuration of the mine. These mine designs were limited to the mine designs as referred to in tables 2 and 3. For the determination of ranges for sensitivity analysis and as a verification and validation of the key inputs, the various mining techniques were modelled and simulated. These simulation results were used as inputs into the HSF financial models.

3.2 The Research Instrument

The primary research instrument used was SimMine which is a mine simulation software that is used to plan, simulate and evaluate the mining process in an underground mining operation. The outcomes of the various simulation models are then verified against industry benchmarks. The process of using SimMine is as shown in Figure 13 below:

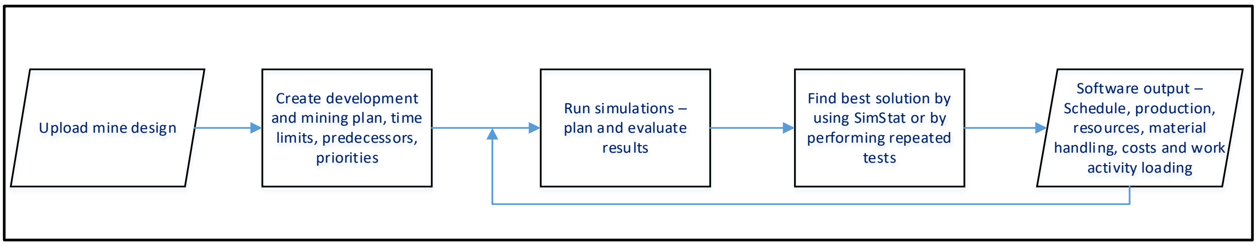

Figure 13 - SimMine Simulation Modelling Flow (Raju,2012)

In addition, feedback from the various professionals, on the usefulness of the simulated NPV and IRR results to achieve better decision making, unstructured discussions were used. The unstructured discussions are appropriate for collection of feedback as this provides the freedom to the respondent to answer whether he or she thinks the outcomes of the financial models are useful for decision making regarding investment in either of the mining methods compared.

3.3 Procedure for Data Collection

Projects completed by mining companies in the platinum industry were used to source project data on non-explosive and explosive mining methods that were sampled to determine the key value driving parameters. This included financial models as well as detailed feasibility reports of these projects. The feedback on the outcomes from the financial runs were collected via discussions with the various professionals.

3.4 Data Analysis and Interpretation

In order to determine the potential value add of utilising new hard rock cutting technologies (i.e. non-explosive mining methods) in the development and production of a platinum mine, a simulation model was developed and validated against existing work that had been carried out by OEM's and benchmarks collected from the hard rock mining industry. While using this data, has weakness in that circumstances from one mine to another may differ, an assumption is made that the rock hardness and mine design is more-or-less similar and within limitations is comparable to South African PGM mining conditions. The above-mentioned model served as the revised base case against which the new scenarios were compared.

According to Albrecht, (2010) classical thinking comprises three types of simulations: discrete event, continuous and Monte Carlo. The current thinking and work show that these lines are becoming less distinct. Discrete event method (DEM) (Nance,1993) is a model (mathematical and logical) of a physical system that has changes at a precise point in simulated time. DEM is a simple and versatile way of showing a dynamic system. DEM uses a series of instantaneous occurrences or discrete events. Using basic concepts like entities, queues, gates and servers, one builds complex models to explain fundamental questions such as latency, utilization, and bottlenecks. The simulation method adopted in this study was the Discrete Element Method.

Simulation can be used as a tool whenever complex flows exist (Albrecht,2010). Complex flows are characterised by randomness, several non-deterministic parameters or queues or traffic that is not linear and therefore impossible to solve analytically (Robinson,2004). The term complexity can be divided between combinational complexity and dynamic complexity. "Combinational complexity is related to the number of components in a system or the number of combinations of system components that are possible" (Robinson, 2004 pg.20). "On the other hand, dynamic complexity is not necessarily related to size. Dynamic complexity arises from the interaction of components in a system over time" (Sterman, 2000 pg.55).

At present simulation in the mining industry has been used for various purposes such as autonomous versus manual trucks efficiency, mine to mill production system, developing of new underground tunnels, optimizing the fleet in the underground, surface mine, simulation on haulage systems (including ore handling), scheduling maintenance comparison of the timing and efficiency between two different drills or even already existing operations (Okolnishnikov,2015).

3.5 Rock Cutting Technology

To choose the simulation tool that is the most appropriate for the use in this study, various mining simulation packages were considered to arrive at the best simulation software for this purpose. In simplified or deterministic case studies there is no need to use highly advanced programs (Bentley et al,1987). The important factors that should be considered when selecting a suitable simulation software are described by Banks,(1998) and briefly listed below: possibility of entering the data, processing characteristics and capabilities, possibilities of getting the specified output, environment, vendor and cost (first consider other features then productivity and finally need to price). For this study various mine simulation packages were considered namely: Simio, Gemcom, Arena, Whittle, Minex, Maptek, Simulia, Bentley, Intergraph, MinCom, SimMine, and SimMine was selected for the development and production modelling in this study.

SimMine was first developed by Onnerlöv in Sweden in 2000 and has since evolved to include more complexity and features currently available (Botha,2015). The most significant enhancement was to develop a direct import link to Mine 2-4d designs (SimMine,2020). This allows the importation of development heading and stope properties as well as sequencing. SimMine can also import mine designs in .dxf formats and simulate mine development and Life-of-mine (LOM) production. The key input criteria to the SimMine tool is the mine layout, design criteria and mine design dependencies (Botha,2015).

The model results provide input for a financial model to determine the best techno-economic continuous rock cutting methods. Dynamic simulation models consider real-world dynamic nature of systems and incorporate random elements, including equipment breakdowns, consequential delays (e.g. waiting for secondary equipment) competition for resources and other elements of interference to determine if the required output is achievable (Okolnishnikov,2015). SimMine® simulation software can simulate shaft sinking, station cutting and general capital development (SimMine,2020). The software can model both drill & blast and continuous rock cutting development as well as stoping. It has full 3-D animation functionality to display mine development sequence, equipment movement and progress. The mine layout is quite complex and includes various interdependencies that are included in the SimMine model. It also considers equipment performance and equipment availability parameters to establish the relative performance of the mining options.

3.6 Model Creation

A model is a simplified replication of the real-world system. The process of building the model requires a considerable amount of time. The preliminary steps consisted of a strategy plan and a decision if a simulation model should be used, or whether another approach adopted, to the required level of answer needed (e.g. experimental design, deterministic analysis among others). The decision process involved the support of the company in that it does not come only from a modeler (Banks,1999). Figure 14 represents an example, of steps in a simulation study. Each step is very important and should be carefully studied and efficiently processed, as failing in one may result in failing in the whole process of simulation (Banks,1998).

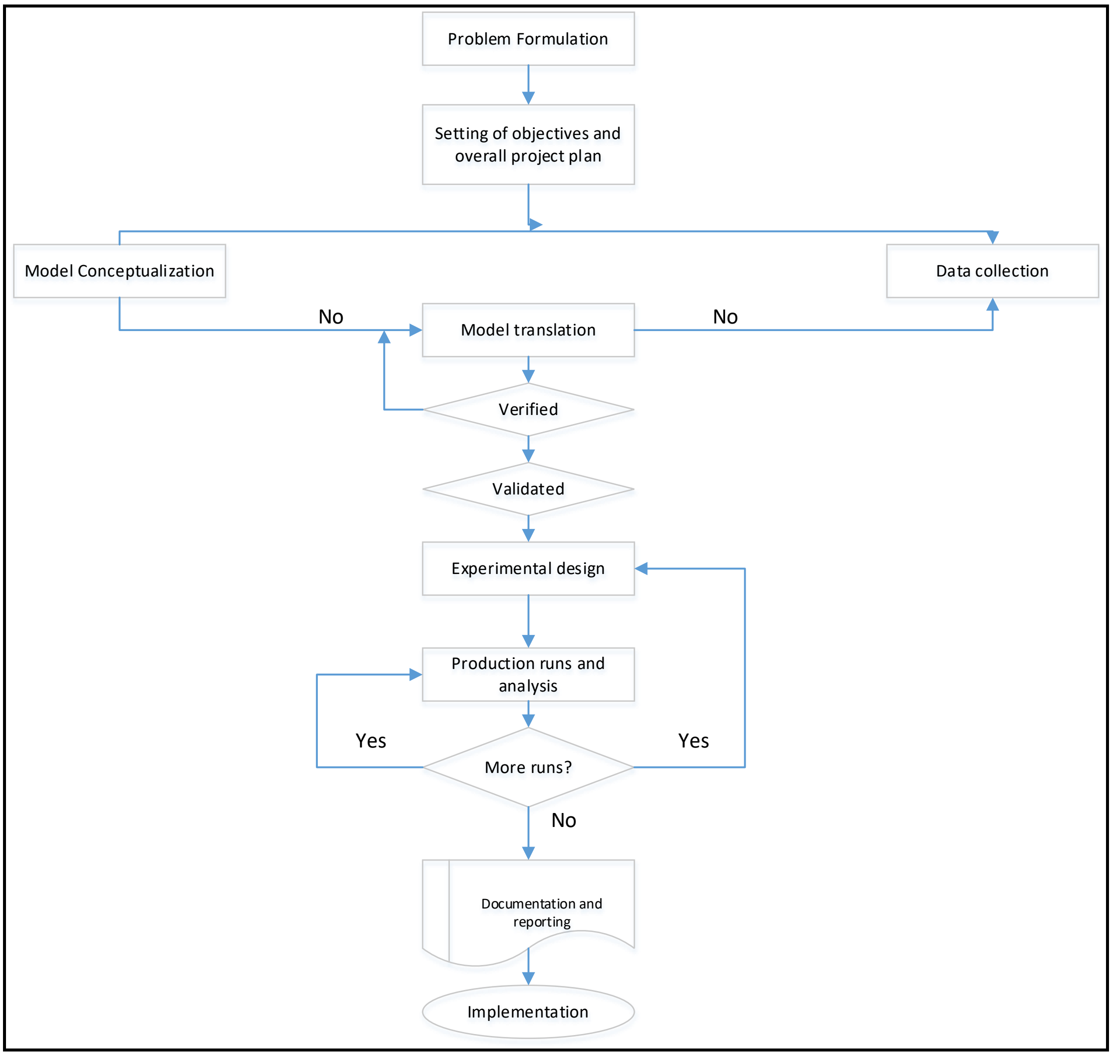

Figure 14- Common Steps in Simulation Model Set-ups (Banks,1998)

During the data collection and data processing, careful consideration of the following should be kept in mind:

- Wrong or bad input data (the most preferable is having an appropriate type and amount of raw data to be able to fit the data to the most appropriate distribution)
- Initial bias (the transient state in which the steady state is not reached (Thesen and Travis,1990))
- Lurking (confound) variables (the variables that associate the independent variable with a dependent variable (Sargent,1994).
- Autocorrelation (correlation ≠ causation; normally a plot showing the similarities between the

observations) (Thesen and Travis,1990)

• Non-representative sample (the sample which is included in the response factor and can be responsible for creation of the unwanted results) (Sargent,1994)

• Avoid data enhancement (additional feature that blurs the important view should be avoided) (Robinson,2004 and Banks,1998)

• Placing the data in the context (trying to not extrapolate, as too much data may cause bad effects on the model construction and the results of the model created) (Robinson,2004 and Banks,1998)

3.7 Verification and Validation

The model validation process is defined to mean "substantiation" that a simulation model within its domain of applicability possesses an acceptable range of accuracy consistent with the planned application of the simulation (Schlesinger et al,1979) whereas simulation model verification is often defined as "ensuring that the computer program of the model and its implementation is correct (Schlesinger et al,1997) .The verification and validation process was considered after the interval of every stage of step cycle described in Figure 14. The most clear and productive output of verifying and validating took place at the stages where the model was run at least once but it was often possible to even start correction and checking before translation of the data. Performing periodic/systematic checks gave a heads up in advancing (quickness and picking up the mistakes) with the enhancement of the model to the state of entirely ideal system but tackling the important factors included in the system at the appropriate and right level of details (Balci,1990). What should be understood by this phrase are: To check and look for the mistakes very often by not only via mathematical methods but also by using a common sense in doing and looking at the formulation of the problem, conceptual model, etc.

The life cycle is shown to give an idea of how this periodical verification or validation may be schemed as shown in Figure 15. For verification, ensuring that the conceptual model is accurately transformed into a computer models is necessary. Examples of the methods used in verification were: Testing the logic (avoid repeating and infinite loops), building diagnostic into the model, making flowchart of the model simulated, testing the model (checking the animation), running the model under varying conditions, checking for unexpected output (for example unexpected utilization), tracking statistics using business graphs/tables, tracing a model/entity, or using one of these debugging techniques (such as: defensive debugging, declarative debugging, sequential debugging or interactive debugging) (Sargent,1994).

In figure 15, Life Cycle of a simulation study (Balci,1990 pg.26), a series of checks were performed to ensure the correctness of the model and the reflections of the real system. There is a computer application called Interactive Run Controller (IRC) that helps with debugging the software but additional systematic code reading by the programmer was necessary in order to hunt, find and reduce the problems. Validation required the comparison of the model results with the real system ensuring that the model was sufficiently accurate. The difficulty of validation depended on the system's complexity. Examples of the methods used in validating are: Degenerate test (the systems behavior when the condition gets really bad), extreme condition test (the system behavior in the extreme case scenarios), face validity (credibility of the system), comparison to historic data (tabulate the data), internal validity (the model reaction to changes in random numbers), by conducting a Turing test people knowledgeable in the system were asked to differentiate between model and real data inputs, experts were asked to identify when they thought the results will differ and to justify their reasoning, sensitivity analysis (checking the responses).

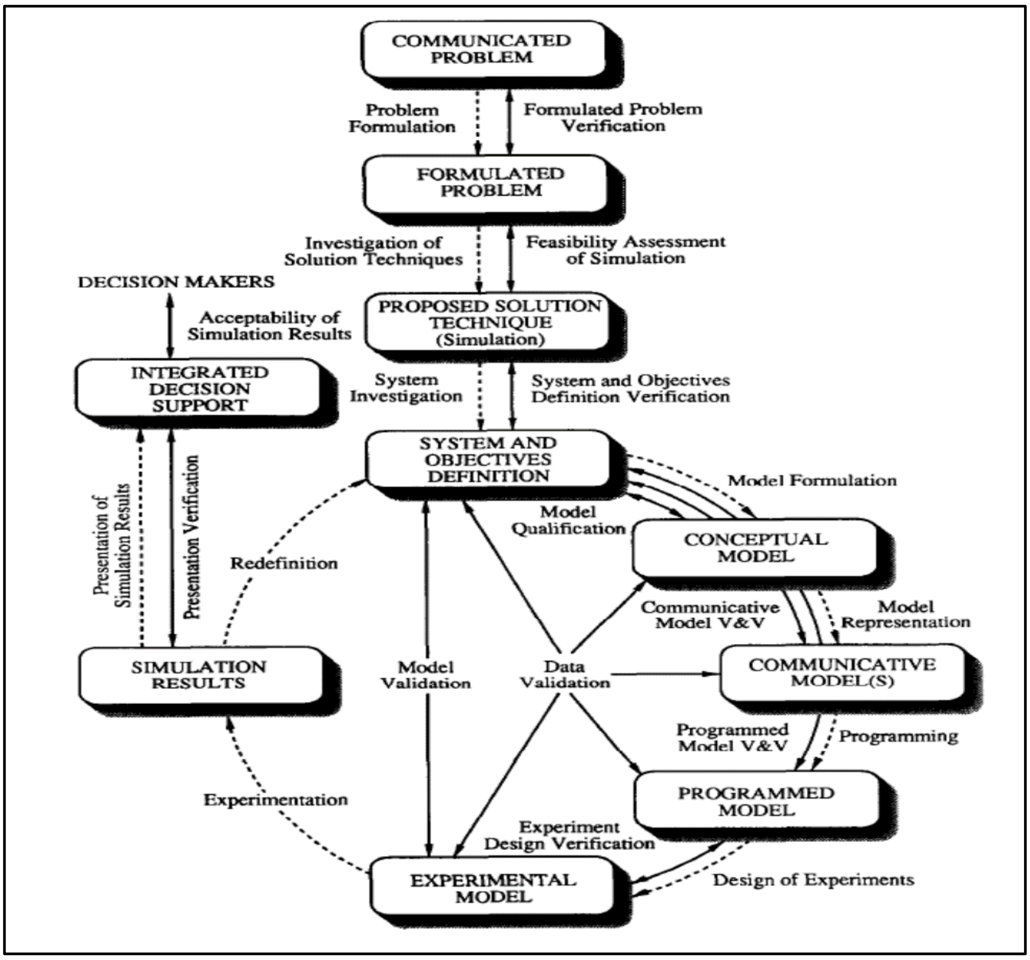

Figure 15- Lifecycle of a Simulation Study (Balci,1990 pg.26)

For a non-existent system, the team checked for similar systems, checked animation and looked for guidance from specialists that were accustomed to this problem, simulation and the industry (Robinson, 2004).

3.8 Simulation Model Set-up

For this study the author opted to utilise the flexible and user-friendly SimMine® simulation software, specifically designed to model underground mining operations. SimMine is a unique industry-proven simulator that reduces modelling time. It can directly import the planned mine layout. This simulation study was utilised to test and evaluate whether the design of the rock cutting, and drill and blast system can achieve the required production rates to support a credible business case.

A simulation model was constructed to evaluate as many as possible scenarios, making use of the SimMine® simulation software. The author sourced the model inputs, layouts and other operational procedures that is shift lengths and maintenance data; which was included in the model. The author also made input recommendations based on experience gained from other mining operations. The model inputs supplied was coded into the model for the simulation. The model outputs were viewed in a custom results report produced by SimMine®.

The simulation model consisted of the layout highlighting the planned primary and secondary development cycle completed by the conventional drill and blast suites of equipment and included the stoping done by the slot borer machine. The network layout was used to establish the relationship between development in the primary belt drives and stoping, by evaluating the advance rate for each of the defined process steps during operation, the availability of the cutting equipment and any availability of material handling system pertaining to the cutting operations. In order to model the specific layout in SimMine, a centre-line file of the planned layout had to be drawn using AutoCAD® as can be seen in figure 16 the Mine 2-4D designs were used as direct inputs in the SimMine simulation software.

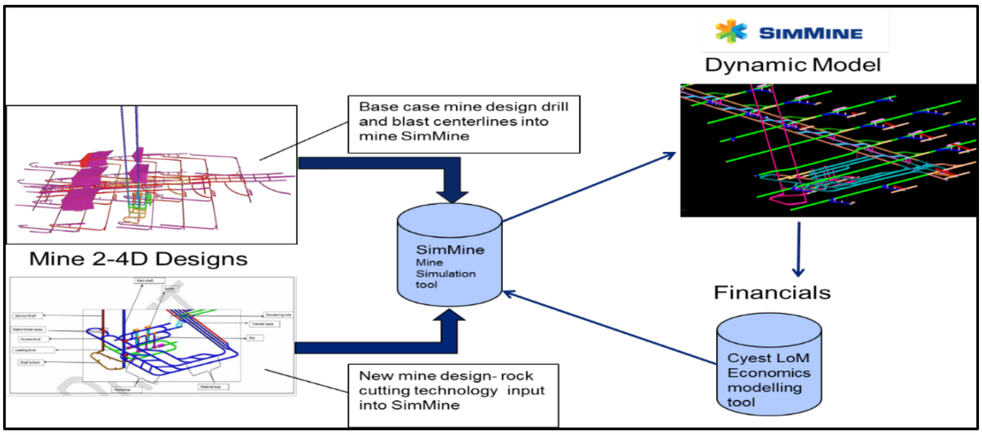

Figure 16- Simulation Model Setup, (Raju,2012)

To ensure the different mining methods were compared to the base case which was a scattered breast drill and blast option the following scenarios were modelled in SimMine, Scattered Breast, Hybrid Underground Mining, XLP/ULP Mining Method, Slot Borer stoping, MN220 Long wall stoping method, their layouts are as shown in appendix 4. The various mining layouts were compiled in Mine 2-4D and imported into SimMine.

SimMine is an input-based resource scheduling tool, which imports the mine design. Then allocates resources to the design, with availabilities and advance rates or durations to complete various task / activities. 3D animation can be set up to view the advance of the schedule on the mining layout that was imported as well as all results can be exported to MS Excel in a pivot table format. SimMine addressed all the shortcomings of traditional project scheduling and spread sheet-based solutions in the following way:

- SimMine does not assume deterministic development and production rates. Instead it simulates development advance and production based on detailed system and equipment constraints using the mine design layout and sequence (imported from Mine2-4D).
- More accurate equipment parameters, including Individual Equipment Fleet Size and Availability. Availability is based on statistical distributions of the mean time between failure (MTBF) and mean time to repair (MTTR);
- Ore Handling System, complete with individual elements (load bays, crushers, conveyors, silos, hoist etc.) and their respective capacity constraints and availability parameters;
- Individual sequential Development and Mining Activity cycle (drill, charge, blast, clean, support, / cutting rates, etc.). Equipment tramming between ends and stopes / tips are also simulated based on dynamic tramming distances as mine advances. Equipment interference (at intersections, tips, loading points, etc.) are also modelled.
- Process time variation expressed in statistical distributions (e.g. triangular, normal, exponential distributions). Because of the interdependencies of system elements, variation in process times can have a significant deteriorating effect on system performance;
- Dynamic speed variations on vehicles (full and empty) for different incline gradients.
- Additional underground light duty vehicles traffic can be included to model interference with production equipment;
- Able to quantify dynamic consumables & environmental Factors (Heat, Noise, Ventilation, etc.);
- Other elements in the simulator includes:
- Interdependencies (Sequencing); which are imported from the Mine2-4D layout as well as additional sequencing where required when the mine design has been imported;
- Production is calculated from first principle factors including, drill length, rock density, heading width and height; those tons are then transported according to the ore handling technique for that mining area.
- Rock Types (accounting for density differences and the percentage overbreak);
- Individual shift and maintenance schedules per mining area / machine;
- Visual Playback showing Equipment Movement in 3-D, either in translucent or rock face rendered

formats.

3.8.1 Geological Deposit Model Inputs

The Upper Critical Zone stratigraphy of the Rustenburg Layered Suite, which contains the units of economic interest, the Merensky Reef and UG2 layers, comprises well-developed cyclic units divided into six sub-units as follows: Bastard, Merensky Reef, Merensky Footwall, Intermediate Footwall, UG2 and UG1.

3.8.2 Resource Statement Model Inputs

It was decided to limit the size of the resource for the Study, due to the geological complexity of the various resource blocks. The block selected for the evaluation is the most stable block, Area 2.7 on the Merensky reef (Figure 17). This block comprises 65% of the Merensky reef resource.

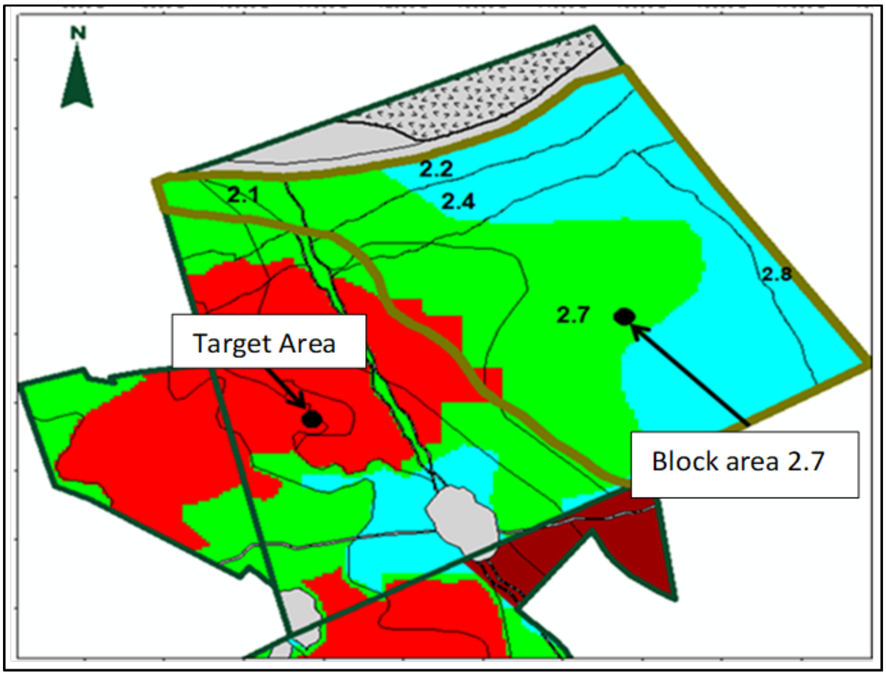

Figure 17- Target Area Merensky Reef Structure (Minxcon Report,2016)

Figure 17 shows the typical Merensky resources that were used for the evaluation of the project area for the Study, (Minxcon report,2016).

Table 4- Typical Merensky Reef Resource Statement (Block 2.7 only), (Minxcon Report,2016)

Merensky Reef											
Resource Block	Resource category	Dip (°)	Dip Area (Mm²)	Resource Cut (m)	Density (g/cm³)	Geo Loss %	Tonnage after Geo Loss (M/tons)	4E Grade (g/t)	Content (4E Moz)	Confidence % Tonnage
2.7	Measured	8.0	0.48	1.57	3.34	26.23	1.84	5.84	0.35	4
	Indicated	8.8	8.69	1.25	3.38	25.45	27.31	7.13	6.26	57
	Inferred	9.2	6.48	1.16	3.36	25.16	18.92	7.53	4.58	39
	Total	**9.0**	**15.65**	**1.22**	**3.37**	**25.36**	**48.07**	**7.24**	**11.19**	**100**

3.8.2.1 Reserve Calculations

The Merensky Reef stratigraphy was broken up into components as summarised in Table 5.

Table 5- Typical Merensky Reef Geozone (Minxcon Report,2016)

Width	Unit	
Hangingwall Overbreak	cm	5
Resource Width (weighted aver.)	cm	122
Footwall Overbreak	cm	10
Density	Unit	
Hangingwall Density	t/m3	3.38
Resource Density	t/m3	3.37
Available to mine Resource Density	t/m3	3.37
Footwall Density	t/m3	3.28
Grade	Unit	
Hangingwall Grade	g/t	0.33
Resource Grade (4E)	g/t	7.24
Available to mine Resource grade (4E)	g/t	7.24
Footwall Grade	g/t	0.39
Prill Split	Unit	
Platinum	% of 4E Grade	64.9
Palladium	% of 4E Grade	26.6
Rhodium	% of 4E Grade	4.3
Gold	% of 4E Grade	4.2
Nickel	% of tons	0.24
Copper	% of tons	0.11

3.8.2.2 Footwall Developed Scenarios

The modifying factors stated in the Target Study area (Table 6) have been applied in the Footwall Developed Scenarios reserve calculations.

Table 6- Typical Footwall Developed Scenarios Modifying Factors (Merensky Reef) (Minxcon Report,2016)

Name	Modification Type	Modification Factor
Geological	Loss	25.36%
Mining	Loss	2.00%
Pillar	Loss	11.20%
Other	Loss	6.00%
Mining into potholes/over stope	Gain	2.00%

Table 7- Typical Footwall Developed Scenarios Mine Structure (Merensky Reef) (Minxcon Report,2016)

Scenario	Avg. Back length (m)	X-Cut spacing	No of levels	No of half levels	No of blocks per half level
All	250	218	15	30	10

The scheduled depletion of reserves from the Target study area and of the Footwall Developed Scenarios are summarised in Table 8.

Table 8 – Typical Footwall Developed Scenarios Reserves (Merensky Reef) (Minxcon Report,2016)

Scenario	Resource width (m)	Reserve (Mt)	Grade (g/t 4E)	Content (Moz 4E)
Styldrift 2 Concept study	1.22	51.28	5.46	280.20
Revised Base Case	1.22	51.39	5.46	280.67
C02	1.22	51.39	5.46	280.67
C03	1.22	51.39	5.46	280.67
C04	1.22	51.39	5.46	280.67

3.8.2.3 On Reef Developed Scenarios

The on Reef Developed Scenarios modifying factors, mine structure and estimated Reserves are summarised in Table 9 to Table 11.

Table 9 – Typical On Reef Developed Scenarios Modifying Factors (Merensky Reef) (Minxcon Report,2016)

Name	Modification Type	M01 – M03	M04	M05	M06
Raise Pillars	Loss	6.12%	Layout does not incorporate raise pillars (1)		
Geological	Loss	25.36%	25.36%	25.36%	25.36%
In stope pillars	Loss	5.06%	5.93%	5.93%	8.47%
Over Stope	Gain	2%	2%	2%	2%

Note (1) As advised by Rock Engineering

Table 10 – Typical On Reef Developed Scenarios Mine Structure (Minxcon Report,2016)

Scenario	Avg. Back length (m)	Avg. Block length (m)	Avg. Block width (m)	No of levels	No of half levels	No of blocks per half level
M01a	315	245	200	12	23	11
M01b	315	245	200	12	23	11
M03	315	245	200	12	23	11
M04	310	240	200	12	23	11
M05	310	240	200	12	23	11
M06	238	168	200	15	29	11

Table 11- Typical On Reef Developed Scenarios Reserves (Merensky Reef) (Minxcon Report,2016)

Scenario	Area (Mm2)	Reserve (Mt)	Grade (g/t 4E)	Content (Moz 4E)
M01a	8.97	52.6	4.14	217.65
M01b	8.97	52.6	4.14	217.65
M03	8.68	50.6	4.21	213.0
M04	9.37	50.55	4.58	252.89
M05	8.78	43.89	5.12	244.22
M06	8.25	38.43	5.41	226.70

3.8.1 Summary of Simulation Model inputs and Assumptions

Below is a summary of the assumptions that were used.

- The estimate was determined in South African Rand's as decided upon at the commencement of the study.
- All inputs are shown in 2011 constant money and then escalated to nominal terms using the official escalation rates.
- The nominal values are then deflated using the official inflation rates to 2013 terms.
- The real cash flows are discounted using a real discount rate of 11.5%, providing an NPV in 2014 terms. This rate was used consistently for all business case comparisons for consistency.
- Revenue is generated from PGMs (platinum, palladium, rhodium and gold), nickel and copper.
- Recoveries and production numbers with metal pricing are used to calculate revenues.
- Royalties incorporated are as per the legislation in effect in September 2012.
- Income tax rates are as per the 2013.

3.9 Developing a cost model for input to Simulations

To determine the techno-economic benefit of utilising hard rock cutting technologies in the development and production of a PGM mining operation, a model was developed and validated against existing work that had been carried out for drill and blast, shaft sinking with drill and blast development and stoping. The drill and blast model then served as the Base Case against which new scenarios were compared.

3.9.1 Cost Modelling

The platform utilised in developing the cost model, was a proprietary object-oriented solution ('Qerent Modeller', Cyest Corp.com) that operates based on parent-child relationships. The value-add provided by this software was its ability to rapidly run multiple data sets through the model and immediately generate the desired outputs. Furthermore, the platform requires that any changes to the model inputs, resources, activities and outputs, need to be made once only, where the effects of these changes filter through immediately to all data sets. The objectives in developing the evaluation model were therefore to run multiple production schedules with different inputs and continuously refine the model logic, without compromising the efficiency of the overall process. As a result, the necessary feedback regarding the outcomes of changes to the model logic, inherent assumptions, or insertion of the latest production schedules could be quickly provided to all key stakeholders.

The logic in which these inputs were entered the software were based on the principles of activity-based costing (ABC), figure 18. This approach recognises that a relationship exists between resource consumption (input costs), activities, and outputs (products and services) (Cooper et al,1991). Given this relationship, input costs could theoretically be allocated to products or services through the activities that are performed within an organisation. The key to this process was to link an activity / allocation driver to the cost associated with the resource that is consumed in performing a given activity. An activity / allocation driver is therefore defined as a variable that underpins the cost of a activity and relates it to the output (Elkington,2019). The figure 18 below outlines the principles of an ABC model (Cokins,2001). The input costs associated with the outputs can be quantified by applying activity / allocation drivers to the resources consumed in the performance of the activities that give rise to these outputs (Rudenno,2012).

An organisation's activities are typically classified into two categories: 'primary' and 'secondary'. A primary activity is carried out by a resource that is directly associated with the output that is produced, whereas a secondary activity is carried out by a resource that provides a support function for the primary activity. This model used the mine design criteria (MDC), production schedule and other technical documentation as inputs to derive cost rates for mining and engineering activities (Rudenno,2012). The derived rates are used in the activity-based cost model which is driven by the production schedule to obtain the shaft head operating cost. Shaft head operating cost are subsequently used as inputs into the financial evaluation model to obtain the project value (Rudenno,2012).

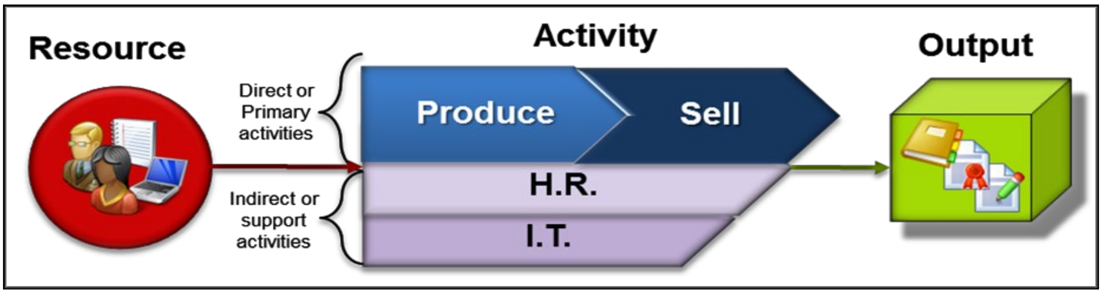

Figure 18: Principles of an ABC model (Drury,2001)

3.9.2 Activity Based Costing

Activity based costing is a methodology that builds costs up from all activities and sub-activities that constitute a process or operation (Drury,2001). Costs are driven by their applicable cost drivers to simulate the process of individual activities, incurring costs as they are planned in the production schedule. For example, conventional stoping costs are driven by square meters produced from the stoping activity and include consumables, sundries, contractor costs required to perform that activity (Hicks,1999). These costs are rolled up to contribute to shaft head OPEX, figure 19 below shows this rollup.

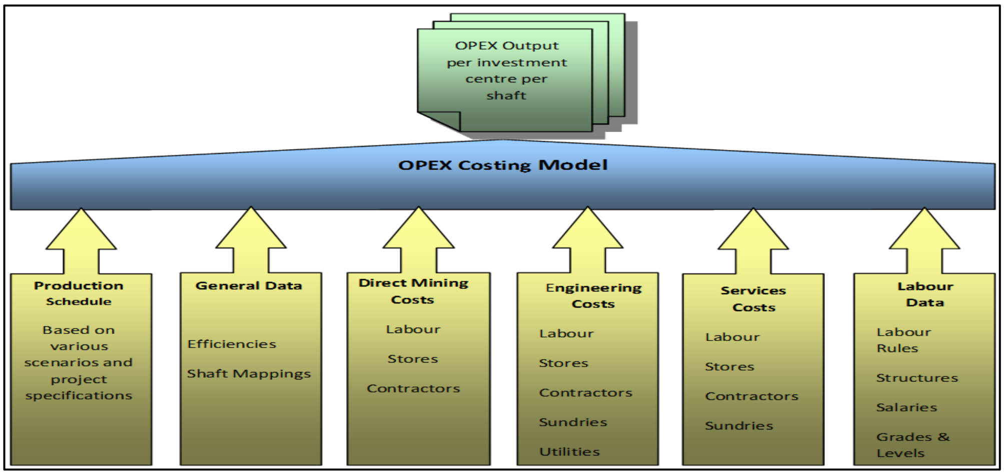

Figure 19: ABC model (Raju,2014)

3.9.3 Costing Structure

Mining Operation divides shaft head costs into three major costing categories, namely: Engineering; Services; and Direct Mining (Meyer,2006). The Engineering cost category caters for the operating cost of major infrastructure, maintenance and underground mining services. The Services costing category caters for all support services such as Finance, Safety, Health and Environment and management costs. This category also includes Mining Technical Services which catered for all ventilation, planning, survey and secondary support (rock engineering activity) costs. The Direct Mining category catered for all costs incurred on the half level / strike level and stoping levels. For each costing activity (SHE, Planned Maintenance, etc) the following cost resource elements captured like, Labour, Stores, Sundries, and Utilities. The labour cost element catered for the direct labour related to stoping. The Stores costing element catered for all consumables used in the activity whilst the sundries cost element caters for all skills levies, stationery and contractors. The utilities cost element catered for compressed air, power and water and was allocated to the Shaft Structure Operate, costing activity and encompasses the entire utility bill for the operation and feeds into the cost elements structure as per figure 20 below.

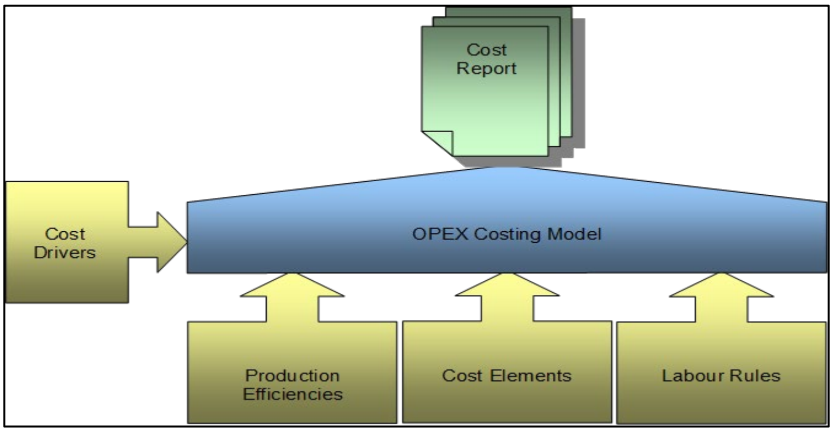

Figure 20: Costing Structure (Raju,2014)

3.9.3 Model Framework

The model that was developed for this analysis allowed the generation of the operational expenditure rates from proposed equipment data and information. The Base Case Scenario was populated with information as supplied from the database of the engineering consultants. The cutting technology costs for the remaining scenarios were sourced from internal data bases and consulting companies. A generic costing model was developed for the Original Equipment Manufacturers that allowed the generation of operational and capital expenditure numbers from the production schedules. The model was populated with activity costs, labour structures and rules and efficiencies for the relevant study area and scenarios and comprises rules and relationships internal to the functioning of the models.

3.9.4 Cost Inputs

The following cost inputs were used for the compilation of the various models:

- Production inputs from the Mine Scenario Planner scheduling software and unit operating costs are used to calculate Operational Expenditure (OPEX).
- Capital costs are derived from project development schedules and capital cost rates.
- All cash flows including Revenue, OPEX, Capital, Stay - In Business Capital, and Working Capital are then used to calculate a free cash flow after royalties, income taxes and taxes on dividends have been considered.
- Working capital requirements are incorporated as per the 2013 consensus assumptions as supplied by S&P Global.
- On-mine operating costs include mining, concentrator and central services (indirect) costs
- Off-mine costs are calculated at the PGM forecast averages and include smelting, refining, transportation and other marketing and corporate costs related to the metal produced in line with the 2013 consensus assumptions supplied by S&P Global.

3.9.5 Capital Estimates

Capital is calculated using the outputs of the mine scheduling model in terms of tonnages and metres developed and multiplying these with the cost rates as applied in this Study. These resultant costs make up the mining subhead of the capital estimate. Other subhead costs such as mechanical, civil, concentrator and other are incorporated as per this study. The Replacement Capital has been included as per the Study. SIB Capex is included as a proportion of the simulated OPEX for each option analysed and was calculated as follows:

- Footwall Developed scenarios:
    - Mining Stay in Business (SIB) cost was estimated as 8% of shaft-head costs and indirect

costs, and (variability of these costs was as per the triangular distribution of the OPEX that was simulated via the software)
  - Processing Stay in Business (SIB) cost was estimated as 12% of concentrator OPEX.
- On reef developed scenarios:
  - Mining SIB 12% of shaft-head costs and indirect costs, and
  - Processing SIB 12% of concentrator OPEX. (variability of these costs was as per the variability of the OPEX that was calculated)

Capital costs are driven by volumes and length, provided by the mine schedule. The shaft costs were derived from the metres sunk multiplied by the rate per metre used by the engineering consultants. All other costs were provided per volume of material excavated. The above calculations provide the initial Capex values for the mining subhead. For all other Capital subheads, the same values as per this Study were used. These include the mechanical, civil, processing and other subheads.

3.9.6 Capital Footprint

The capital footprint of the Footwall Developed Scenarios has been defined, as all activities developed in the shaft, stations, decline barrel and every second raise line per half level while that of the On Reef Developed Scenarios has been defined as all activities developed in the services shaft, vent shafts, main shafts, stations and decline barrels. The conventional blind sinking of the main, service and ventilation shafts were costed as per the detail in Table 80 Appendix 2, while the mechanical blind shaft boring costs are summarised in Table 81 Appendix 2. The mechanical blind shaft boring machine is introduced from scenarios C02 to C04.

3.9.7 Revenue

Revenues were calculated using the 2013 Consensus Market Forecast (CMF) pricing and the metal content of ore processed (provided by the mine schedule). Recoveries for the concentrator are determined using the grade/recovery curve used in the study. For the smelter and the refinery these are sourced from the 2013 CMF. The mine schedule provides tonnages of material (Merensky) milled as well as the contents (grades) of metals within the material. These are a function of the current area being mined. Grade and recovery relationships used by the analysis were incorporated to derive recoveries at the concentrator. Recoveries at the smelter and Refinery are as per the 2013 CMF, as is the pricing of metals sold after the Refinery pipeline is considered.

The net sales revenue is highly dependent and variably based on the mineral grade profile, and the commodity prices. The cost build-up structure is usually made up of the following costing elements as stated in table 12 below.

Table 12- Mining Project Typical Cost Bucket

Cost 1	Cost 2	Cost 3	Cost 4	Cost 5	Cost 6	Cost 7
Mine development cost	Concentrator costs	Cost1+Cost2	Cost3 +	Cost 4 +Capex	Cost 5 +Tax	Cash flow after tax
Stoping cost	Smelting costs		Other indirect costs			
Other direct mining cost	Refining costs		Royalties			
Engineering cost	Group centralised services costs		Stay-in-business costs			
Shaft service costs	Purchasing costs					
On mine central services costs						

The amount spent on each cost item (cost1 to cost5) depends on the production profile, which in turn is developed from the mine extraction strategy of the operation. Given that the calculation of the NPV.IRR depends on all the operational and capital costs represented by items from cost1 to cost 6 in table 12, it can be stated that the NPV/IRR modelling process is dependent on the production profile. Conventionally this production profile is developed using benchmarks from tried and tested mining methods sourced from existing in-company databases and competitor mining houses, namely sourced benchmarks. On the other hand, for mining methods adopting new mining technology these benchmarks that are used as inputs into the production profile development are not readily available and if used could give the mine planner vastly over or understated outputs. This could in turn give the financial modeller an inaccurate cost 7 and subsequent inaccurate NPV and IRR for this mining operation.

In order to compile a production profile for the mine, the decision environment is made up of many variables namely: Orebody type, Mining method, Access method, Development rates, Mining Rates, Fleet sizes, Development end sizes, Geo-techniques, Ventilation, Logistics, Rate of advance. One of the ways to ensure that the production profile outputs are optimum is to use simulation models to mimic the operational systems. These mimic systems must incorporate probabilistic elements to cater for all expected and unexpected events. The outputs will then cater for the inherent risks associated with new technology mining methods.

3.10 Survey Method Adopted to Obtain Industry Experts Opinions on New Technology Deployment

An informal discussion was employed as part of the verification process for this research, the details for which will be described in more detail below. Jotform Survey engine an electronic medium of communication with respondents was used as the tool for communication with the respondents. The questionnaires were electronically posted to the respondents with the request to complete the questionnaires and return by mail.

3.10.1 Layout of the Questionnaire

Questionnaires are one of the most widely used research instruments and a number of techniques and processes for developing and constructing questionnaires are presented by Welman and Kruger (2001). Their recommendations included that the respondents remain anonymous, they are experts in the field of the study, always be considerate of the respondent's knowledge base and maintain neutrality.

For this study, closed-ended questions were employed, more precisely multiple-choice questions with a combination of (i) various options, as well as (ii) structured questions that provides for picking a response from alternatives, Appendix 1. The questions were posed to experienced mining engineers, program managers, project managers, finance managers, heads of departments, whom were familiar with financial decision making. The questions were articulated using words and concepts with which they were familiar.

3.10.2 Structure of the Research Questionnaire

The research project questionnaire was separated into the following four parts:

- Part A: Position or Role of the respondent: - questions relating to participation role in the company and the type of mining company.
- Part B: Factors affecting the decision to invest in New Technology Mining Projects: - questions relating to factors that influence investment decision making regarding new and innovative mining methods in the platinum industry.
- Part C: Key factors in business case development: - questions concerning what is required to develop a mining business case.
- Part D: Investment Decision Key Indicators: - these questions focused on the key financial indicators for project investment decision making. It also tested the use of simulation to verify the inputs in these calculations

The results from the technical simulation studies and financial evaluations are used for comparative results, and emphasis in the research project questionnaire is on the factors critical for project investment. The questionnaires were sent to different management levels including, consultants, mining engineers, capital project managers, vendors and technology engineers in the platinum group of metals industry. The participants selected for the analysis were chosen through a convenience sampling method from different mine mechanization technology-driven divisions within the platinum industry which include, table 13 below:

- Mining House Technical Staff (Engineer, Scientist, Designer)
- Mining House Management Staff (Project Management, Engineering Management, Junior Management, Senior Technical Division (Engineering and Mining)
- Consultant (EPCM, Management Consultant)
- Vendor (Equipment Manufacture, Equipment Supplier, Technology Developer)

The selected experts were working in diverse hard rock mining within a targeted sector of a platinum group of metals (PGM) company in South Africa. The number of employees selected were twenty-eight and all of them responded.

Table 13- Experts Opinions Responses

TARGET SAMPLE	SAMPLE SIZE	RESPONSE
Mining House Technical Staff (Engineer, Scientist, Designer)	5	5
Mining House Management Staff (Project Management, Engineering Management, Junior Management, Senior	17	17
Consultant (EPCM, Management Consultant)	4	4
Vendor (Equipment Manufacture, Equipment Supplier, Technology Developer)	2	2
Total	**28**	**28**

Chapter 4 - Results and Analysis

4.1 Preamble

The various options were modelled using a triangular distribution based on minimum, maximum and mode where for a symmetric distributions mean is equal to mode. Due to the size and complexity of these simulation models the use of a triangular distribution provided a simplistic representation of the probability distribution of the various input variables (Mun,2008). The simulation models were then run for each of the options as per the outcomes from this distribution analysis. The results from these simulation runs coupled with the expert views are covered in this chapter.

4.2 Model Results

Figure 21 shows the total reef tons mined per option simulated. The models were run using the maximum, mode and minimum limits of the input values. The erratic profile of the production output is as a result of the travel time of machinery between mining sections and tramming for hauling purposes. The output as shown in figure 21 although depicted as a saw tooth format demonstrates a shift of the production tonnages to start steady state ramp-up much earlier than the drill and blast options.

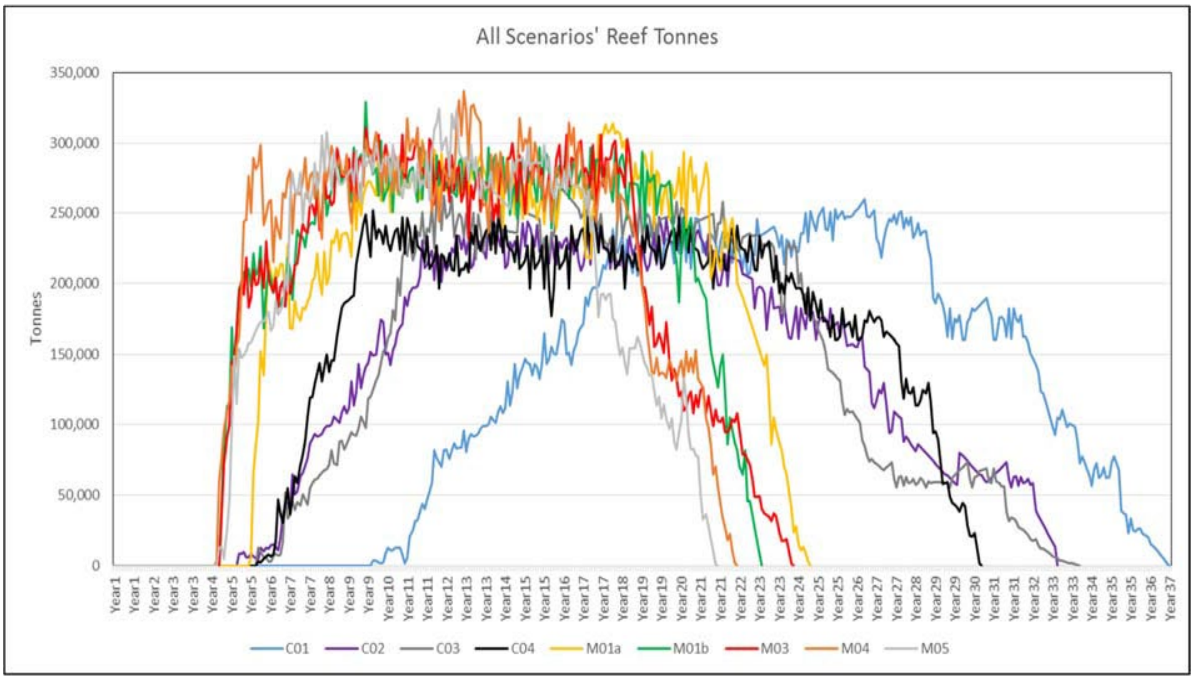

Figure 21- Total Reef Tonnes All Scenarios

Figure 22 shows the breakdown of the equipment times, operating, controllable, operational stops, non-controllable, scheduled maintenance, unscheduled maintenance and consequential delays. The analysis shows that the slot borer operates for 46% of the day which equates to 11.1 hours. The RMDS only 7.5

hours. The RMDS spent 4.7 hours dealing with consequential delays. The TBM, MTM and VBBM spent 3.2 hours on unscheduled maintenance during the daily operating cycle.

Table 14 - Model Results to Production Ramp-up

Off-Reef Scenarios		On-Reef Scenarios	
Scenario	Production Ramp-up	Scenario	Production Ramp-up
Revised base case(C01)	12		12
C02	6	M01a	2
C03	6	M01b	1
C04	3	M02	1
		M03	1
		M04	1
		M05	1
		M06	1

Faster ramp ups are achieved by all scenarios from M01a to M05 because of the on-reef layout used. This provides early on-reef tonnages from the decline barrels and strike drives, which also decreases the life of mine. M05 is the scenario that finishes first in the end of Year 21, with M04 second ending in the start of Year 22. Figure 18 also showed that the SimMine developed model's, production profile ramping up slower than the static Excel based models after starting with the on-reef barrels a year later than the static model. The figure shows the static model finishing the station areas two years after starting, by sinking and equipping the shaft using a mechanized, Vertical Blind Boring Machine(VBBM) (one year) and cutting both station levels using the RMDS (one year) with a cutting rate of 235 meters a month per machine. The SimMine model finished the station area 3.58 years after starting, by also sinking and equipping the shaft using the VBBM (one year) but taking 2.58 years to complete the station cutting for both levels at an average cutting rate of 127 meters per month achieved by each machine.

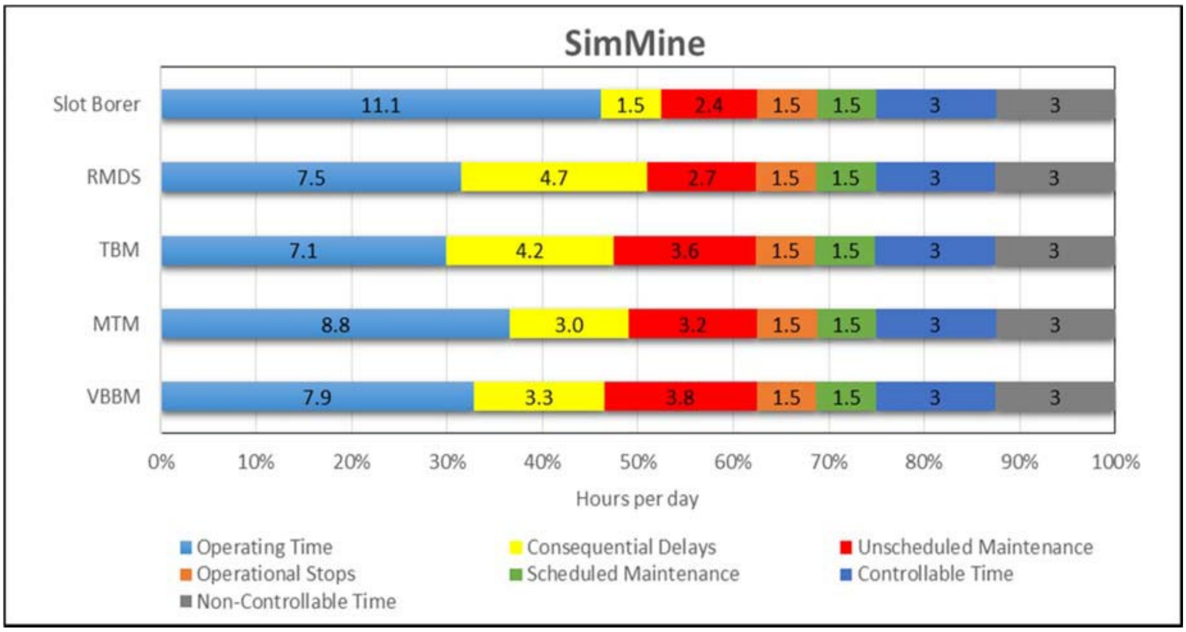

Figure 22- SimMine Machine Cycle Breakdown

Figure 23 shows the scenario comparison for C02, C03 and C04 to the revised base case as depicted in the dashed lines.

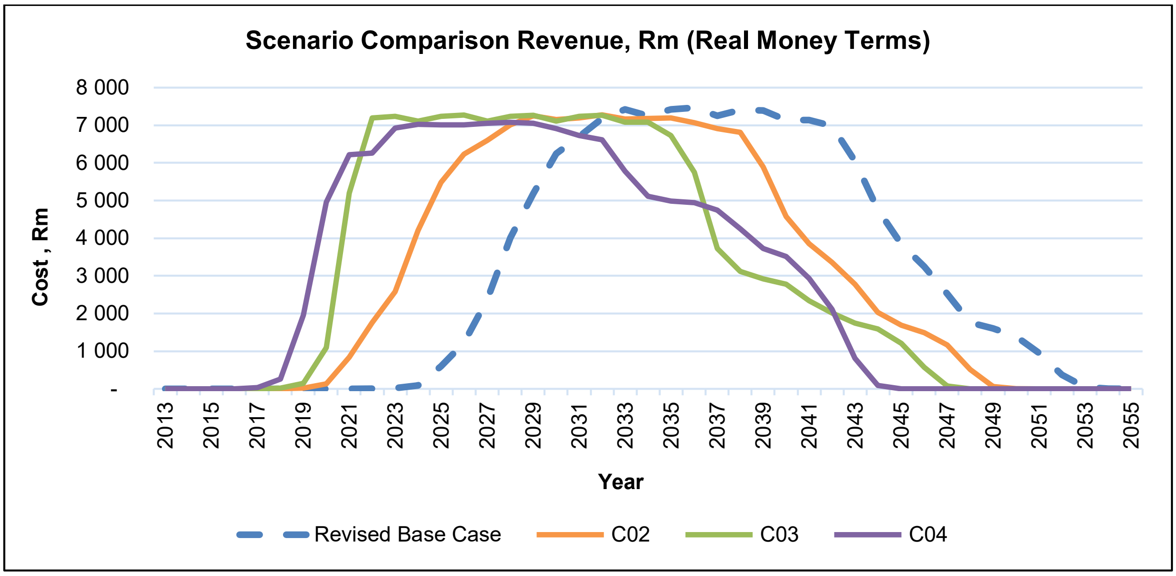

Figure 23: Footwall Developed Scenarios Revenue comparison

Figure 24 shows the scenarios as compared to M01, M01b, M03, M04, M05 and M06 to the base case as depicted in the dashed lines.

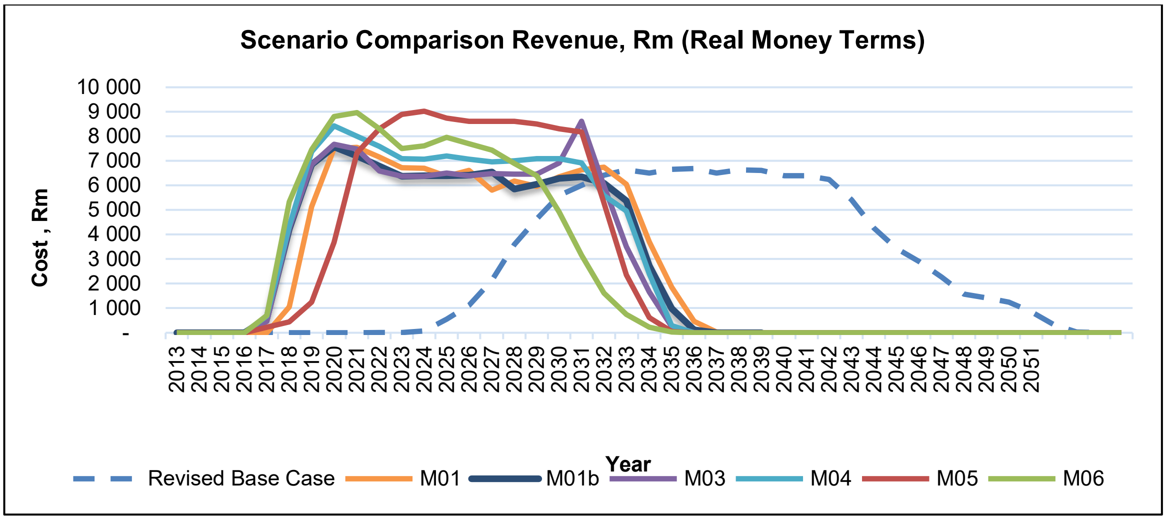

Figure 24: On Reef Developed Scenarios Revenue comparison

4.3 Results

All inputs are shown in 2011 constant money terms and escalated to nominal terms using the official escalation rates (2013 CMF as per S&P Global). The nominal values are then deflated using the official inflation rates to 2014 terms. The real cash flows were then discounted using a real discount rate of 11.5%, providing an NPV in 2014 terms. This rate was used consistently for all business case comparisons.

4.4 Footwall Developed Scenarios

Figure 25 shows the footwall developed scenarios as compared to the base case using rapid development methods. The mine design catered for an off-reef mining method and then the cutting technology was incrementally added into the design for the various areas as described in table 2.

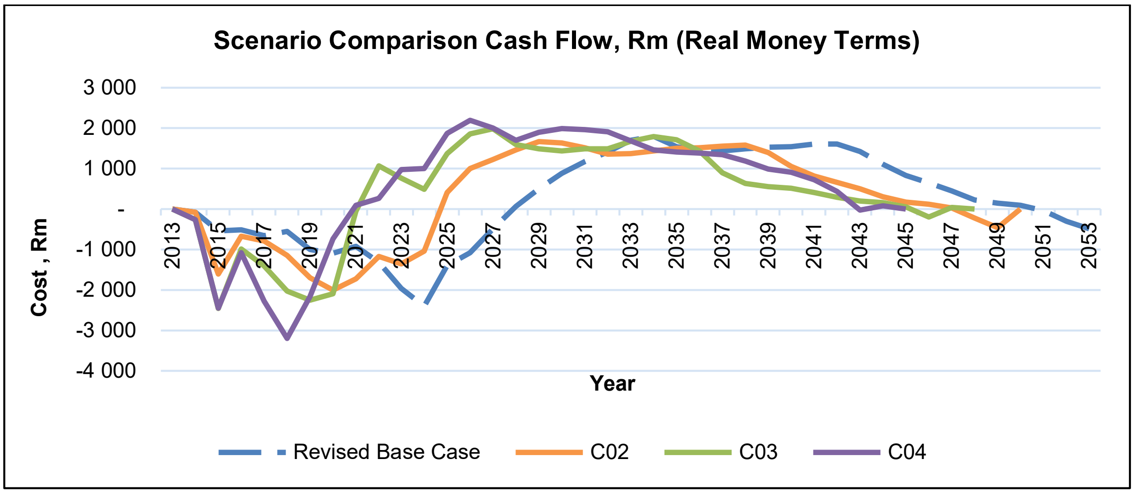

Figure 25: Footwall Developed Scenarios Cash Flow comparison

It can be seen from the cash-flow graphs in figure 25 that the cashflows C02 to C04 were shifted earlier as rapid development was introduced into the mining methods as compared to the base case.

4.5 Results from Revenue Analysis

Mining companies exist because they have the capability of providing value to their customers and in the process generating revenues that are in excess of costs. Profits are created when revenues exceed costs but loses are created when costs exceed revenues. Table 15 shows the base case revenue compared to C02, C03 and C04.

Table 15: Footwall Developed Scenarios Revenue comparison

	Revised Base Case	**C02**	**C03**	**C04**
Revenue (Rm)	133 047	133 519	134 487	132 135

Table 16 shows the revenue from the various on-reef scenarios that were modelled compared to the base case.

Table 16: On Reef Developed Scenarios Total Revenue comparison

	Revised Base Case	**M01**	**M01b**	**M03**	**M04**	**M05**	**M06**
Revenue (Rm)	133 047	104 620	105 279	105 341	113 117	107 068	101 809

The revenue for the three mining options are very similar to the base case revenue output. These revenue profiles do not show the impact of rapid development and rapid stoping operations. For the on-reef options one can also notice that the revenue from the base case is much higher than the mechanised options, this implies that the revenue is not the best indicator to compare these different mining methods.

4.5.1 Footwall developed Scenarios

The off-reef mining scenarios yielded the following Life of Mine Rand per ton and reef tons per year. The footwall or Off-Reef options were run using prices and the inputs stated in table 88 and obtained from Appendix 1,2 and 3 as sourced from the industry benchmarks.

Table 17: Footwall Developed Scenarios Steady State comparison

Scenario	**LoM R/t**	**1st reef tonnes year**	**Steady state year**
Revised Base Case	988	2022	2032
C02	988	2018	2028
C03	919	2018	2022

C04	801	2017	2022

The NPV's yielded for the Off-reef options all were negative as per table 18 below. The table also shows the comparison between the base case and the calculated NPV's for each scenario.

Table 18: Footwall Developed Scenarios NPV comparison

Scenario	NPV (Rbn)	IRR (%)	NPV Diff between base case and options run (Rm)
Revised Base Case	-3.39	NA	
C02	-3.53	NA	-138
C03	-2.76	4%	768
C04	-2.80	6%	-45

The options run, where the mine development was done in the foot wall or Off-reef, which is numbered C02, C03, C04 in the table 9 above, yielded the NPV's and IRR's as stated. This showed that all the options were negative and showed an IRR that was below the investment hurdle rate of 11%.

In figure 26 the bar graph on the extreme left shows the base case NPV. The bars to the right show the NPV's of the various scenarios as was described in table 2.

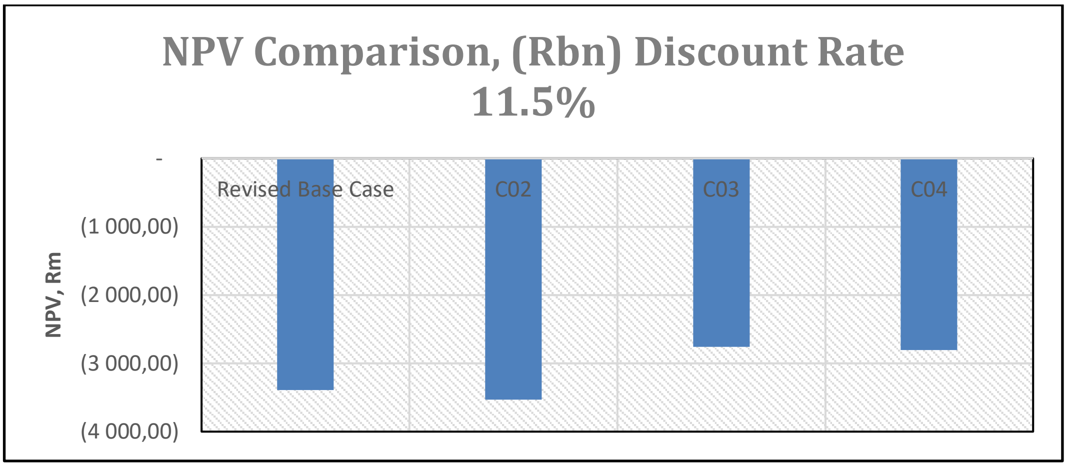

Figure 26: Footwall Developed Scenarios NPV comparison

4.5.2 On-Reef Developed Scenarios

The On-reef options outputs from the various models as described in table 3 is as stated in table 18 below. It shows the Life of Mine (LoM) in years. Table 19 also show the reef tons produced and the life of mine which is much less that the 32years planned for the base case.

Table 19: On Reef Developed Scenarios Life of Mine parameters

Scenario	Reef Tonnes (Mt)	LoM years	1st reef tonnes year	Steady state year
Revised Base Case	51.39	32	2022	2032
M01	52.63	19	2017	2020
M01b	52.63	20	2017	2020
M03	50.60	19	2017	2020
M04	50.55	18	2017	2020
M05	43.90	18	2017	2022
M06	38.44	18	2017	2021

Figure 27 indicates that the On-Reef options provided much better cashflows then the base option. The on-reef options were modelled as per the various scenarios listed in table 3. The hard rock cutting technology was introduced in an incremental manner into the mine design.

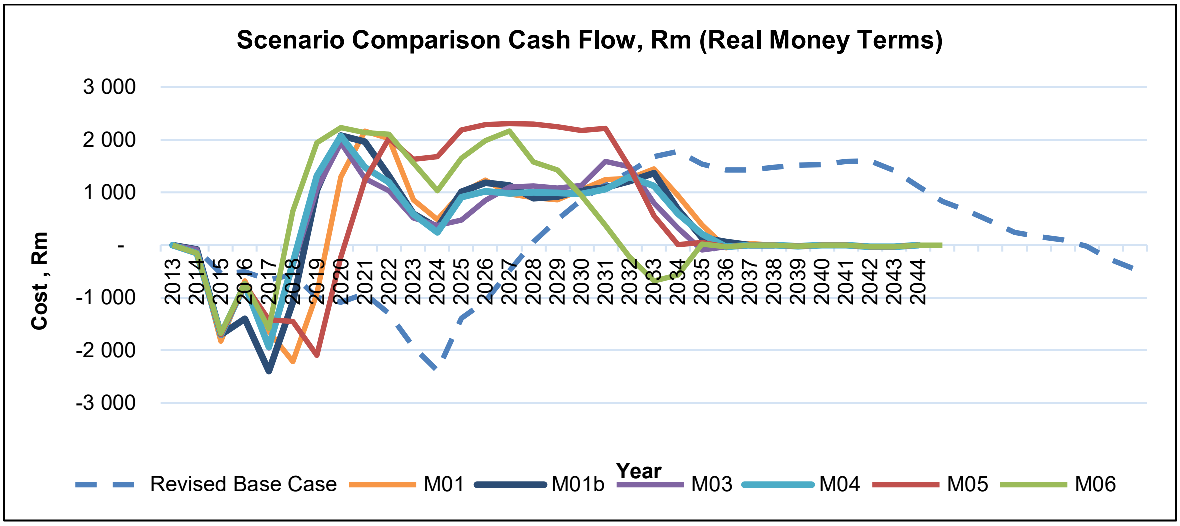

Figure 27: On Reef Developed Scenarios Cash Flow comparison

Table 20 shows the outputs from the financial analysis of the various on-reef mining options. The mining options considered was based on an on-reef mining layout were access to the reef horizon was achieved via a shaft boring machine as shown in figure 88. Once on the reef horizon the haulages and other access tunnels were all pre-developed using a RMDS, MTM or TBM. The layouts were further designed to cater for a slot borer (figure 88) which adopted a retreat mining method a MN220 which adopted an up-dip mining method to mine the reef. The ULP mining system was also modelled as shown in figure 90 (Appendix 4). These were all compared to the base case which was the conventional breast mining method.

Table 20: On Reef Developed Scenarios NPV comparison

Scenario	NPV (Rm)	IRR (%)
Revised Base Case	-3 39	NA
M01	-225	11%
M01b	313	13%
M03	764	15%
M04	1 118	16%
M05	657	13%
M06	3 420	28%

The NPV and IRR values as shown in table 11 showed that as the time to reach the ore body become shorter these values moved from a low of – 225 million rands to a positive 3.42 billion rands. The IRR improved from a value of 11% to a maximum of 28%.

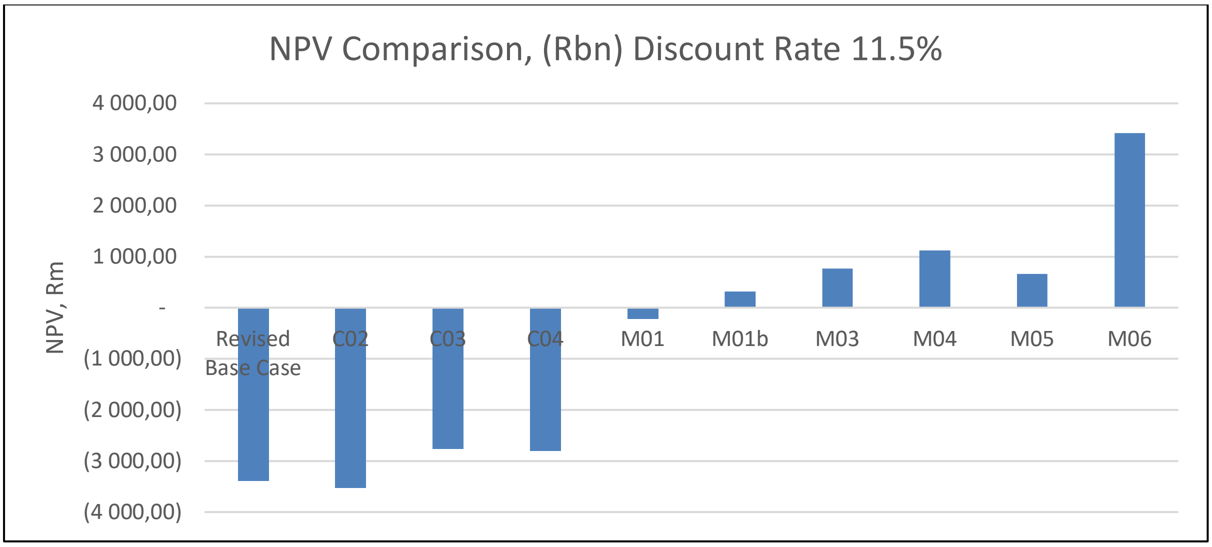

Figure 28: On Reef Developed Scenarios NPV comparison

Figure 28 showed the revised base case and the off-reef and on-reef scenarios as described in table 2 and table 3. The scenarios catered for the different reef mining horizons and the introduction of the various mechanized equipment to the mine design.

4.6 Results of the Expert Opinions

The aim of this section is to evaluate the data gathered based on an electronically mailed questionnaire and during the empirical phase of this research project. Data were gathered with the emphasis on the factors affecting the decision to invest capital in mechanised underground mining operations in SA. The research outcomes are presented the three categories:

- Factors affecting decision to mechanise with focus on financial decision making.
- Key financial investment making indicators
- Simulation software for business case optimisation

4.7 Response rate

The questions in Part A of the research questionnaire offered the chance to assess the position and role of the respondent in the organisation as illustrated in Figure 2929 below. The representativeness of the (28) twenty-eight respondents was made up of 18 percent of Mining House Technical Staff, 14 percent of Consultants working on the platinum group of metals capital projects and the 78 percent majority was Mining House Management Staff.

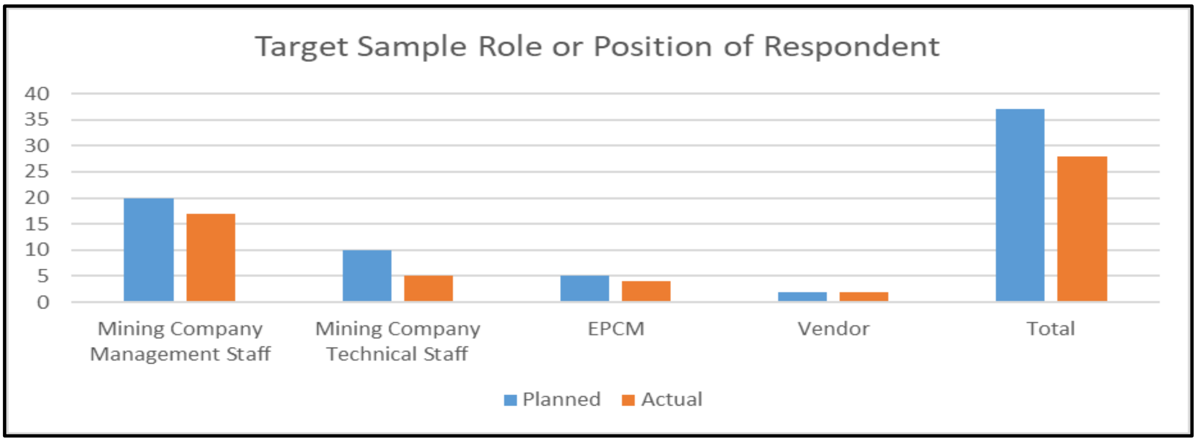

Figure 29: Category of respondents according to position and role in the organisation

In figure 30 the questions tested the respondent's knowledge of the use of key factors for business case development in the mining industry.

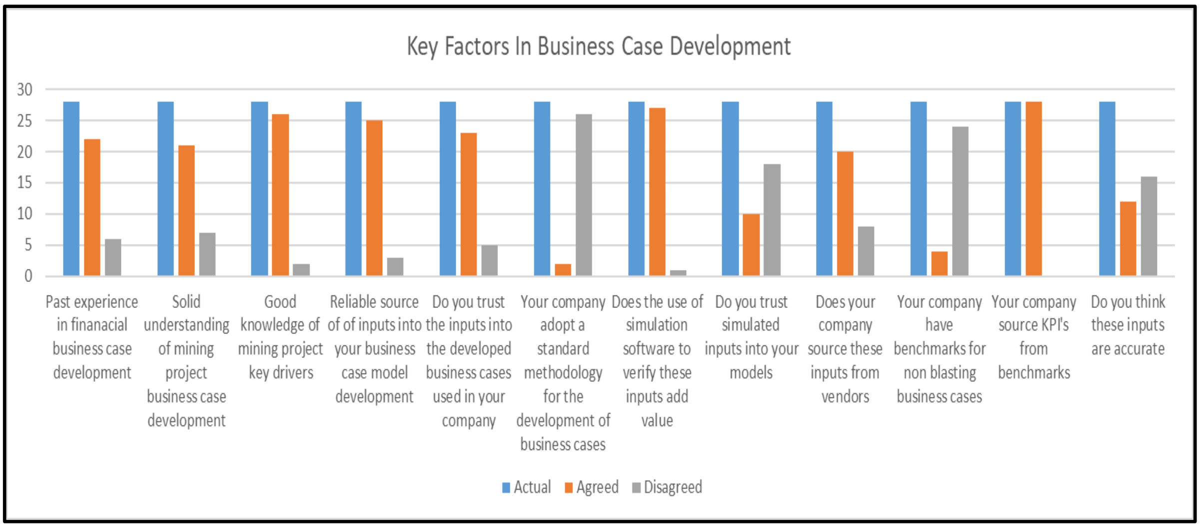

Figure 30- Key Factors that Influence Business Case Development

The key indicators for making decisions to select a viable mining investment was analysed and the outcomes are shown in the bar graphs in figure 31 below.

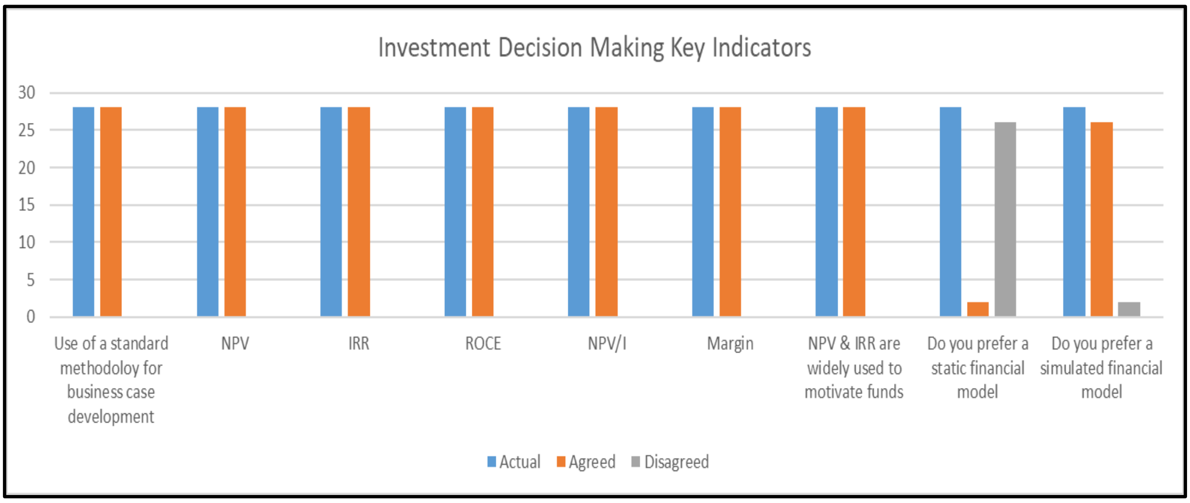

Figure 31- Key Investment Decision Making Indicators

In figure 32 the respondents were asked about some of the key factors for the decision to used mechanised mining methods in a chosen mining project.

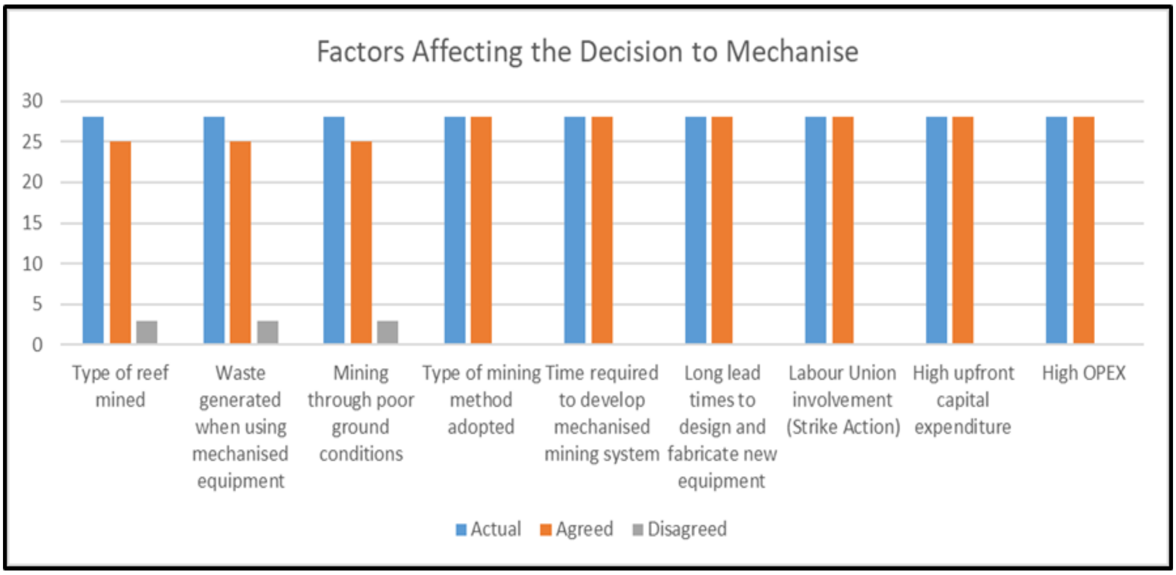

Figure 32: Factors affecting the decision to mechanise

Various factors were posed to the respondents and the answers were analysed in the form of the bar graph as shown in figure 32.

Chapter 5 – Discussion

5.1 Preamble

Investment in mining projects is extremely capital intensive and very risky (Botin et al,2011). Investors in mining projects will not invest in mining projects if it does not show a positive NPV and an IRR above the company required weighted average cost of capital (WACC) (Smith,2011). According to the discounted cash flow (DCF) method, which is the most widely used method for evaluating mining projects, net present value is generally calculated as an indicator of project profitability. The data acquisition process for developing business cases for new technology mining methods is reliant on either limited proof of concept testing or original equipment manufacturers sources.

5.2 Re-cap of Existing Economic Evaluation Process for Investment Funding

The process that is usually followed to develop a mining investment business model entails conducting detailed studies in the following areas of resource development, Geology, Resources estimation, Reserve estimation, Geo-technical, Mine design, Process design and logistics, Infrastructure, Environmental, Capex & OPEX estimation, Construction, Land and Legal, Marketing and finally the Financial Evaluation. These financial models which are usually developed by an approved competent engineering company are compiled using tried and tested mining methods and are usually too narrow or one dimensional in nature Smith, (2011). This one-dimensional analysis usually outputs a narrow range of cashflow and revenue calculations and this in turn could lead to an over or underestimated NPV and IRR calculation. For this specific reason the output of this study would like to suggest the framework as depicted in figure 10 to be adopted for future investment analysis.

5.3 Discussion of Results

5.3.1 Production Profiles

In underground mining operations, production ramp-up can only commence after the mining infrastructure, such as access shafts and development to prepare the orebody for steady state production is completed. This period from the time the overburden is cleared to the start of ramp-up could take between 18 to 36 months depending on the selected mining method (Extractive Hub,2020). In this phase of project implementation cash is spent by the company with no return or net positive cash-inflows (Ivanhoe Mine News,2017).

For this study new mining layouts were developed to cater for the different mining methods and cutting technologies adopted in the shaft area, stations, decline barrel, working cost development and stoping. The entire mining operation for the different scenarios were modelled in SimMine. This was done in order to compare the impact of using different combinations of cutting machines in underground mining. Due to the size of these simulation models and the limited processing capacity of the IT technology presently the options were only run three times per mining method, based on a triangular statistical distribution, using the minimum, maximum and mean input values (Botha, et el, 2010). There were four different scenarios for the off- reef options thus 12 runs were executed. The on-reef scenarios had 6 different mining options, this equated to 18 runs. In total 30 runs were executed for the entire study in order to arrive at the final production and grade profiles as presented in chapter 4.

For off-reef options, in Scenario C02 the mechanical blind shaft boring technology for sinking the shaft was planned in the models. The stations, project development, declines, working cost development and stoping was planned to be mined using conventional drill and blast methods. Scenario C03 was planned using mechanical blind shaft boring technology for shaft sinking, the MTM machine for the shaft stations and project flat development, a TBM in the declines and uses conventional drill and blast methods for the working cost development and stoping. Scenario C04 was planned using mechanical blind shaft boring technology for shaft sinking, the MTM machine in the shaft stations and project flat development, a TBM in the declines, RMDS for the working cost development and uses conventional drill and blast method in the stopes. The introduction of the MTM, TBM and RMDS machines in the station areas, declines and working cost development has moved the ramp up forward by a year for C04.

For on-reef options, scenario M01a was planned using mechanical blind shaft boring technology for sinking the shaft, MTM (4.7m by 4.7m) for the shaft stations and project flat development, a TBM (6m diameter) for the decline, RMDS (4.5m by 3m) for the working cost development and the slot boring machine in the stopes for reef extraction. Scenario M01b was planned using mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the shaft stations and project flat development, TBM (6m diameter) for the decline, RMDS (4.5m by 3m) for the working cost development and the slot boring machine in the stopes for reef extraction. Scenario M03 was planned using mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the shaft stations and project flat development, decline and working cost development and the slot boring machine in the stopes for reef extraction. Scenario M04 was planned using mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the shaft stations and project flat development, decline and working cost development (4.5m by 2.2m) and the ULP fleet in the stopes for reef extraction.

Scenario M05 was planned using mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the shaft stations and project flat development, decline and working cost development (4.5m by 2,2m) and conventional handheld drills and blasting in the stopes for reef extraction. Scenario

M06 was planned using mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the shaft stations and project flat development, decline and working cost development (4.5m by 2,2m) and the Sandvik MN 220 hard rock cutting machine in the stopes for reef extraction. Scenarios M01b, M03, M04 and M05 have the same project start dates. Scenario M05 has a slower ramp up as the decline and working cost development have been slowed down to accommodate the slower stoping rates. The spike at the end of the grade profiles of M01, M01b, M03, M04 and M06 is due to the development finishing before the stoping which has a higher grade.

If one analyzes the output as depicted in figure 21 it can be noticed that the planned production profiles ramp-up differs for the various mining options modelled. The ramp-up of the base case to steady state takes about 12 years including access development, this implies that capital is spent by the organization in the first 6 years with no return on that investment because in this mining method all the infrastructure construction is carried out off reef or in the waste rock plane. On the other hand, it can be noticed also from figure 21 that as the hard rock cutting technologies are planned into the models the ramp-up to steady state gets shorter due to the speed of development. The off reef developed options ramp-up to full production varies between 12 months to 3 years as a result of the hard rock cutting machines been phased into the development and stoping operations. On the other hand, it is noticed that for the on-reef options where all infrastructure is constructed on the reef plane the ramp-up time to full production varies between 1 year to 3 years.

5.3.2 Business Case Cost Modelling

Once these production tonnages were modelled and verified the financial evaluation process commenced. The costing of the various activities and resources related to the modelled production profiles was compiled as per the cost structure shown in table 12. The various other resources are then allocated to deliver on this optimized planned production profile. The final roll-up of all the different cost buckets (table 12) are then cash flowed to the various years as per the production profile shown in figure 24 for the off-reef options and figure 26 for the on-reef options. It can be noticed that the cashflows of the Off-reef options is negative in the first 9 years as cash is spent to develop the access shafts and for the purchase of mining equipment. After the first 9 years, the first revenue starts to flow into the company as the stoping starts to extract reef tons.

The revenue streams for the off-reef options are shown in figure 22, it can be noticed that the revenue is compared to the base case, as the mining scenarios start to implement the rapid access development machines the stoping starts sooner and the income starts to flow into the company much earlier. This can be seen to have a direct impact on the NPV and IRR of the project. Figure 26 depicts the movement of the business case for the off-reef options that went from -3.39 billion-rand NPV to -2.8 billion-rand

NPV. Although this is still negative, due to the lag in time to get to cut-off grade before positive cashflows and other KPI's such as metal price and exchange rate there is still a positive trend with regard to value add as soon as the non-explosive mining methods were used in the construction of the mine and also for stoping. This figure also shows the value advantage of rapid mine development for underground hard rock mining. Whether one uses, on or off reef mining the quicker one can get to the cut-off grade the quicker one can start getting income into the organization.

In figure 27 it is illustrated that mining in the PGM sector should adopt a non-explosive type of mining method in order to shorten ramp-up time. The build-up to these business cases shows that if one starts the mining operation in an on-reef design format the cashflow starts as soon as the mine starts with the development of the infrastructure. As capital is spent on the purchase of equipment etc, revenue can flow into the organization to start paying back the capital spent from an early start of the project. The on-reef options all demonstrate an improvement to the NPV and IRR of the project. The only difference in these development options are the type of rock cutting technology used. As the options start using the Rapid Mine Development system combined with the slot boring machine the business case start showing an improvement to the bottom line. The initial cashflow show that large sums of capital are spent upfront on the purchase of these rock cutting machines but because the development is completed sooner, cut-off grade is also reached sooner and thereby impacts the cash inflows into the company. The revenue stream from these options is pushed forward and this also has a positive impact on the NPV and IRR of the project.

These models as stipulated in table 2 and table 3 demonstrate that as one introduces non-explosive mining methods into an underground mining operation the business cases of these operations improves. These models considered both off-reef and on-reef options in order to cover as much of the variability experienced in hard rock mining. Simulation models helped by considering the mining operation as a complete system. These simulation models took into consideration the total mining cycle from shaft sinking to final transport of the ore to the concentrator plants. The simulations considered the variability of the cutting rates to other operational bottlenecks and these were built into the complete system in order to arrive at an optimum production profile as shown in the framework depicted in figure 10. The algorithms built in the simulation software catered for the operating time, operational stops, consequential delays, Unscheduled maintenance, controllable time, scheduled maintenance, and noncontrollable time. This helped with the verification of the inputs into the model and improved the confidence of the final output of the production profile.

Once the integrity of the outputs could be verified and validated the study could then go on to apply the financial benchmarks for the supplied production profile. The algorithms that are built into the HSF system ensured that the outputs from the HSF system were of a high integrity. These outputs from the

HSF system depicted an accurate NPV and IRR of the various options. The study thus used various validation techniques to ensure that this comparison between the drill and blast business cases could be compared to the non-explosive business cases. The models demonstrated that non-explosive mining methods can have an impact on the NPV and IRR of an underground PGM mining operation. This also demonstrated that one of the key value drivers in an underground PGM mining operation is speed to getting to paying ore or cut-off grade. The framework shown in figure 10 encapsulates all these principles in a single process that can be followed to develop any bankable PGM mining study.

5.4 Discussion of Assumptions

During this study various assumptions were made by the author in order to arrive at the business case outcomes as illustrated for the options studied in table 2 and 3. A selection of some of the key assumptions are considered in this section. These include the selected distribution function, source of model input data, grade profile and survey questions. The simulation models run for the various options were modelled using a triangular distribution. Triangular distributions are normally used when the researcher does not have a good understanding of the shape of the probability distribution, but knows the endpoints (Mun,2008). For this study it was known what the endpoints were and that the probability distribution was not uniform (i.e. the end points are less probable than the mean), so the Triangular Distribution was chosen (Botha et el,2010). Some advantages of this distribution are (1) the math for the probability density function is easy, (2) the domain of x is restricted to the interval [a,b] and (3) the distribution is defined solely by its first two moments (i.e. the distribution is symmetric) (Schurman, 2011). Due to the simplicity, basic equations, and by considering that the variables (min, max and mode) originate from the opinion of experts and no physical historical data, the triangular distribution has not been considered a reliable statistical representation (Weissberg et el,2005). As a result of these disadvantages, therefore it could be stated that there are certain limitations in this research results.

For instance the values used for the VBBM to develop the main shaft were modelled using the minimum value, 0.256m/hr, and at a maximum advance rate of 1m/hr, using the triangular distribution one can arrive at the mean value which, in this case was 0.51m/hr of advance. Table 76 and 77 Appendix 2 shows the details of the inputs for the development of the MN220 cutting models. These values were obtained from the OEM and was calculated from the machine design parameters. As these non-explosive mining machines were not physically applied on an operation, confidence in this data inputs are reduced, and this could have a negative impact on the simulated production profiles as shown in figure 21. These values were used as inputs into the SimMine models with a combination of the other values such as number of shaft boring machines, 2 -off, number of crews operating the machines, 2 -off, availability of 75% and shift working times used was 24 hours, this combination of machines and operations crews yielded an advance rate of 9.2m/day for the one option.

It must be highlighted that the simulation model input data were either calculated using machine design parameters as obtained from the OEM's or from consultant expert opinions, none of the data used in the simulation models for the non-explosive mining equipment could actually be verified in a physical mining operation. The main access to the underground operations was planned through the vertical shaft system so if these advance rates were incorrectly assumed the production profiles could be incorrectly stated. As stated earlier in this study the production profiles are direct inputs into the development of the financial models for the economic analysis of the various mining options.

Another assumption that must be highlighted at this point is the grade profile. The grade inputs into the final revenue calculation were obtained from the Merensky reef resource statement that is usually published by the consulting company, which included a resource area of 8.97Mm^2, resource cut of 1.57m, density of 3.34g/cm^2, geological losses of 26.23%, tonnage after geological losses of 1.84Mt, 4E grade of 5.84 g/t, content of 4E ounces of 0.35Moz and confidence level of tonnage of 4% as a measured resource category. The reserve calculations were based on table 5 outputs, and the modifying factors used was as per table 6 and 9. The geological and final grade distribution profile was assumed at a 4% measured, 57% indicated and 39% inferred confidence level as supplied by the appointed professional engineering consultants. While these grade inputs were assumed as typical, details of exact grades are not publicly available and so only estimates through expert opinions can be used. Parameters outside the values used here were not considered, hence may or may not be applicable to the results of the study. In addition, these may not be generalised as profiles vary widely from reef to reef, gradient to gradient and area to area.

The list of survey questions was developed initially for a face to face informal process of administration but due to time constraints these questions were then formulated into a Jotform online format as shown in appendix 1, a screen copy of the exact questions in Jotform. These questions were tested in a face to face format with certain mining production and project managers as a form of verification, however the Jotform format was not tested before submitting it to the various recipients. The answers from the Jotform process was then reformatted to filter out the information related to new technology, simulations and the use of the various financial indicators adopted in the mining industry for investment decision purposes. It is possible that there were respondents that did misunderstand some questions, although precaution was taken to check each response for obvious errors.

Chapter 6- Conclusion and Recommendations

6.1 Preamble

Organizations developing hard rock cutting technologies have made major advances towards a complete hard rock mining system. The only obstacle in its path to complete deployment in an underground hard rock mining environment is investor scepticism toward this type of technology. Investors are extremely cautious when investing capital in mining operations that do not utilize tried and tested mining methods (Smith,2011). The risks associated with the deployment of rock cutting technology compared to drill and blast can be quantified, if the appointed consulting engineers adopt the methodology of developing financial models using inputs sourced from simulation software such as SimMine.

6.2 Comment on Key Findings

6.2.1 Mining Method Selection For this Study

Important natural constraints were considered when selecting the mining methods for this study, these are explained in this section. Once an ore body has been probed and outlined and enough information has been collected to warrant further analysis, the important process of selecting the most appropriate method for mining can begin. With respect to the basic principles employed, relatively few mining methods are used in today's mining operations. This is as a result of the uniqueness of each ore deposit and variations in these mining methods being limitless (Hamlin,1998). Some of the underground mining methods include room and pillar, block caving, sub-level stoping, sublevel caving, longwall, shrinkage stoping, cut and fill, big hole stoping and vertical crater retreat. Due to the general Platiniferous nature of the reefs uncounted in platinum mining a scattered mining approach is usually selected with a breast mining layout for stoping the panels (Kendall, et al,2000). Some limited number of ore bodies are also conducive to room and pillar mining methods, but this mining method requires the orebody to not dip more than 9 degrees. Considering these naturally imposed constraints narrow tabular mines generally adopt either an off-reef or on-reef scattered breast mining method. This study was limited to various configurations of the scatter breast mining method for the on and off reef layouts.

6.3 Objectives, Research Question and Purpose Statement

This study has answered the central research question as to what the financial benefit would be when adopting non-explosive mining methods compared to drill and blast. This was demonstrated by comparing the base case which was developed on a drill and blast mining method with various non-explosive mining methods. The study used a simulation software to arrive at the inputs into the development of the mine production profile and subsequently the financial models. This was done to increase the confidence of these inputs to calculate the key financial indicators of the various non-explosive mining options compared to drill and blast mining.

In order to increase the confidence in the outcomes the study took into consideration two different mining concepts to exploit this orebody. This was in the form of on-reef and off-reef mining scenarios, by doing this the study could ensure like for like options were compared. The options run were the mine development was done in the foot wall or Off-reef, which is numbered C02, C03, C04 in the table 3, yielded the NPV's and IRR's as stated in table 17. This showed that all the options were negative and showed an IRR that was below the company's investment WACC. On the other hand, these calculations also highlight the value add to the business as non-explosive mining systems were phased into the mine construction and stoping operations. This showed that the NPV got better with the addition of hard rock cutting systems. Table 19 shows the value add of mining directly on reef and the impact that non-explosive mining methods has on the final business case. As the options moved towards total mechanization the NPV got better and the IRR showed a positive business case. This was well over the WACC for investment in this mine.

The above values depict the incremental value add of using non-explosive mining methods compared to drill and blast options. The base case in the on-reef option showed a negative NPV but by incrementally using non-explosive rapid mining methods it could be seen that the financial indicators showed a positive business case. Option M06 showed a high NPV and a good IRR, which was well above the WACC for this mining operation.

6.4 Limitations of this research

This study was limited to only a vertical mine access system. The study did not consider a decline access system and open cast mining operations. The study also only considered a limited variety of hard rock cutting equipment, since the completion of this study many new types of hard rock cutting equipment were introduced into the market. As this is a very high-level study of this specific mining operation more detailed consideration should be taken of the overhead structure for a modern type of mining operation.

The study used few points per reading to arrive at a proposed statistical distribution, namely the triangular distribution. It is unlikely that this is the most optimum distribution to use since more readings may have led to a more realistic distribution. The sourcing of data from machine design parameters and expert opinion were limiting as only a few OEMs could provide data and non-explosive mining equipment could not be verified. Results are hence limited to only the specifications used.

Although a face-to-face format of questionnaire was pre-tested, a pilot test for the survey questionnaire format was not done. It is possible that some questions were misunderstood, hence reducing the confidence of the results. Grade profiles cannot be generalised as variations are vast and not representative in this study. Results are limited to profile specifications made.

While this study has demonstrated evidence of benefit in using non-explosive mining compared to explosive mining by using an incremental approach whereby more and more non-explosive components are incrementally added and outcomes measured, the study cannot confidently provide an accurate measure of the figures by which one method is greater than the other. Only an indication can be reasonably claimed by this study.

6.5 Recommendations

Although the study analysed various mining options and considered on-reef and off-reef underground mining methods there is still a need to study the following:

- A detailed study into the complete mining value chain using simulation for new technology mining method.
- What would be the best process to follow if one would conduct a bankable feasibility for a new technology mining project.
- Development of a mining system simulation, that can instantaneously calculate the NPV and IRR of a new technology mining project as economic conditions internal or external to the project changes over time.
- There are numerous variables that make up the development of mining project business cases, one would need to model this as a complete solution in future.
- What would be the impact of introducing swarm robotic mining systems into underground mining projects
- The mine must be designed from the onset to cater for hard rock cutting technology and not be based on drill and blast layouts.

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8.0 Appendix 1: Questionnaire

Mining Business Case Development

1.Do you work for a mining company?

- [x] Yes
- [ ] No

2.What type of commodity does your company mine?

Platinum

3. Where is your company based?

South Africa

4. What is your role in your company?

Project Manager

5. Have you developed mining project business case?

- (•) Yes
- ( ) No

6. Do you understand the process that is followed to develop a business case for a mining project?

- [x] Yes
- [ ] No

7. What are the key inputs in the development mining project business case?

Geology, Geological structure, resource statement, grade profile, mine design, mining method, production profile, cost benchmarks

8. Where do you source your inputs to your business cases?

In house company database

9. Do you trust the inputs into the developed business cases?

- [x] Yes
- [ ] No

10. Does your company adopt a standard methodology for the development of business cases?

- [ ] Yes
- [x] No

11. Does your company use any simulation software to verify these inputs?

- [ ] Yes
- [x] No

12. Do you prefer simulated inputs or industry benchmarks?

- [x] Yes
- [ ] No

13. Do you think the inputs are the same for a non blasting vs blasting mining method business case?

- [x] Yes
- [ ] No

13. Do you think the inputs are the same for a non blasting vs blasting mining method business case?

- [ ] Yes
- [x] No

14. Does your company have benchmarks for non blasting business cases?

- [ ] Yes
- [x] No

15. Where does it source this input from?

OME's

16. How accurate do you think these inputs are?

Not very accurate

16. How accurate do you think these inputs are?

Not very accurate

17. Would you prefer to use a standard methodology for business case development?

I do

18. Which of the following have you come across before?

- [x] Net Present Value
- [x] Internal Rate Of Return
- [x] Return on Capital Employed
- [x] Net Present Value/Investment
- [x] Margin
- [ ] Other

- [x] Return on Capital Employed
- [x] Net Present Value/Investment
- [x] Margin
- [ ] Other

19. Which of the above do you prefer?

NPV and IRR

20. Which do you think is the most accurate depiction of the business case?

NPV and IRR

Next

21. What type of reef does your company mine

UG2

22. Amount of waste generated when using mechanized equipment

11%

23. How does your mining team mine through poor ground conditions

Log this as waste

24. What type of mining method does your company adopt

Bord & Pillar

25. How long does it take to develop new technology

5 to 10 years

26. How long does it typically take to design, fabricate new mining equipment

26. How long does it typically take to design, fabricate new mining equipment

5 years

27. When does your company involve the labour unions and DMRE

Early in the process

28. Does high upfront capital impact the decision to invest in new technology

Yes it does

29. Does high OPEX impact the decision to adopt new technology in mining

Yes it does

30. Do you prefer to use static excel based models or dynamic simulated models

Dynamic

31. Do you trust the use of simulated inputs in you financial models

Yes I do

9.0 Appendix 2 – Cutting Machine Simulation Model Inputs

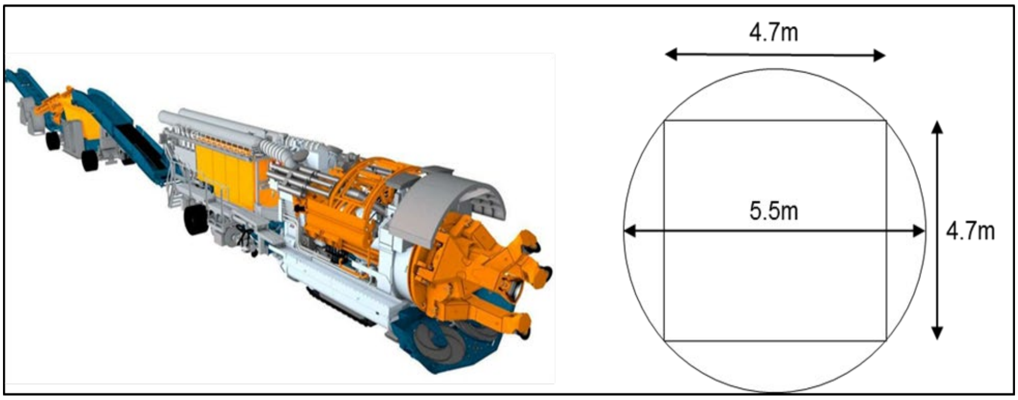

Figure 33- Aker Wirth Mobile Tunnel Miner (Van Der Berg D. 2014)

Table 21- MTM machine specification and parameters used (Van Der Berg D, 2014)

Parameters used (Mine Scenario Planning)		
	Unit	**MTM**
Shape		Circle/square
Width	m	4.7
Height	m	4.7
Advance	m/month	138
Aker Wirth – MTM6		
	Unit	**MTM**
Cutting Arms		6
Turning radius	m	30
Machine Length	m	75
Cutting Radius	m	6.2

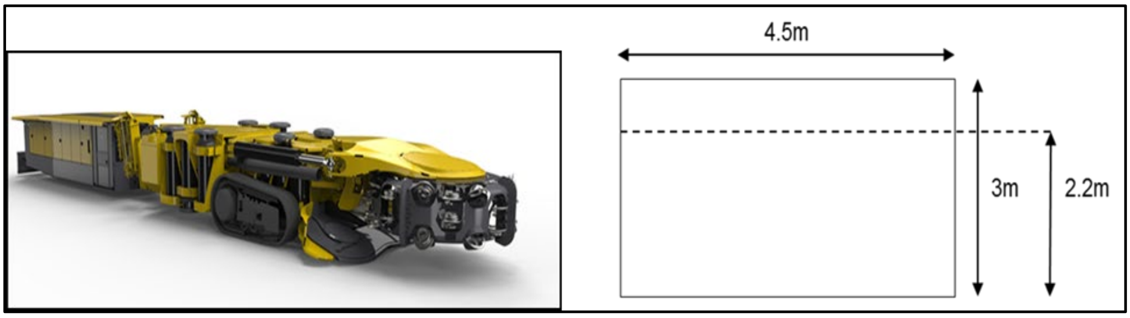

Figure 34 - Atlas Copco Rapid Mine Development System (Janicijevic and Valicek, 2015)

Table 22- RMDS machine specification and parameters used (Janicijevic and Valicek, 2015)

Parameters used (Mine Scenario Planning)		
	Unit	**RMDS**
Shape		Rectangle
Turning radius	m	25
Width	m	4.5
Height	m	3 or 2.2
Advance	m/month	300

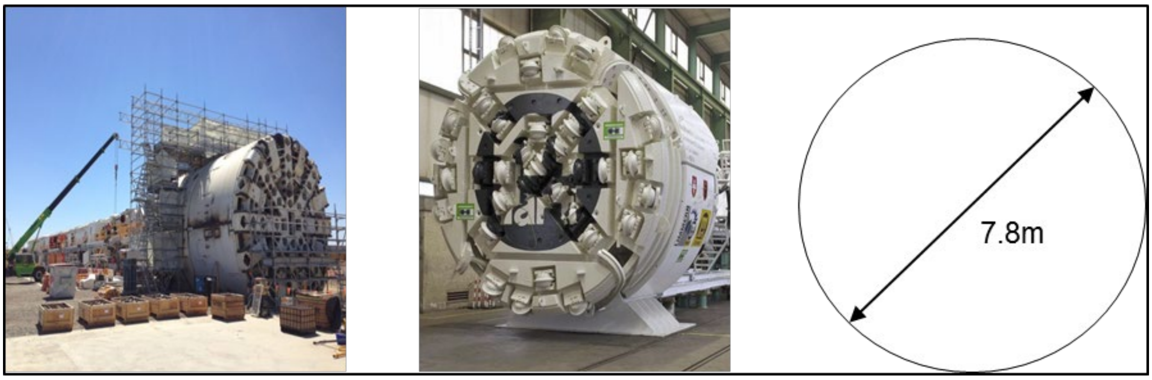

Figure 35- Tunnel Boring Machine (Van Der Berg D, 2014)

Table 23 - Tunnel Boring Machine Specifications and Parameters Used (Van der Berg D, 2014)

Parameters used (Mine Scenario Planning)		
	Unit	**TBM**
Shape		Circle
Turning radius	m	75
Diameter	m	7.8
Advance	m/month	270
Robbins and Herrenknecht		
	Unit	**TBM**
Turning Radius	m	150
Length	m	150
Cutting Diameter	m	As per client

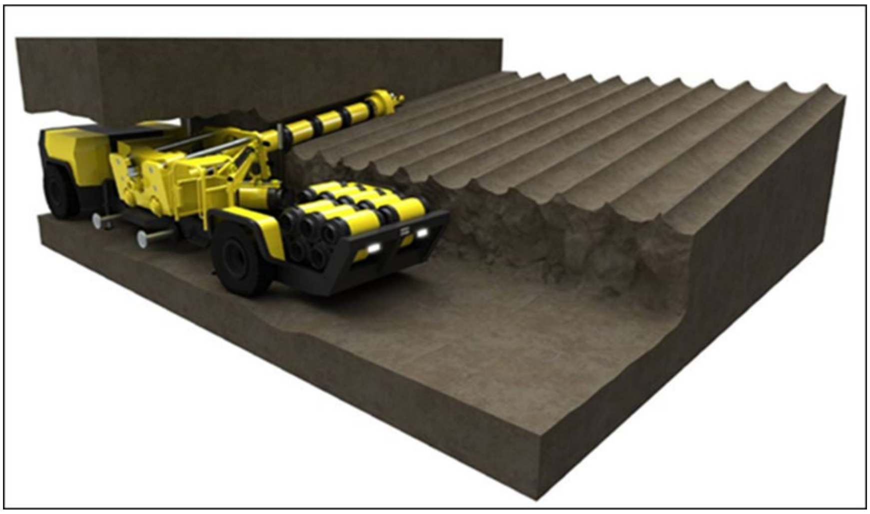

Figure 36 - Atlas Copco Slot Borer (Janicijevic and Valicek, 2015)

Table 24- Slot Borer specification and parameters used (Janicijevic and Valicek, 2015)

	Unit	Value
Hole diameter	m	1.25
Hole length	m	20
Hole spacing	m	1
Height of machine	m	3
Mass of excavated material from one hole	t	79
Time to excavate one hole	h	5.1
Operating hours per day	hr/day	13.5
Daily excavation rate over 24h:2.4 holes	t	190
Advance	m^2/month	1 327

Figure 37- Ultra Low Profile (ULP) Equipment (Fourie F, 2014)

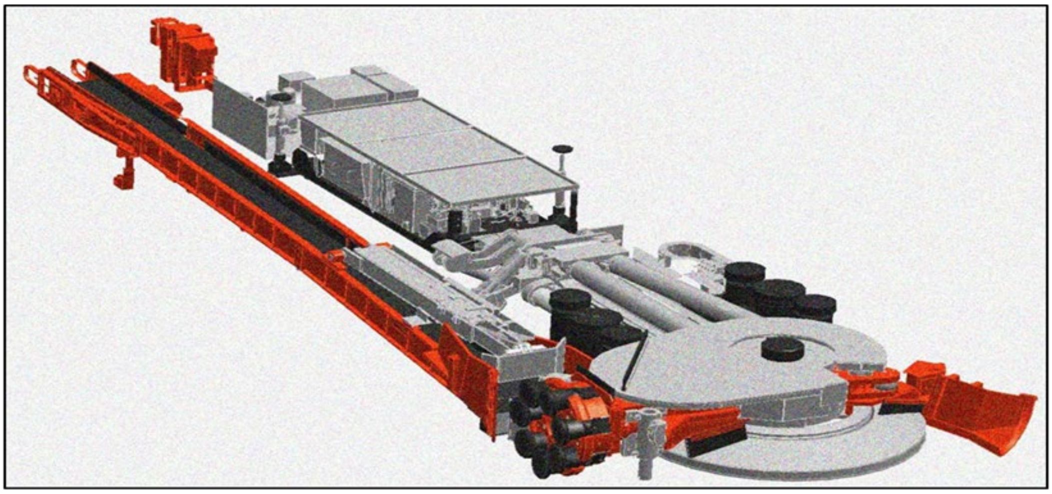

Figure 38- MN220 Machine (Pickering R, 2006)

Table 25 - MN220 Parameters Used in Study

Parameters used (Mine Scenario Planning)		
	Unit	MN220
Shape		Rectangle
Turning radius	m	50
Width	m	4.3
Height	m	1.1
Advance	m/month	600

10.0 Appendix 3- Model Inputs Per Option

Table 26- Option C01 Model Inputs

Option C01		
Crew	Number of Crews	Locations (s)
Shaft sinking crew	2	Main Shaft, Service Shaft
Drill and blast crew	6	Barrels (Declines)
Drill and blast crew	2	Station Area
Drill and blast crew	84	Haulage
Conventional crew	19	Ledge
Conventional crew	119	Stope (8 crew/half level)

Table 27- Option C02 Model Inputs

Option C02		
Crews	Number of Crews	Location (s)
VBBM	2	Main Shaft, Service Shaft
Drill and blast crew	6	Barrels (Declines)
Drill and blast crew	2	Station Area
Drill and blast crew	85	Haulage
Conventional Crew	19	Ledging
Conventional Crew	116	Stope (8 Crews/Half level)

Table 28- Option C03 Model Inputs

Option C03		
Crew	Number of Crews	Location (s)
VBBM	2	Main Shaft, Service Shaft
TBM	3	Barrels (Declines)
MTM	2	Station Area
Drill and Blast Crew	96	Haulage
Convention Crew	19	Ledging
Conventional Crew	116	Stope (8 Crews/Half level)

Table 29- Option C04 Model Inputs

Option C04		
Crew	Number of Crews	Location (s)
VBBM	2	Main Shaft, Service Shaft
TBM	3	Barrels (Declines)
MTM	2	Station Area
Drill and Blast Crew	10	Haulage
Convention Crew	18	Ledging
Conventional Crew	113	Stope (8 Crews/Half level)

Table 30- Conveyor Capacity

Conveyor	Units	Value
Decline Conveyor	t/hour	700
Strike Conveyor	t/hour	180

Table 31- Rock Cutting Machine Inputs

Cutting Machine	Shift Working Time	Availability	Cutting Rate	Min/m	Adv/day	Days/ Mnts	m/Mnts
VBMM-Main Shaft	24 hrs	75%	0.51m/hr	117.6	9.2m/day	26	240
VBBM-Service Shaft	24hrs	64%	0.51m/hr	117.6	9.2m/day	26	240
TBM	12hrs	75%	1.56m/hr	38.5	12m/day	23	270
MTM	12hrs	75%	0.67m/hr	90	6 m/day	23	138
RMDS-235	12hrs	75%	1.14m/hr	52.9	10.2m/day	23	235
RMDS-262	12hrs	75%	1.27m/hr	47.4	11.4m/day	23	262
Slot Borer	16hrs	75%	3.92m/hr	15.3	48 m/day	23	1104

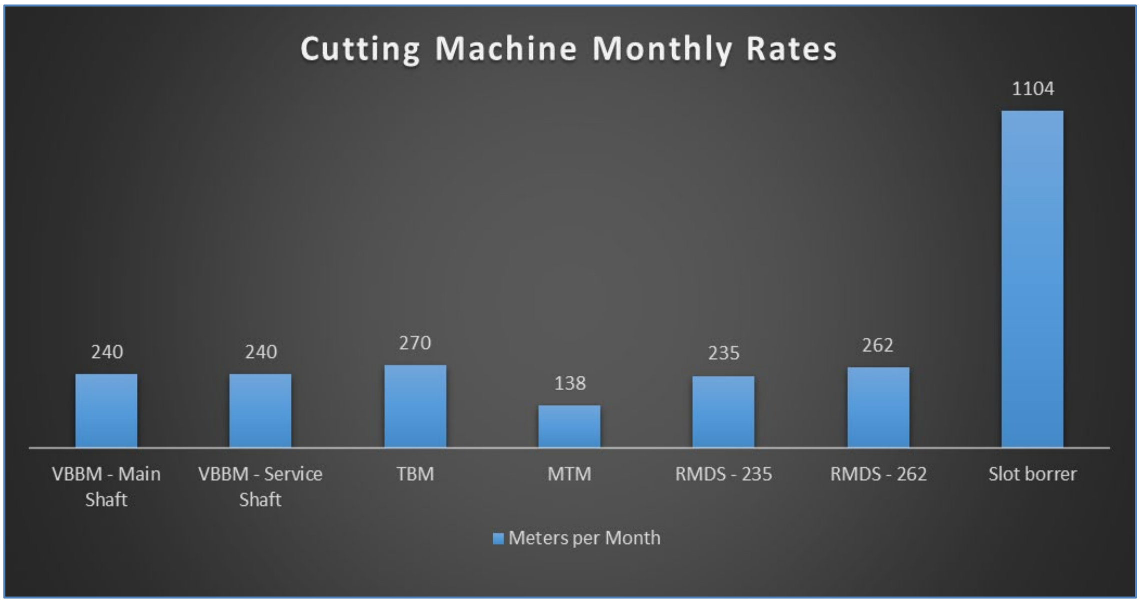

Figure 39- Cutting Machine Monthly Rates

Table 32- Option M01a Model Inputs Cutting Machines

	Option M01a	
Cutting Machine	Number of Machines	Location(s)
VBBM	2	Main Shaft, Service Shaft
TBM	3	Barrels (Declines)
MTM	2	Station Area
RMDS	20	Strike Transport Drive, Strike Belt Drive and Strike Drive
Slot borer	27	Stopes

Table 33 - Option M01b Model Inputs

	Scenario M01b	
Cutting Machine	Number of Machines	Lotion(s)
VBBM	2	Main Shaft, Service Shaft
TBM	3	Barrels (Declines)
RMDS	2	Station Area
RMDS	20	Strike Transport Drive, Strike Belt Drive and Strike Drive
Slot borer	27	slopes

Table 34- Option M03 Model Input

	Scenario M03	
Cutting Machine	Number of Machines	Location(s)
VBBM	2	Main Shaft, Service Shaft
RMDS	3	Barrels (Declines)
RMDS	2	Station Area
RMDS	20	Strike Transport Drive, Strike Belt Drive and Strike Drive
Slot borer	23	Slopes

Table 35- Option M04 Model Inputs

	Scenario M04	
Cutting Machine	Number of Machines	Location(s)
VBBM	2	Main Shaft, Service Shaft
RMDS	3	Barrels (Declines)
RMDS	2	Station Area
RMDS	20	Strike Transport Drive, Strike Belt Drive and Strike Drive
ULP Fleet	12	Ledge and Stopes

Table 36- Option 23M05 Model Inputs

	Scenario M05	
Cutting Machine	Number of Machines	Location(s)
VBBM	2	Main Shaft, Service Shaft
RMDS	3	Barrels (Declines)
RMDS	2	Station Area
RMDS	20	Strike Transport Drive, Strike Belt and Strike Drive
Conventional Crews	23	Ledge
Conventional Crews	140	Stones

Table 37- Development Locos Inputs

Development Locos	Units	Average
Capacity	1	35
Waste Loading Time / Span	min/span	30
Waste Tipping Time / Span	min/span	8.0
Loco Speed	km/h	5.5

Table 38- Handheld Bolting Specification

Handheld Bolting Specification	Units	Average
Bolting Time	Minutes	84
Bolter Availability	%	85

Table 39- Handheld Drill Specification

Handheld Drill Specification	Units	Average
Drill Length	meters	0.9
Drill Time	minutes	313
Drill Availability	%	85

Table 40- Charger Specification

Charger Specification	Units	Average
Charging Time	minutes	60
Charger Availability	%	85

Table 41- Drill Rig Specification

Drill Specification	Units	Average
Drill Length	meters	1.8
Drill Time	minutes	130
Drill Speed	km/h	3
Drill Availability	%	85

Table 42- Bolting Specification

Bolting Specification	Units	Average
Bolting Time	minutes	120
Bolter Speed	km/h	3.0
Bolter Availability	%	85

Table 43- Dozer Specification

Dozer Specification	Units	Average
Capacity	t/h	70
Dozer Speed	km/h	5.5
Dozer Availability	%	85

Table 44- Charger Specification

Charger Specification	Units	Average
Charging Time	minutes	60
Charger Speed	km/h	6.0
Charger Availability	%	85

Table 45- Development Locos

Development Locos	Units	Average
Capacity	t	35.0
Waste Loading Time / Span	min/span	120
Waste Tipping Time / Span	min/span	8.0
Loco Speed	km/h	5.5

Table 46- Bolting Specification

Bolting Specification	Units	Average
Bolting Time	minutes	84
Bolter Speed	km/h	3.0
Bolter Availability	%	85

Table 47- Drill Rig Specification

Drill Specification	Units	Average
Drill Length	meter	3
Drill Time	minutes	153
Drill Speed	Km/h	3
Drill Availability	%	85

Table 48- Charger Specification

Charger Specification	Units	Average
Charging Time	minutes	60
Charger Speed	km/h	6.0
Charger Availability	%	85

Table 49- Loader Specifications

Loader Specification	Units	Average
Bucket Capacity	t	8
Bucket Fill Factor	%	80
Loading Time	S	20
Tipping Time	S	30
Loader Speed	km/h	5.5

Table 50: -Merensky Reef Resource Statement

MERENSKY REEF										
RESOURCE BLOCK	**RESOURCE CATEGORY**	**DIP (°)**	**DIP AREA (M m^2)**	**RESOURCE CUT (m)**	**DENSITY (g/cm ^{3})**	**GEO LOSS %**	**TONNAGE AFTER GEO LOSS (M/tons)**	**4E GRADE (g/t)**	**CONTENT (4E MOZ)**	**CONFIDENCE % TONNAGE**
2.7	Measured	8.0	0.48	1.57	3.34	26.23	1.84	5.84	0.35	4
	Indicated	8.8	8.69	1.25	3.38	25.45	27.31	7.13	6.26	57
	Inferred	9.2	6.48	1.16	3.36	25.16	18.92	7.53	4.58	39
	Total	**9.0**	**15.65**	**1.22**	**3.37**	**25.36**	**48.07**	**7.24**	**11.19**	**100**

Table 51: Off Reef Development parameters

Sub-Activity	Width(m)	Height(m)	Length(m)	Efficiency(m/Mth)	No. of Ends
Stations					
Personnel Ramp	5.5	2.1	153.1	45	2
Connecting X-Cut	5.6	2.1	75.1	45	2
Personnel Holing	6.5	2.1	130.1	45	2
FW Assembly Bay	8	5	5	45	2
Workshop East	5.6	3.5	51.7	45	2
FW Workshop East	5.6	3.5	80.2	45	4
594 Refuge Bay	4	4	20	45	3
S# X-Cut S	5.6	3.5	60	45	3
M# C-Cut S	5.6	3.5	53.3	45	2
Secondary Haulage	5.6	3.5	221.3	45	2
Access X-Cut 2B	3.5	2.5	84	45	4
Conveyor Hlg Free	5.6	3.5	240.8	45	2
MCC 3	6	3.5	49.5	45	5
Settler Feed	5.6	3.5	168.5	45	4
Main Artery	5	4	74.9	45	2
594 Refuge Bay	4	4	20	45	3
Settler Haulage	7.22	3.5	264.8	45	2
Settler Haulage	7.22	3.5	143.2	45	2
Settler Haulage	7.22	3.5	277.3	45	2
Sub Station X-Cut	5.6	3.5	6	45	2
Spill Load Ramp	5.6	3.5	24	45	2
Settler Haulage	5.6	3.5	6	45	2
Silo Bottom 3	6.5	3.5	114.5	45	7
Boxhole Pilot	4	diameter	496.1	45	7
Shaft Bottom Ramp	4	3.5	422.2	45	1
Shaft Bottom Ramp	4	3.5	24	45	3

Table 52 - Decline Development

Sub-Activity	Width(m)	Height(m)	Length(m)	Efficiency(m/Mth)	No.of Ends
Declines					
M_Dcl	4	3.5	214.5	45	1
Acc_LvL	4	3.5	213.6	45	1
L_Mpass	3.54	3.54	27.3	35	2
HLG	3	3.3	318	45	2
C_Dcl	4	3.5	180.5	45	1
B_Dcl	6.5	3.5	215.3	45	2
Conn_Dcl	4	3.5	133	45	2
T_way	2.4	2.4	81.9	45	2
Transfer Bay	5.6	3.3	50	45	1
Battery_Bay	3	3.3	30	45	1
Loco_Bay	3	3.3	30	45	1
Conn_HLG	3	3.3	70.5	45	1
Eng_Wshop	3	3.3	30	45	1
Conn_Wshop	3	3.3	42	45	1
Refuge Bay	3	3.3	15	45	1
Tip_Exc	3	3.3	22	45	1
L_Upass	4	diameter	27.4	35	1
L_Wpass	4	diameter	25.51	35	1

Table 53- Working Cost Development

Sub-Activity	Width(m)	Height(m)	Length(m)	Efficiency(m/Mth)	No. of Ends
Working Cost Development					
HLG	3	3.3	210	45	2
Refuge Bay	4	4	20	45	1
Timber Bay (MER)	2	3.3	20	45	2
X-Cut (to MER)	3	3.3	433.6	45	2
Slot - Box (MER)	3	1.5	6	46	1
Stub - Box (MER)	1.5	1.5	16.4	46	1
Travelling Way (MER)	2.4	2	59.95	20	2
Step Over (MER)	2.4	2.4	19.4	20	2
Raise (MER)	1.6	2.4	250	25	2
Funkhole (MER) Stub	1.5	1.5	31	18	2
Drop-R - Box (MER)	1.5	1.5	191.2	46	2

Table 54: Off Reef Station and Project Development parameters

	Width (m)	Height (m)	Length (m)	Efficiency (m/month)
Stations				
Personnel Ramp	4.7	4.7	153.1	138
Connecting X-Cut	4.7	4.7	75.1	138
Personnel Holing	4.7	4.7	130.1	138
FW Assembly Bay	8	5	5	45
Workshop East	4.7	4.7	51.7	138
Workshop	4.7	4.7	80.2	138
594 Refuge Bay	4	4	20	45
S# X-Cut S	4.7	4.7	60	138
M# C-Cut S	4.7	4.7	53.3	138
Secondary Haulage 1	4.7	4.7	221.3	138
Access X-Cut 2B	4.7	4.7	84	138
Conveyor Hlg Free	4.7	4.7	240.8	138
MCC 3	4.7	4.7	49.5	138
Settler Feed Haulage	4.7	4.7	168.5	138
Main Artery Connect	4.7	4.7	74.9	138
594 Refuge Bay	4	4	20	45
Settler Haulage BA M	4.7	4.7	264.8	138
Settler Haulage BA N	4.7	4.7	143.2	138
Settler Haulage BA S	4.7	4.7	277.3	138
Sub Station X-Cut BA	5.6	3.5	6	45
Spill Load Ramp BA	5.6	3.5	24	45
Settler Haulage	5.6	3.5	6	45
Silo Bottom 3	4.7	4.7	114.5	138
Boxhole Pilot	4	diameter	496.1	45
Shaft Bottom Ramp	4.7	4.7	422.2	138
Shaft Bottom Ramp Cubby	4	3.5	24	45

Table 55: Shaft Sinking Advance rates (m/month)

	Depth (m)	Study Advance Rate (m/month)	Base Case Advance Rate (m/month)
Main shaft	1 191	78	45
Pre-sink Main Shaft	94	24	24
Service Shaft	1 154	78	34
Pre-sink Service Shaft	94	24	24

Notes (1): Actual sinking rates from no.1 Shaft

Table 56: Development Advance Rates (m/month)

Activity	Study Advance Rate (m/month)	Base Case Advance Rate (m/month)
Level Connector	450	45
Connecting Haulage	60	45
Haulages	60	45
Material, Chairlift and Belt Decline	60	45
X-Cut (to UG2 and Merensky)	60	45
Connecting Decline	60	45

Table 57: Merensky Reef Geozone

Width	**Unit**	
Hangingwall Overbreak	cm	5
Resource Width (weighted aver.)	cm	122
Footwall Overbreak	cm	10
Density	**Unit**	
Hangingwall Density	t/m^3	3.38
Resource Density	t/m^3	3.37
Available to mine Resource Density	t/m^3	3.37
Footwall Density	t/m^3	3.28
Grade	**Unit**	
Hangingwall Grade	g/t	0.33
Resource Grade (4E)	g/t	7.24
Available to mine Resource grade (4E)	g/t	7.24
Footwall Grade	g/t	0.39
Prill Split	**Unit**	

Platinum	% of 4E Grade	64.9
Palladium	% of 4E Grade	26.6
Rhodium	% of 4E Grade	4.3
Gold	% of 4E Grade	4.2
Nickel	% of tons	0.24
Copper	% of tons	0.11

Table 58: Production Parameters

ROW	DESCRIPTION	UNIT	MERENSKY	CALCULATION
a	Drill hole length	m	1.1	Standard drill steel
b	Blasting efficiency	%	71	
c	Effective advance	m	0.78	a x b
d	Effective panel length	m	29.9	Includes "in stope" pillars
e	Panels per crew	Number	2	
f	Shifts	Number	16	23 – 7 lost blasts
g	Panel advance	m/month	6.2	c x f / e
h	Area per panel	m²	187	d x g
i	Area per crew	m²	373	e x h
j	Crews per half level	m	8	
k	Ledging area	Number	260	
l	Area per half level	m²	3244	(i x j) + k

Table 59: Footwall Developed Scenarios Modifying Factors (Merensky Reef)

Name	Modification Type	Modification Factor
Geological	Loss	25.36%
Mining	Loss	2.00%
Pillar	Loss	11.20%
Other	Loss	6.00%
Mining into potholes/over stope	Gain	2.00%

Table 60: Footwall Developed Scenarios Mine Structure (Merensky Reef)

Scenario	Avg. Back length (m)	X-Cut spacing	No of levels	No of half levels	No of blocks per half level
All	250	218	15	30	10

Table 61: Footwall Developed Scenarios Reserves (Merensky Reef)

Scenario	Resource width (m)	Reserve (Mt)	Grade (g/t 4E)	Content (Moz 4E)
Base Case	1.22	51.39	5.46	280.67
C02	1.22	51.39	5.46	280.67
C03	1.22	51.39	5.46	280.67
C04	1.22	51.39	5.46	280.67

Table 62: Shaft Sinking Advance rates (m/month)

	Depth (m)	Diameter (m)	Efficiency (m/month)	Equipping (Months)
Pre-sink Main shaft	60	10.5	24	n/a
Main shaft	1 225	10.5	240	8
Pre-sink service shaft	60	8.7	24	n/a
Service shaft	1 187	8.7	240	3

Table 63: On Reef Developed Scenarios Modifying Factors (Merensky Reef)

Name	Modification Type	M01 – M03	M04	M05	M06
Raise Pillars	Loss	6.12%	Layout does not incorporate raise pillars (1)		
Geological	Loss	25.36%	25.36%	25.36%	25.36%
In stope pillars	Loss	5.06%	5.93%	5.93%	8.47%
Over Stope	Gain	2%	2%	2%	2%

Note (1) As advised by Rock Engineering

Table 64: On Reef Developed Scenarios Mine Structure

Scenario	Avg. Back length (m)	Avg. Block length (m)	Avg. Block width (m)	No of levels	No of half levels	No of blocks per half level
M01a	315	245	200	12	23	11
M01b	315	245	200	12	23	11
M03	315	245	200	12	23	11
M04	310	240	200	12	23	11
M05	310	240	200	12	23	11
M06	238	168	200	15	29	11

Table 65: On Reef Developed Scenarios Reserves (Merensky Reef)

Scenario	Area (Mm^2)	Reserve (Mt)	Grade (g/t 4E)	Content (Moz 4E)
M01a	8.97	52.6	4.14	217.65
M01b	8.97	52.6	4.14	217.65
M03	8.68	50.6	4.21	213.0
M04	9.37	50.55	4.58	252.89
M05	8.78	43.89	5.12	244.22
M06	8.25	38.43	5.41	226.70

Table 66: Shaft Sinking Advance rates (m/month)

	Depth (m)	Diameter (m)	Efficiency (m/month)	Equipping (Months)
Pre-sink Main shaft	60	10.5	24	n/a
Main shaft	1 225	10.5	240	8
Pre-sink service shaft	60	8.7	24	n/a
Service shaft	1 187	8.7	240	3

Table 67: Working Cost Development parameters

Sub Activity	Width (m)	Height (m)	Total Length (m)	Advance (m/month)
HLG	4.5	3	57 390	235
Refuge Bay	4	4	6 000	45
X-Cut	4.5	3	130 080	235
Slot - Box 3 (MER)	3	1.5	1 800	46
Stub - Box 3 (MER)	1.5	1.5	4 920	46
Slot - Box 4 (MER)	3	1.5	1 800	46
Stub - Box 4 (MER)	1.5	1.5	4 920	46
Timber Bay	2	3.3	4 800	45
Travelling Way (MER)	2.4	2	17 985	20
Step Over (MER)	2.4	2.4	5 820	20
Raise (MER)	1.6	2.4	72 300	25
Funkhole (MER) Stub	1.5	1.5	9 300	18
Drop-R - Box 4 (MER)	4.5	3	32 070	235
Drop-R - Box 3 (MER)	4.5	3	57 360	235

Table 68: Decline parameters

	Width (m)	Height (m)	Total Length (m)	Efficiency (m/month)
M_Dcl	6m diameter		3 787	270
Acc_LvL	6m diameter		1 709	270
L_Mpass	4m diameter		410	35
HLG	6m diameter		4 770	270
C_Dcl	6m diameter		3 722	270
B_Dcl	6m diameter		4 717	270
Conn_Dcl	6m diameter		2 011	270
T_way	2.4	2.4	655	45
Transfer Bay	5.6	3.3	750	45
Battery_Bay	3	3.3	450	45
Loco_Bay	3	3.3	450	45
Conn_HLG	3	3.3	1 058	45
Eng_Wshop	3	3.3	450	45
Conn_Wshop	3	3.3	630	45
Refuge Bay	3	3.3	225	45
Tip_Exc	3	3.3	351	45
L_Upass	4m diameter		411	35
L_Wpass	4m diameter		383	35

10.1 Calendar

The mining calendar is derived from an eleven-shift fortnight with 3 production shifts per day. There are also 3 shifts operating in the logistics, shaft infrastructure, pumping and similar areas. Blasting takes place once a day – at end of the day shift for the conventional mining options. The number of working days per year takes cognisance of the operating schedule, official public holidays and weekends.

Table 69: Working Days

Description	Days
Days per annum	365
Official public holidays	12
Saturdays	26
Sundays	52
Total working days	275
No. of months	12
Working days per month	23

Figure 40: Shaft Layout

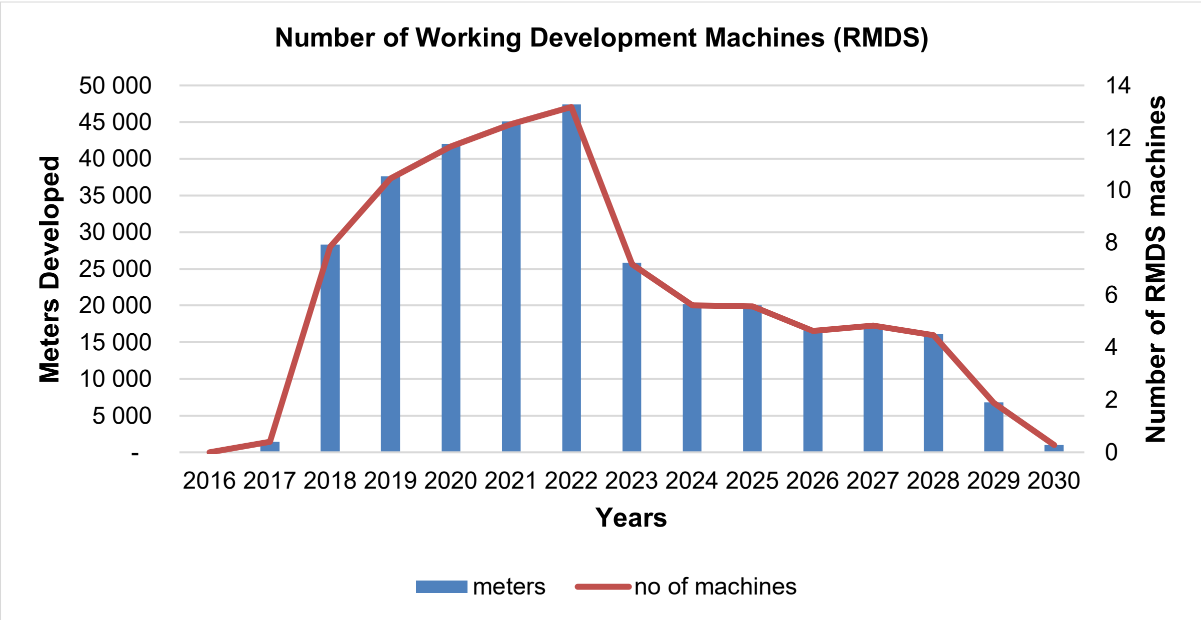

Figure 41: Number of Working Cost Development Machines

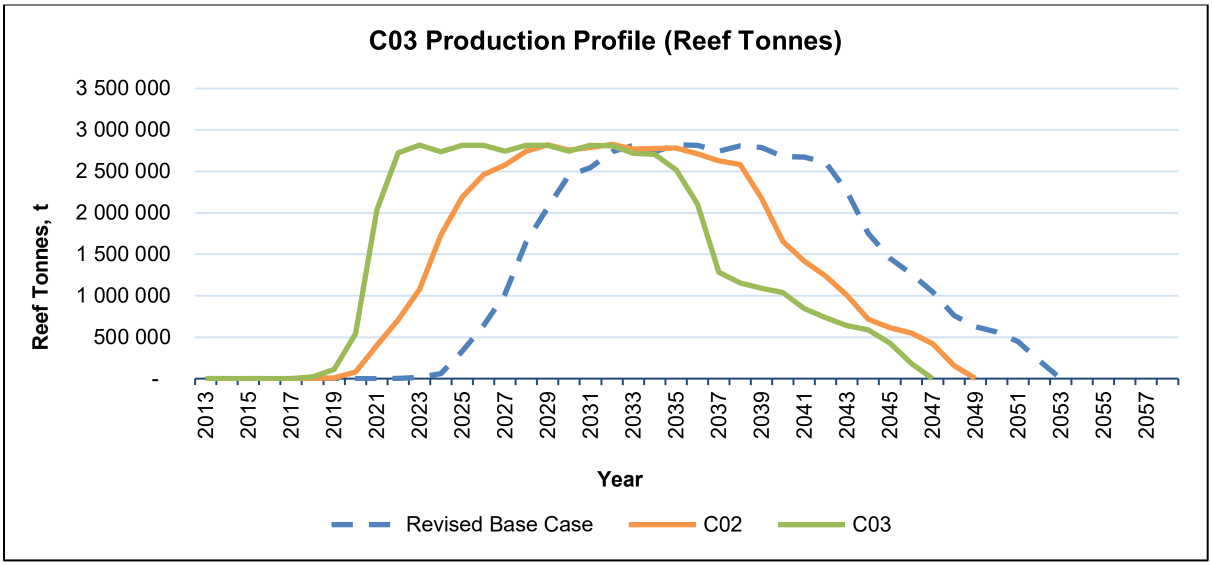

Figure 42: C03 production profile

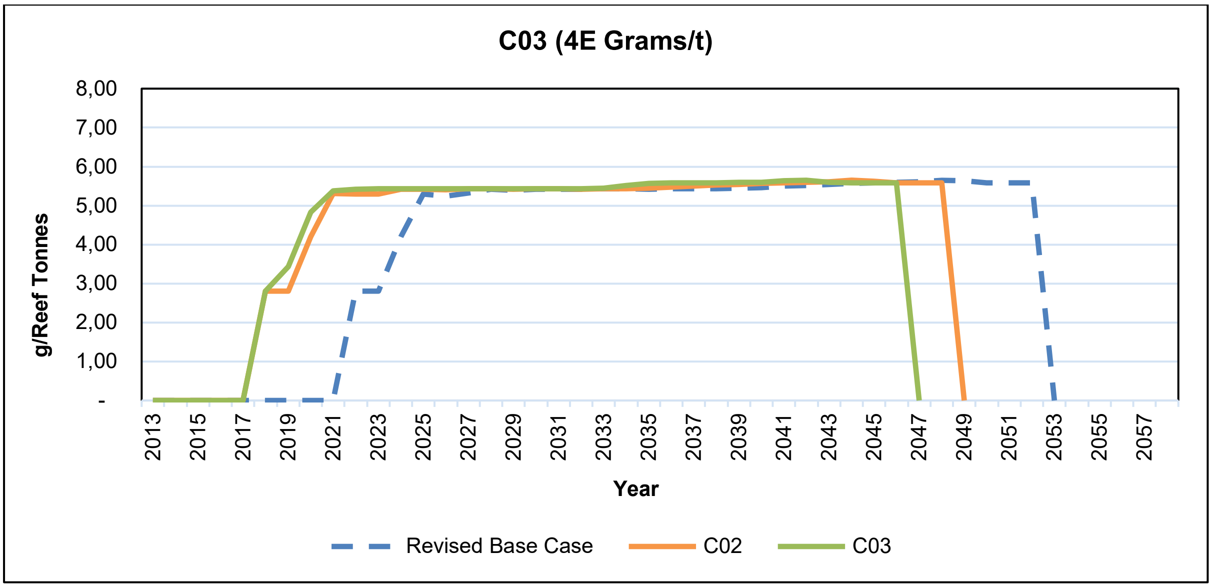

Figure 43: C03 4E grade profile

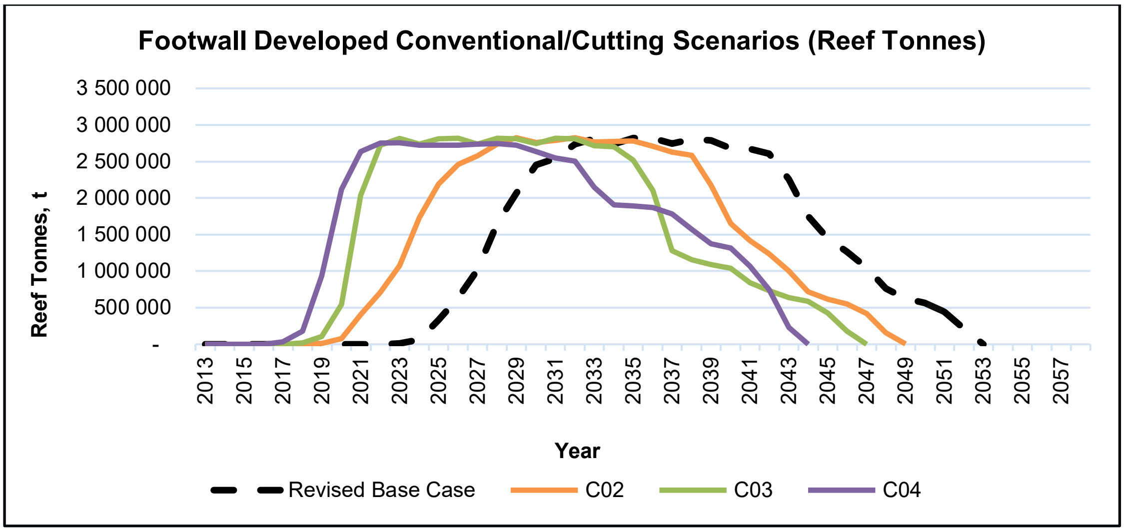

Figure 44: C04 production profile

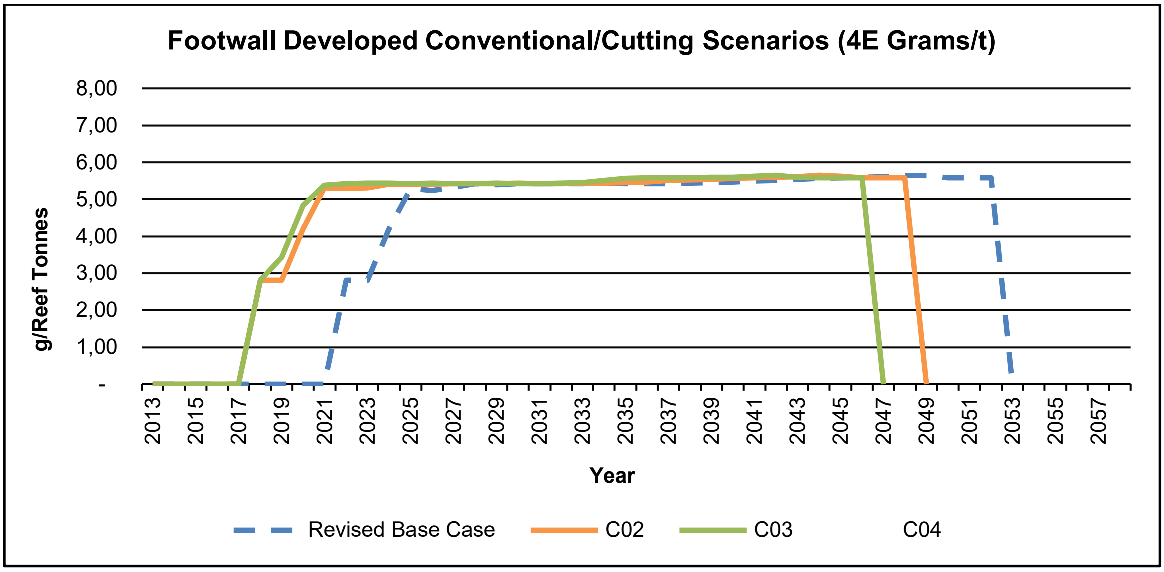

Figure 45: C04 4E grade profile

Table 70: Shaft Sinking Advance rates (m/month)

	Depth (m)	Diameter (m)	Efficiency (m/month)	Equipping (Months)
Pre-sink Main shaft	60	10.5	24	n/a
Main shaft	1 225	10.5	240	8
Pre-sink service shaft	60	8.7	24	n/a
Service shaft	1 187	8.7	240	3

Table 71: Stations parameters

	Width (m)	Height (m)	Total Length (m)	Efficiency (m/month)
Main Shaft Stn 1	7.6	5.5	45	45
Conveyor Belt X/Cut	4.7 or 4.5	4.7 or 3	166	138 or 235
Silo Dome	12	4	36	45
Conn to Conv Belt	4.7 or 4.5	4.7 or 3	71	138 or 235
Conveyor Belt Access	4.7 or 4.5	4.7 or 3	237	138 or 235
UG2 Transfer Pass	6	6	16	18
Mer Transfer Pass	6	6	54	18
Main Shaft Stn X/Cut	4.7 or 4.5	4.7 or 3	324	138 or 235
Workshop Complex	4.7 or 4.5	4.7 or 3	585	138 or 235
Material Loop In	4.7 or 4.5	4.7 or 3	898	138 or 235
Material Conn	4.7 or 4.5	4.7 or 3	499	138 or 235
Transfer Silo	5.6	3.5	24	18
Material Loop Out	4.7 or 4.5	4.7 or 3	504	138 or 235
Refuge Bay #	4	4	20	45
Sub Station	6	5	77	45
Access to top of Settlers	4.7 or 4.5	4.7 or 3	91	138 or 235
Dirty Water X/Cut	4.7 or 4.5	4.7 or 3	137	138 or 235
Settler Dome	18	3.7	36	45
Drain Hole	0.5	0.5	71	45
Service Shaft Stn	7.6	5.5	25	45
Service Shaft Stn X/Cut	5.6	3.5	48	45
RAW #	4.7 or 4.5	4.7 or 3	167	138 or 235
Main Shaft Station HLG	7.6	5.5	30	45
Main Shaft Belt Stn HLG	7.6	5.5	15	45
Conveyor X/Cut HLG	4.7 or 4.5	4.7 or 3	81	138 or 235
Sub Stn HLG	6	5	81	45
Silo Bulkhead Chb	10	8	12	45
Main Shaft Stn X/Cut HLG	4.7 or 4.5	4.7 or 3	352	138 or 235
Pipe Suction X/Cut	4.7 or 4.5	4.7 or 3	76	138 or 235
Sump/Settler X/Cut	10	3.5	20	45
Settler	10	10	120	18

Shaft Bottom Ramp	4.7 or 4.5	4.7 or 3	401	138 or 235

*Note Areas highlighted in grey will either be mined by the MTM or RMDS

The working cost development design parameters and a breakdown of the method of mining are summarised in Table 72 and Table 73.

Table 72: Working Cost Development parameters

	Width (m)	Height (m)	Length (m)	Efficiency (m/month)
Strike Belt Drive	5.0	3 or 2.2	200	235or 262
Strike Transport Drive	4.5	3 or 2.2	200	235or 262
Strike Connector	4.5	3 or 2.2	45	235or 262
Strike Pillar (1)	20	3 or 2.2	200	n/a
Raise Blast	4.5	3 or 2.2	20	40
Raise Merensky (2)	4.5	3 or 2.2	245	235or 262
Strike ASD Drive	4.5	3 or 2.2	200	235or 262

***Note:**

(1) Strike pillar sizes are subject to rock engineering recommendations

(2) Raise will be mined by RMDS machine in scenarios M01a, M01b, M03, M04 and M06

(2) Raise will be mined conventionally in scenario M05

Table 73: Strike ASD and Raise machine types

Scenario	Strike ASD drives	Raise
M01a	10 mined by RMDS (4.5m x 3.0m)	245m long mined by RMDS (4.5m x 3.0m)
M01b	10 mined by RMDS (4.5m x 3.0m)	245m long mined by RMDS (4.5m x 3.0m)
M03	10 mined by RMDS (4.5m x 3.0m)	245m long mined by RMDS (4.5m x 3.0m)
M04	8 mined by RMDS (4.5m x 2.2m)	240m long mined by RMDS (4.5m x 2.2m)
M05	Mined conventionally with panel	240m long mined conventionally (1.6m x 2.2m)
M06	No ASD drives	168m long mined by RMDS (4.5m x 2.2m)

Table 74: Auxiliary Equipment Fleet

Development Equipment required	Total
LHDs per cutting machine	1
UVs per half level	1
Supervisor Vehicles per half level	1

Table 75: ULP m² breakdown

Description	Units	Siding	Vent Holing	Panel	Total
Effective Face Length	m	2.5	2.2	19	23.70
Stope height	m	1.2	1.2	1.2	1.2
Face advance per blast	m	2.8	1.8	1.8	2.1
System Face Advance	m	20	20	20	20
Panel face advance	m	10	10	10	10
m^2 per blast	m^2	4.2	4.0	36.0	44.2
Mining rate (m^2/month)	m^2	15.2	22.5	202.7	240.40
Total m^2	**m^2**	**182**	**270**	**2 838**	**3 290**

Table 76: MN 220 parameters

Description	Width (m)	Height (m)	Rate
Belt Drive	5	2.2	236 m/month
Transport Drive	4.5	2.2	262 m/month
Strike Connectors	4.5	2.2	262 m/month
Raise	6	2.2	196 m/month
Stope Panel	17	1.25	1 129 m^2/month
Vent holing	5	4	

Table 77 MN220 Level Development Parameters

Level Development Activity Times				
Description	**Unit**	**Min**	**Mode**	**Max**
Drilling	min/length (m)	120	140	180
Charging	min/length (m)	60	67	80
Blast	min/length (m)	-	-	-
Loading and cleaning	minute (m)	Model Dependent		
Bolting	min/length (m)	50	57	70

Table 78 - MN220 Reef Stoping Rate Inputs

MN220 & Blower Unit Input Parameters				
Description	**Unit**	**Min**	**Mode**	**Max**
QTY	ea	2	2	2
Distribution				
Cutting Speed	m/ hour	1,05	1,18	1,4 5
Cutting Speed	min/ meter	41.38	46.63	57.14
Cutting Speed	m/day	13,65	15,38	18,85
Onboard Capacity	m^3	15	15	15
	Fill Factor	90 %	90 %	90 %
	t	31,293	31,293	31,293
Hencon Buffer Capacity	kg	34,770	34,770	34,770
Weekly Service Interval Maintenance	m	55	64	83
Weekly Service Interval Maintenance Duration	min	400	427	480
Monthly Service Duration	min	520	547	600

10.1.1 Mining Sequence

a) Once the dip ledge drive, strike belt drive and transport drives have been pre-developed, a launch ramp will be blasted from the ledge drive to create space for the MN 220 machine.

b) One machine will make cuts that will serve as a turning bay into the first panel.

c) Four machines will cut 3.9m cuts on strike in the 15.6m panels.

d) The machines will operate in a staggered arrangement, the first machine being ahead and the last machine lagging behind the third.

e) Crush pillars will be left behind by cutting 5m ventilation holes.

f) Once the machines approach the end of the block, a bridge will be set up in the pre- developed raise of the next block and the machines will continue on to the next block.

g) The lagging machines will proceed in the same fashion.

h) Once the leading machine reaches the boundary of the strike, it will mine the turning bay by making adjacent cuts at the end of the strike.

i) All four machines will turn into the next panel and will cut in the opposite direction on strike as depicted in the above layout.

10.2 Scenario M01a (MTM, TBM, RMDS and Slot Borer)

Scenario M01a utilises mechanical blind shaft boring technology for sinking the shaft, MTM (4.7m by 4.7m) for the stations and project development, TBM (6m diameter) machine for the decline, RMDS (4.5m by 3m) for the working cost development and the slot boring machine in the stopes.

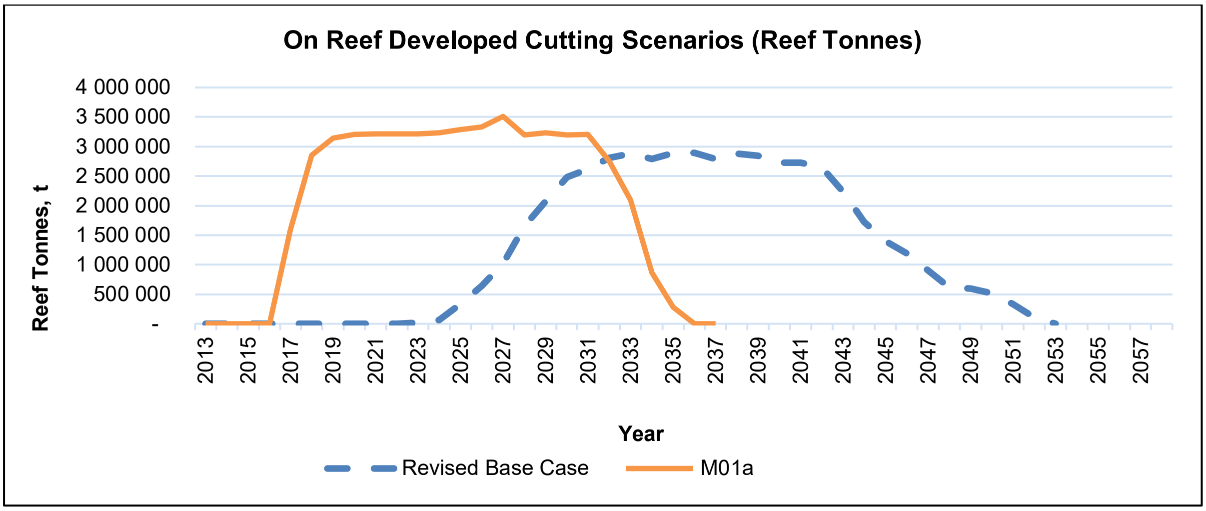

Figure 46: Base Case vs. M01a production profile

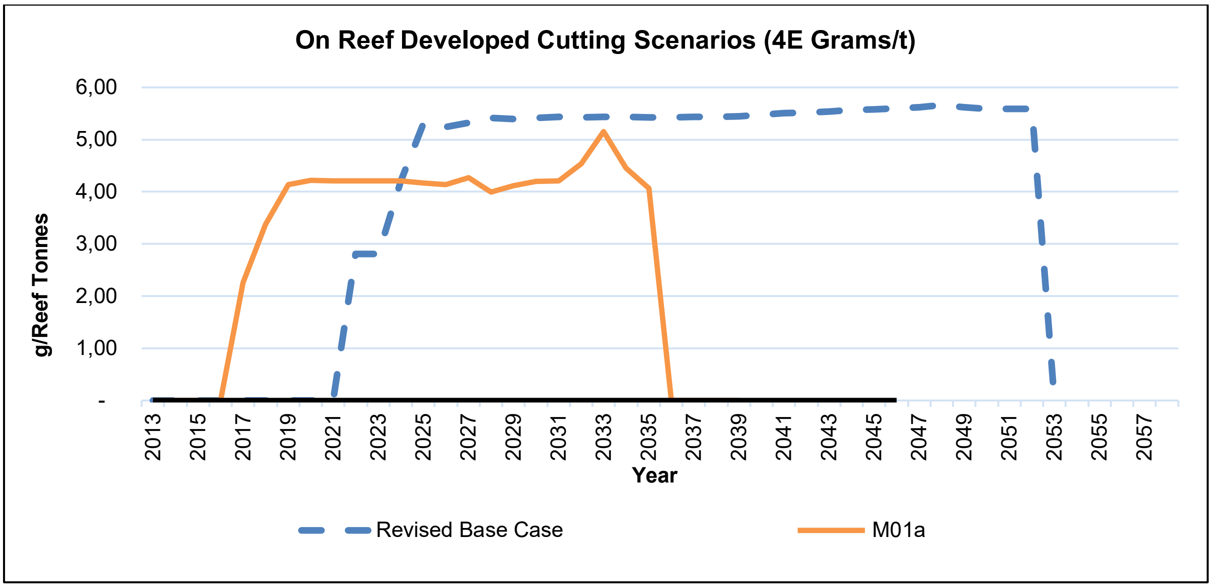

Figure 47: Base Case vs. M01a grade profile

10.3 Scenario M01b (RMDS, TBM, RMDS and Slot Borer)

Scenario M01b utilises mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the stations and project development, TBM (6m diameter) machine for the decline, RMDS (4.5m by 3m) for the working cost development and the slot boring machine in the stopes.

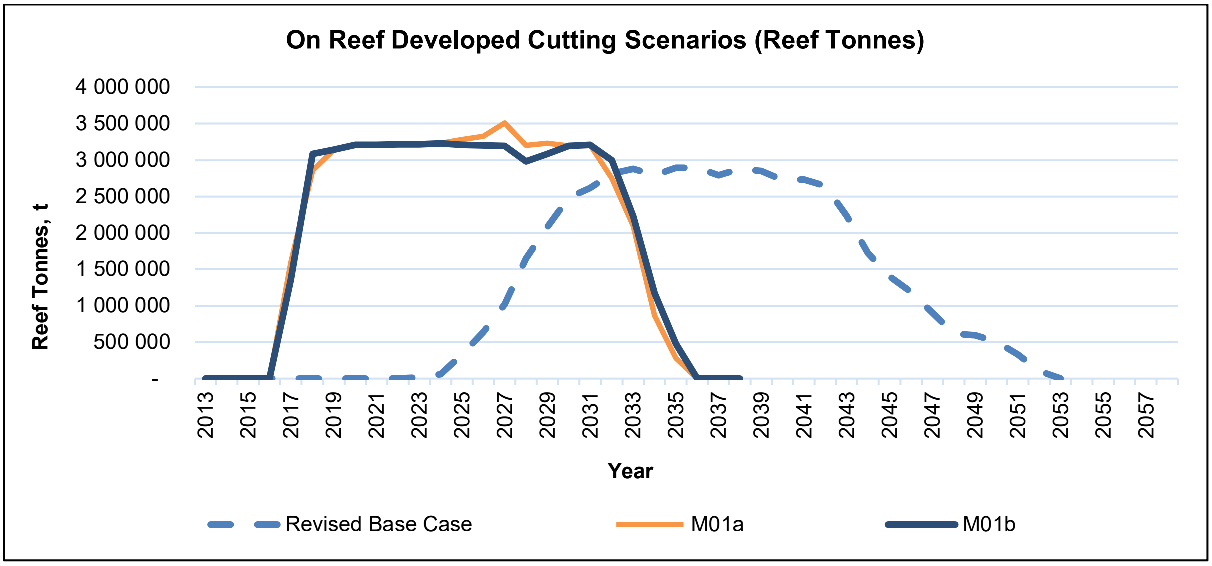

Figure 48: Base Case vs. M01b production profile

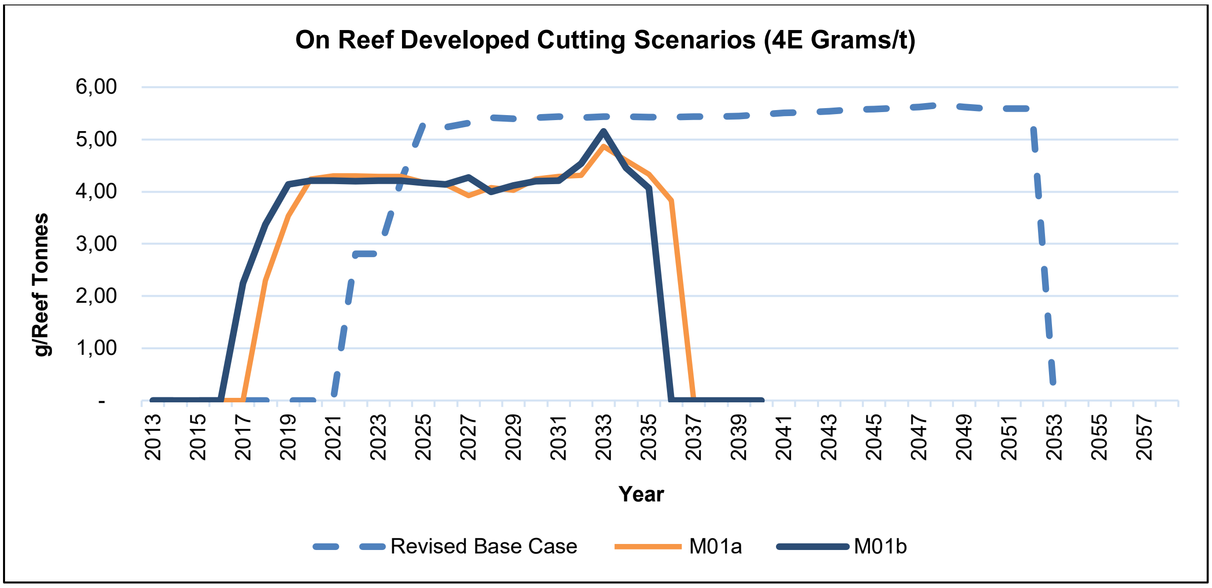

Figure 49: Base Case vs. M01b grade profile

10.4 Scenario M03 (RMDS all and Slot Borer)

Scenario M03 utilises mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by

3m) for the stations and project development, decline and working cost development and the slot boring machine in the stopes.

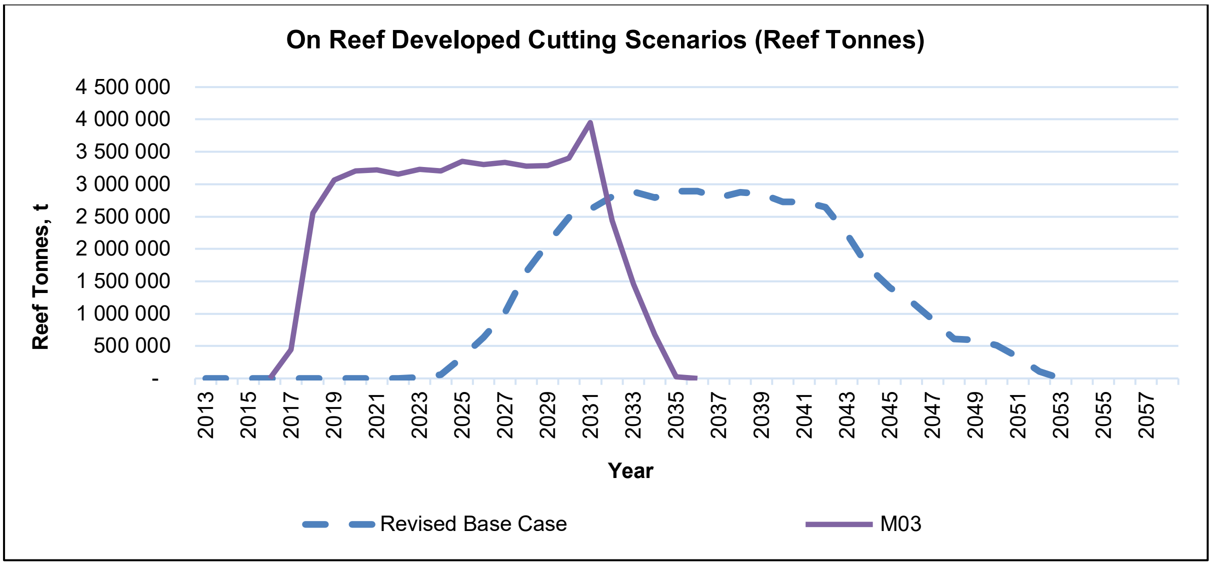

Figure 50: Base Case vs. M03 production profile

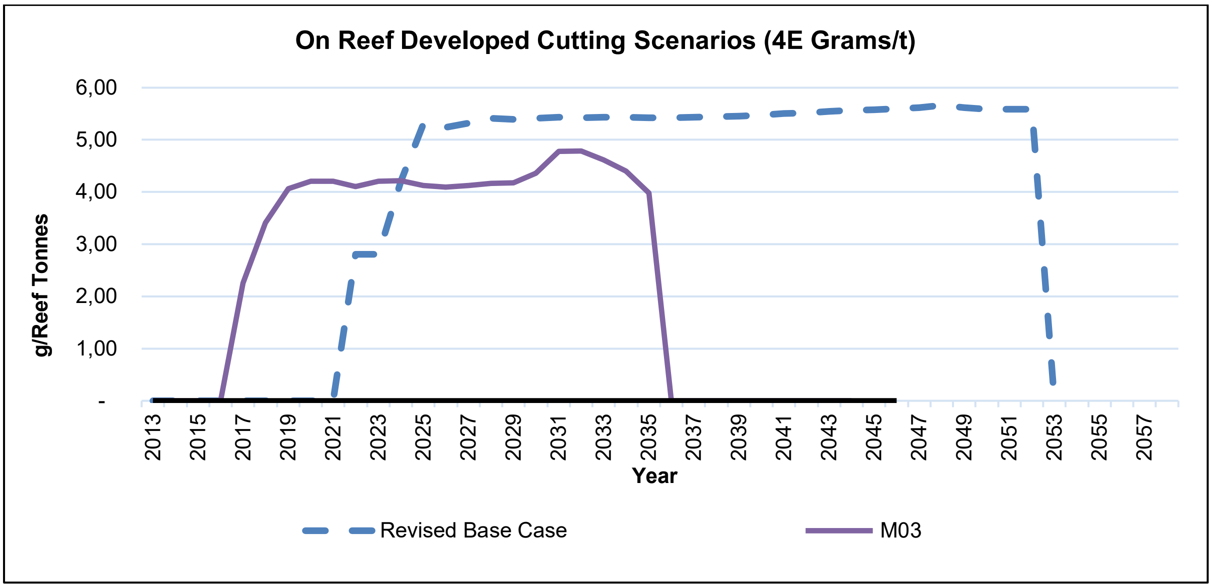

Figure 51: Base Case vs. M03 grade profile

10.5 Scenario M04 (RMDS all and ULP)

Scenario M04 utilises mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the stations and project development, decline and working cost development (4.5m by 2.2m) and the ULP fleet in the stopes.

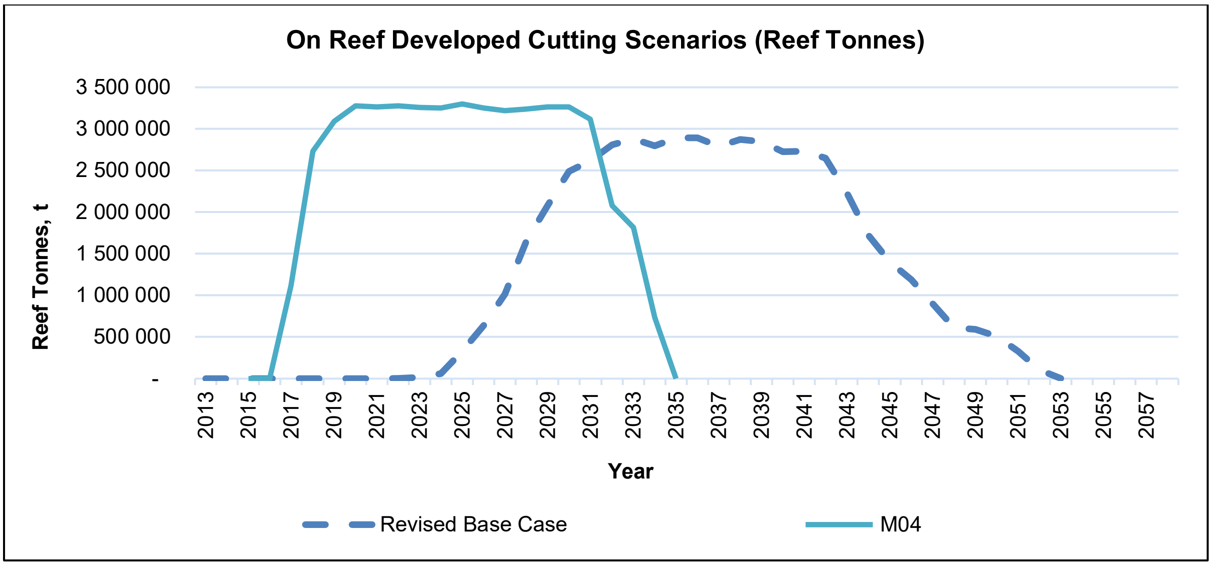

Figure 52: Base Case vs. M04 production profile

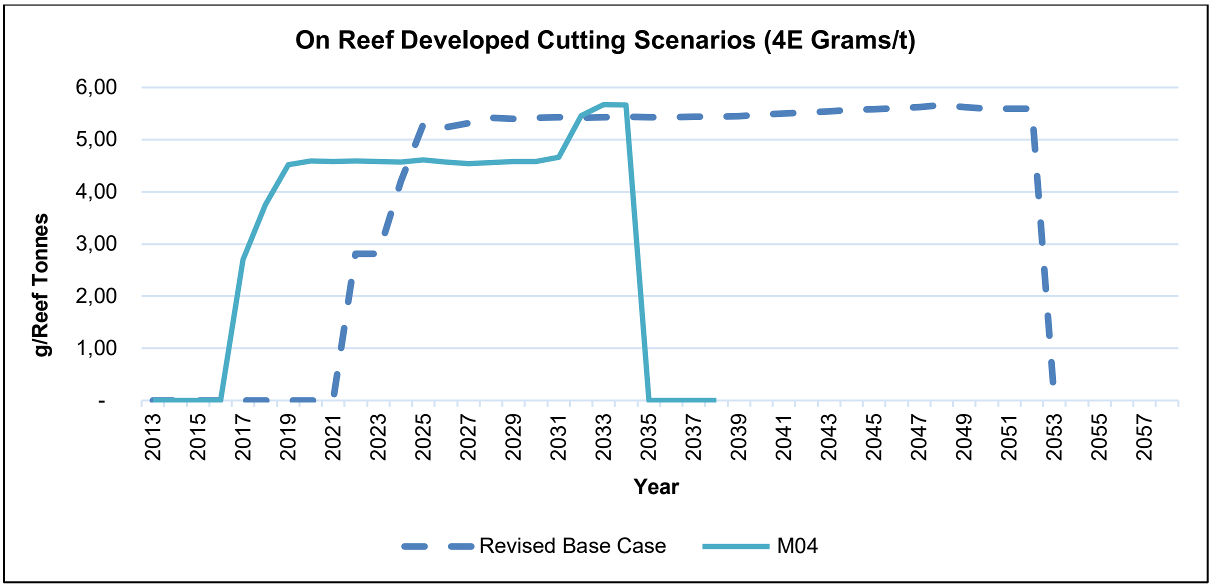

Figure 53: Base Case vs. M04 grade profile

10.6 Scenario M05 (RMDS all and Conventional drill and blast stoping)

Scenario M05 utilises mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the stations and project development, decline and working cost development (4.5m by 2,2m) and conventional drill and blast methods in the stopes.

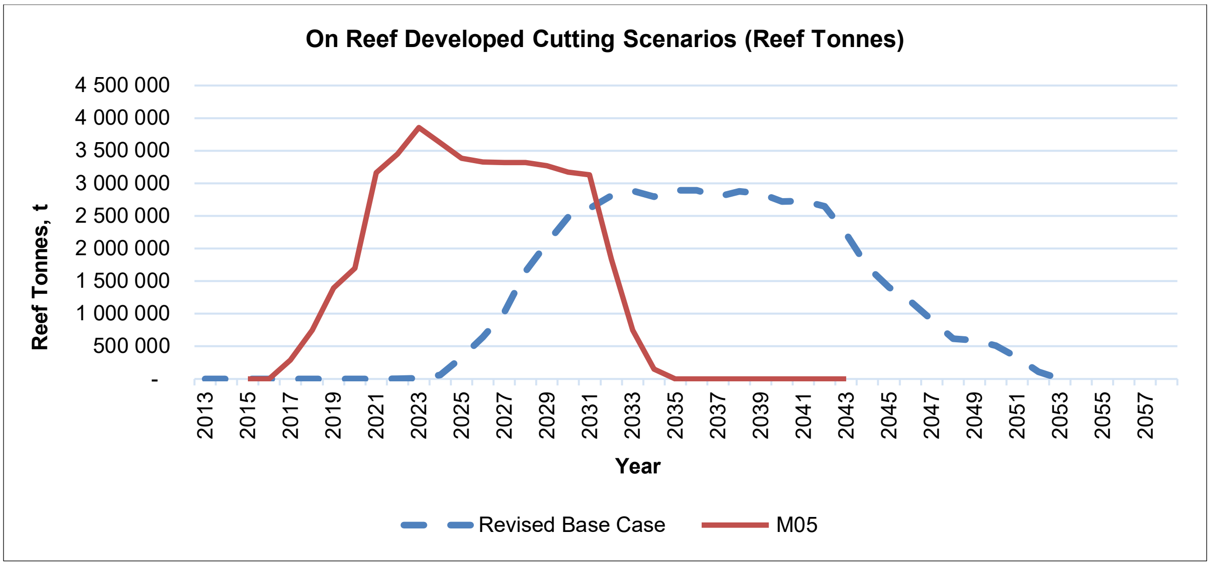

Figure 54: Base Case vs. M05 production profile

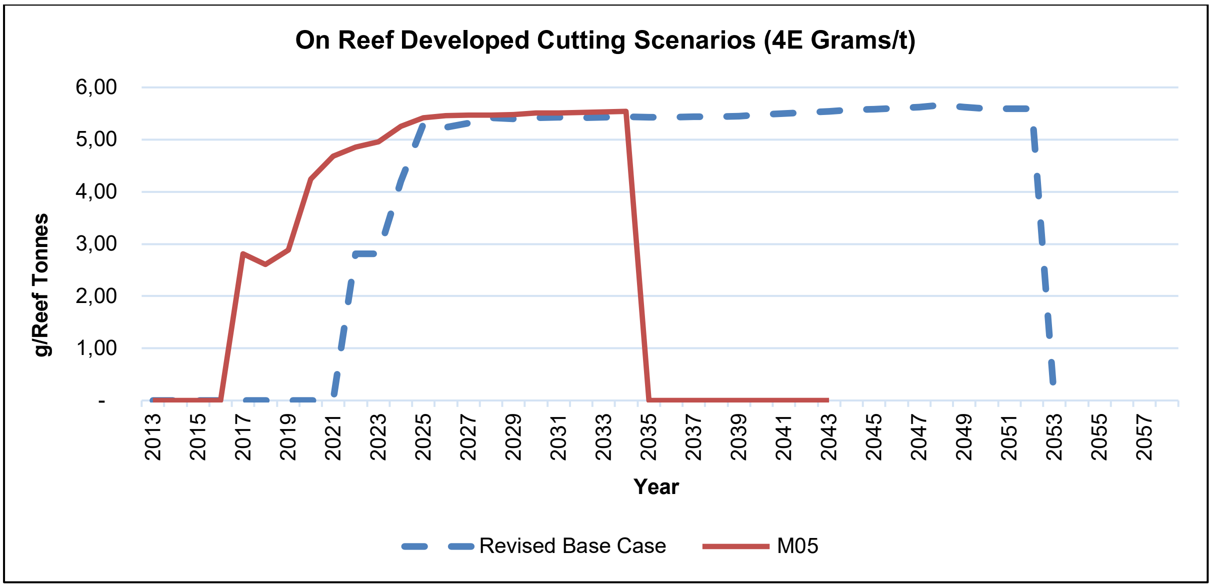

Figure 55: Base Case vs. M05 grade profile

10.7 Scenario M06 (RMDS all and Sandvik MN 220)

Scenario M06 utilises mechanical blind shaft boring technology for sinking the shaft, RMDS (4.5m by 3m) for the stations and project development, decline and working cost development (4.5m by 2,2m) and the Sandvik MN 220 machine in the stopes.

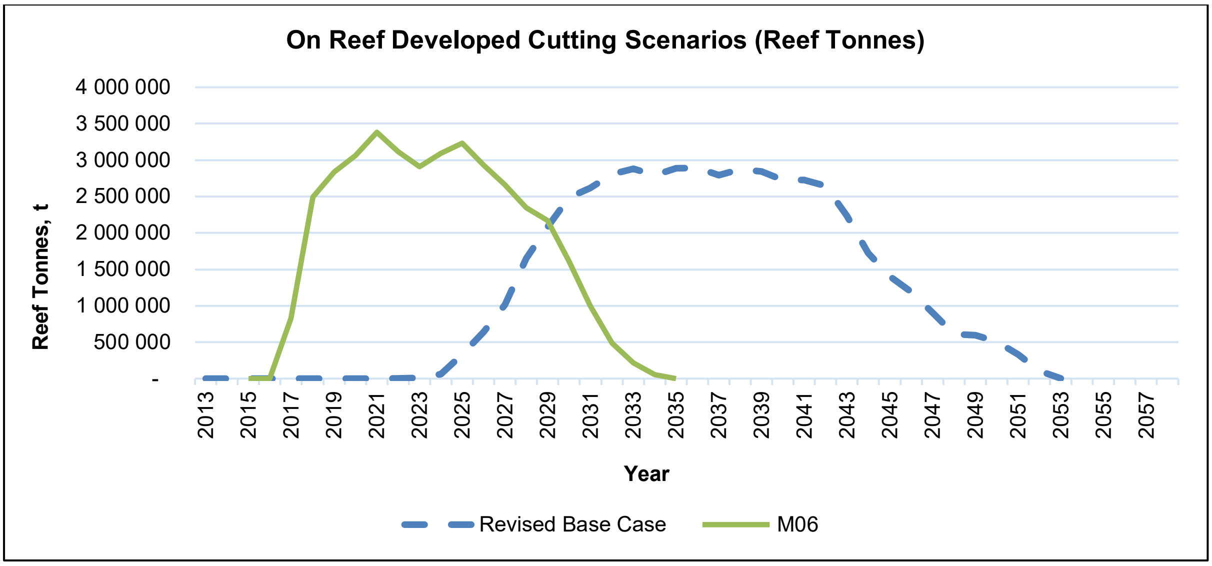

Figure 56: Base Case vs. M06 production profile

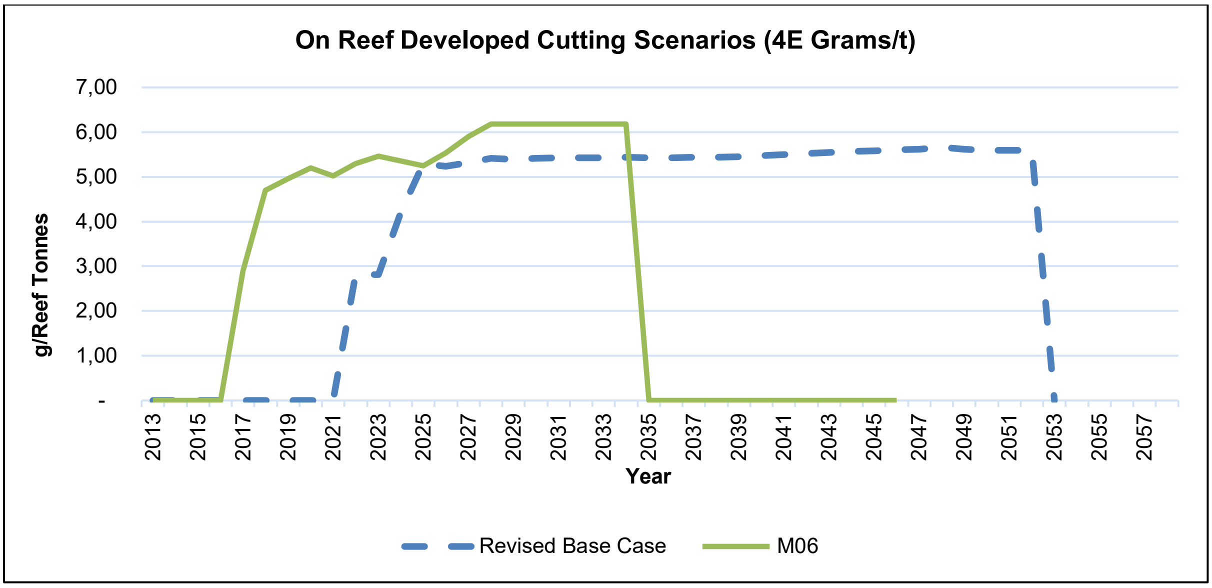

Figure 57: Base Case vs. M06 grade profile

10.8 Summary

Figure ***58*** and Figure ***59*** show the production and grade profiles of all the On Reef Developed Scenarios.

- M01b, M03, M04 and M05 have the same start dates.
- M05 has a slower ramp up – the decline and working cost development have been slowed down to accommodate the slower stoping rates.
- The spike at the end of the grade profiles of M01, M01b, M03, M04 and M06 is due to the development finishing before the stoping which has a higher grade.

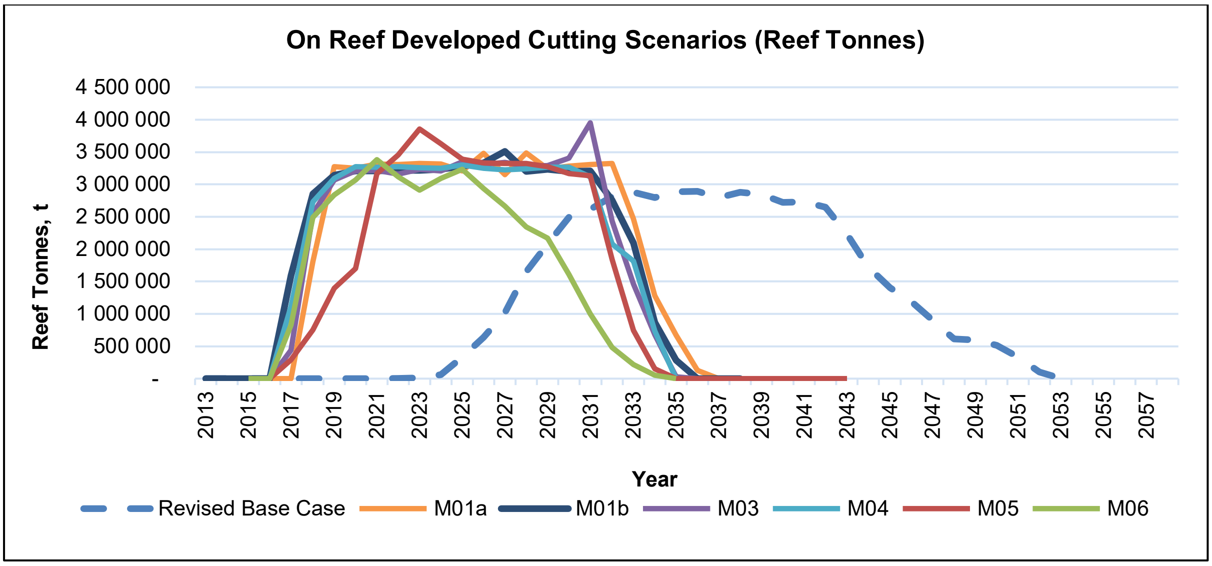

Figure 58: On Reef Developed Scenarios Production Profiles

Table 79: On Reef Developed grade breakdown

Scenario	Decline Barrel Dimensions	Decline Barrel g/t	Working Cost Dev. Dimensions	Working Cost Dev. g/t	Stoping width	Stoping g/t	Avg. grade (g/t)
M01	6m diameter	1.38	4.5m x 3m	2.25	1.25	7.07	4.14
M01b	6m diameter	1.38	4.5m x 3m	2.25	1.25	7.07	4.14
M03	4.5m x 3m	2.25	4.5m x 3m	2.25	1.25	7.07	4.21
M04	4.5m x 3m	2.25	4.5m x 2.2m	3.06	1.37	5.67	4.58
M05	4.5m x 3m	2.25	4.5m x 2.2m	3.05	1.37	5.56	5.12
M06	4.5m x 3m	2.25	4.5m x 2.2m	3.10	1.25	7.07	5.41

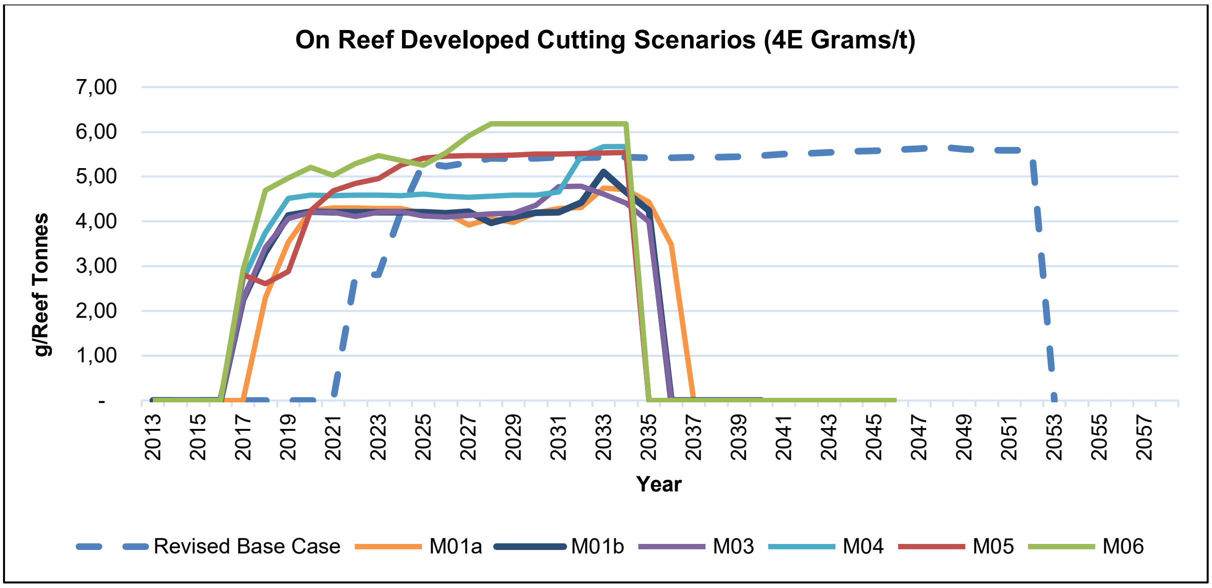

Figure 59: On Reef Developed Scenarios Grade Profile

Table 80: Conventional blind shaft sinking costs

Shaft	Sinking Rate (R/m)
Main Shaft	381 240
Service Shaft	222 724
Ventilation Shaft	32 500

Table 81: Mechanical blind shaft boring Capex

Item	Cost (Rm)	
Mechanical Shaft Sinking	**Main Shaft**	**Service Shaft**
Shaft Boring Machine	260	260
Utilities and services	15	15
Site Facilities	7	7
Shaft Pad and collar	5	5
Boring Construction Cost	305	305
Total Shaft Sinking	**592**	**592**

Table 82: Footwall developed scenarios shaft sinking costs

		Scenario			
SHAFT SINKING	**Unit**	**Revised Base Case**	**C02**	**C03**	**C04**
Main Shaft Pre-sink	m	94	60	60	60
Main Shaft	m	1 191	1 225	1 225	1 225
Service Shaft Pre-sink	m	94	60	60	60
Service Shaft	m	1 154	1 188	1 188	1 188
Vent Shaft (3 ends)	m	3 664	3 664	3 664	3 664
Total meters	**m**	**6 197**	**6 197**	**6 197**	**6 197**
Costs					
Main Shaft Pre-sink	Rm	36	23	23	23
Main Shaft	Rm	454	592	592	592
Service Shaft Pre-sink	Rm	21	13	13	13
Service Shaft	Rm	257	592	592	592
Vent Shaft	Rm	119	119	119	119
Total Costs	**Rm**	**887**	**1 339**	**1 339**	**1 339**

11 *Mining of the Capital Footprint and Development*

The mining of the capital footprint and subsequently the development was costed as per the study where a rate of 45m/crew at R1 900/m^3 was used.

Table 83: Footwall developed scenarios capital footprint costs

	Scenario			
MINING (Capital Footprint)	**Revised Base Case**	**C02**	**C03**	**C04**
Meters cubed	m^3	m^3	m^3	m^3
Main Development & capitalised areas [(1)]	1 144 484	1 144 484	1 174 195	1 355 060
Unit cost (R/m^3)	1 900	1 900	1 900	1 900
Costs Rm				
Main Development & capitalised areas	2 175	2 175	1 459	1 459
Capitalised working cost of cutting machines			1 323	1 323
Total	**2 175**	**2 175**	**2 782**	**2 782**

Note (1) Conventional drill and blasting rate

11.1.1 Cutting Technology Capex

The following tables give the purchase costs of the cutting machines that will be introduced in the horizontal ends of the stations and project development, barrels and working cost development.

Table 84: Development cutting technology capital costs

Cutting Technology	**Capital Cost (Rm)**
MTM *	88
TBM *	150
RMDS (4.5m x 3.0m) *	30
LHD	4.99

*Note All machines must be supported by an ore removal system, 2 LHDs will be deployed behind each machine.

Table 85: Footwall developed scenarios development cutting technology capital costs

		C03		C04	
Machine	**Area**	**Quantity**	**Cost (Rm)**	**Quantity**	**Cost (Rm)**
MTM	Stations and project development	2	176	2	176
TBM	Decline	3	450	3	450
RMDS	Working cost development	Mined conventionally		13	390
LHD	2 per TBM, 2 per MTM, 2 per RMDS	10	49	36	180
Total		**10**	**676**	**39**	**1 196**

11.1.2 Other Capital

The structure of the other capital estimates will be as per the Study. The estimates were broken down into areas as shown in the table below.

Table 86: Footwall developed scenarios concentrator and other capital costs

		Scenario		
OTHER CAPITAL	**Revised Base Case**	**C02**	**C03**	**C04**
Costs				
Concentrator	1 555	1 555	1 555	1 555
Civils	368	368	368	368
Structural	354	354	354	354
Painting	9	9	9	9
Mechanical	2 096	2 096	2 096	2 096
Platework	35	35	35	35
Piping	238	238	238	238
Electrical	830	830	830	830
Instrumentation	204	204	204	204
Consumables	199	199	199	199
Administration	2	2	2	2
Non Process Equipment	65	65	65	65
Total Costs (R)	**5 958**	**5 958**	**5 958**	**5 958**

11.1.3 Total Capital

Table 87: Footwall developed scenarios capital summary (FY 2011 terms)

CAPITAL SUMMARY	Scenario			
Costs (Rm)	Revised Base Case	C02	C03	C04
Shaft sinking	887	1 339	1 339	1 339
Initial capital	6 577	6 577	5 862	5 862
Cutting technology (purchase cost)			676	1 196
Cutting technology (operating costs) – Stations & decline			1 381	1 381
Concentrator	1 555	1 555	1 555	1 555
Total costs (Excl. replacement & escalation)	**9 019**	**9 472**	**10 813**	**11 333**
Cost/m^3	7 881	8 276	13 989	11 803
Cost/m sink (R/m)	356 096	373 958	462 490	495 597

The cumulative capital of the Footwall Developed Scenarios is shown in table 79. Scenario C04 has the highest capital estimate due to the introduction of shaft boring technology and cutting technology in the stations, decline barrel and working cost development.

12 On Reef Developed Scenarios

12.1 Shaft Sinking

The conventional mining of the pre-sink main, pre-sink service and ventilation shafts were costed as per the Study and are detailed in Table 88 while the mechanical blind shaft boring costs are summarised in Table 89.

Table 88: Conventional shaft sinking costs

Shaft	Sinking Rate (R/m)
Pre – sink Main Shaft	381 240
Pre – sink Service Shaft	222 724
Ventilation Shaft	32 500

Table 89: Mechanical blind shaft boring Capex

Item	**Cost (Rm)**	
Mechanical Shaft Sinking	**Main Shaft**	**Service Shaft**
Shaft Boring Machine	260	260
Utilities and services	15	15
Site Facilities	7	7
Shaft Pad and collar	5	5
Boring Construction Cost	305	305
Total Shaft Sinking	**592**	**592**

Table 90: On Reef Developed Scenarios Shaft sinking capital estimate

SHAFT SINKING	**Unit**	**M01 – M06**
Meters		
Main Shaft Pre-sink	m	60
Main Shaft	m	1 225
Service Shaft Pre-sink	m	60
Service Shaft	m	1 188
Vent Shaft	m	3 664
Total	**m**	**6 197**
Costs		
Main Shaft Pre-sink	Rm	23
Main Shaft	Rm	592
Service Shaft Pre-sink	Rm	13
Service Shaft	Rm	592
Vent Shaft	Rm	119
Total Costs	**Rm**	**1 339**

12.2 Cutting Technology Capex

Table 91: Cutting technology, accessories and ULP fleet capital estimates

Machine	Unit	Cost
MTM	Rm	88.0
RMDS	Rm	30.0
TBM	Rm	150.0
SUV	Rm	1.1
UV	Rm	1.1
LHD	Rm	4.9
Tips	Rm	1.2
Conveyor belt	R/m	20 000
Slot Boring Machine	Rm	30
Sandvik MN 220 (includes vacuum and mobile crusher)	Rm	25.7
ULP Fleet		
Dozer (1.5 per team)	Rm	3
Face Drill Rig (1.5 per team)	Rm	4.5
Roof bolter (1.5 per team)	Rm	4.5
LHD (1.5 per team)	Rm	7.5
UV (1.5 per team)	Rm	1.7
Fast Charger (0.25 per team)	Rm	2
Sweeper (0.25 per team)	Rm	2
Total per ULP fleet	**Rm**	**22.1**

Either two RMDS or MTM machines (depending on scenario) will develop the stations. Each machine will be supported by two 8t LHDs. Table ***92*** gives a summary of the capital estimates at the stations.

Table 92: Stations and project development capital estimates

Machine	Unit	M01a	M01b	M03	M04	M05	M06
MTM	Rm	176	0	0	0	0	0
RMDS	Rm	0	60	60	60	60	60
LHD	Rm	19.72	19.72	19.72	19.72	19.72	19.72
Total	**Rm**	**195.72**	**79.72**	**79.72**	**79.72**	**79.72**	**79.72**

Table 93: Decline barrel machine quantities (maximum)

	Unit	M01a	M01b	M03	M04	M05	M06
MTM	no	0	0	0	0	0	0
RMDS	no	0	0	3	3	3	3
TBM	no	3	3	0	0	0	0
LHD	no	6	6	6	6	6	6
SUV	no	9	9	9	9	22	23
UV	no	9	9	9	9	22	23
Conveyor belt	m	3 774	3 774	3 774	3 774	3 774	3 774

Table 94: Working Cost Development machine quantities at steady state

	Unit	M01a	M01b	M03	M04	M05	M06
MTM	no						
RMDS	no	14	14	14	12	10	5
LHD	no	28	28	28	24	20	10
Active half levels (SS)	no	9	9	9	9	22	23
Tips	90	90	90	90	90	220	230
Conveyor belt	m per shaft	50 400	50 400	50 400	50 400	50 400	50 400

Table 95: Number of stoping machines and teams at steady state

	Unit	M01a	M01b	M03	M04	M05	M06
Slot Boring Machine	no of teams	27	25	25			
ULP Fleet (5.5 machines per team)	no of teams				11		
Conventional: Stoping	no of teams					140	
Conventional: Ledging	no of teams					22	
MN 220	no of teams						24

Table 96: On Reef developed scenarios concentrator and other capital costs

OTHER CAPITAL (Rm)	M01 to M06	Comment
Concentrator	1 555	100% of Revised Base Case
Civils	328	89% of Revised Base Case
Structural	188	53% of Revised Base Case
Painting	8	91% of Revised Base Case
Mechanical	1 094	52% of Revised Base Case
Platework	32	91% of Revised Base Case
Piping	93	39% of Revised Base Case
Electrical	1 233	148% of Revised Base Case
Instrumentation	213	105% of Revised Base Case
Consumables	216	110% of Revised Base Case
Administration	2	100% of Revised Base Case
Non Process Equipment	64	98% of Revised Base Case
Total Costs (R)	**5 026**	

*The capital estimates for M01a to M06 have been adjusted to accommodate the cutting technology.

Table 97: On Reef developed scenarios total capital comparison

	Unit	M01a	M01b	M03	M04	M05	M06
Concentrator and other capital	Rm	5 026	5 026	5 026	5 026	5 026	5 026
Shaft Sinking	Rm	1 339	1 339	1 339	1 339	1 339	1 339
Station	Rm	196	78	78	78	78	78
Barrel	Rm	566	566	219	207	220	225
Working cost development	Rm	1 758	1 718	2 130	1 609	1 741	1 761
Stoping	Rm	810	780	1 200	186		643
Capital foot print		2 267	2 026	1 594	1 365	1 784	1 698
Total	**Rm**	**11 962**	**11 533**	**11 586**	**9 810**	**10 188**	**10 770**

The cumulative capital of the On Reef Developed scenarios is shown in table 89. Scenarios M01, M01b, M03 and M06 have the highest capital due to the use of the capital intensive TBM machine in the decline barrel (M01 and M01b), the slot boring machine and the MN 220 machine. All the On Reef Developed scenarios have a higher total capital estimate than the Revised Base Case.

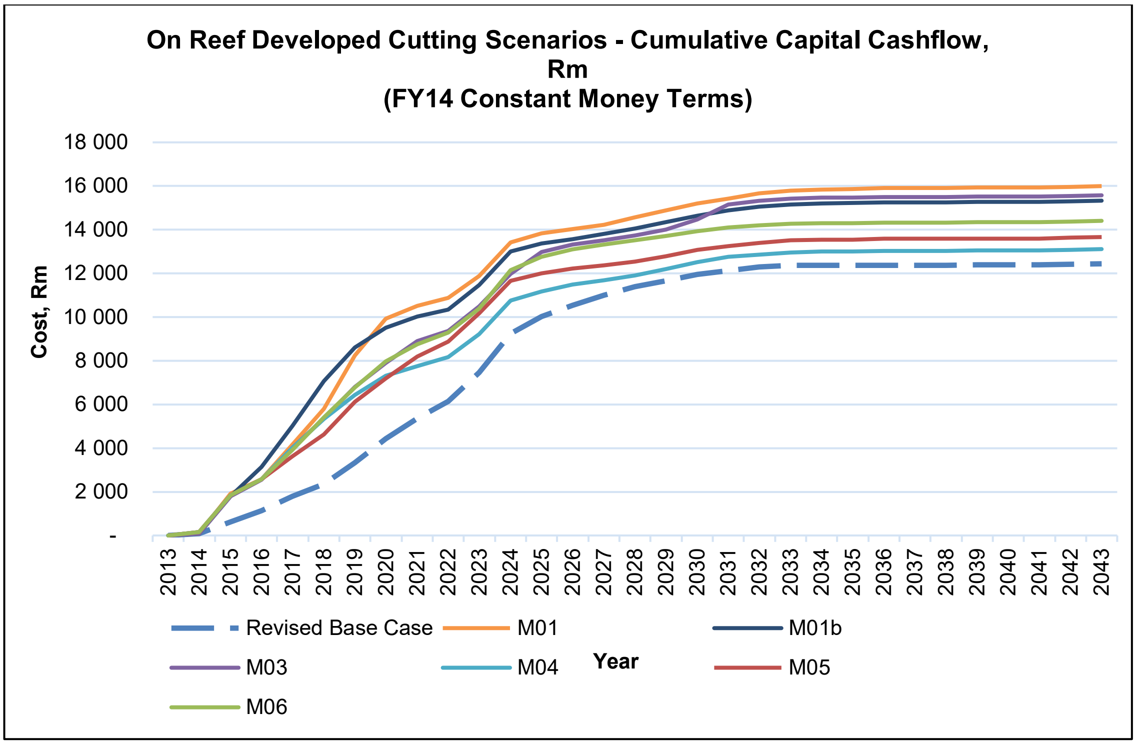

Figure 60: On Reef Developed Scenarios Cumulative capital comparison

Table 98: Footwall developed scenarios operating costs

	Unit	Cost
Development Stores		
Flat Development Materials	R/m	3 076
Travelling Way Materials	R/m	1 278
Boxhole Materials	R/m	828
Raise and Redevelopment Materials	R/m	1 652
Reclamation and Equipping materials	R/month	7 285
Maintenance Allowance	R/t	98
Utilities	R/t	48
Water	R/month	8 000
Stoping stores	R/m^2	251
Concentrator	R/t	115.08

Table 99: MTM operating costs

MTM	Unit	Costs
Roof support, consumables, energy & labour	$/m	2 000
Roof support, consumables, energy & labour	R/m	20 960

Table 100: TBM operating costs

TBM	Unit	Costs
Spares and maintenance	R/m	14 127
Power	R/m	687.7

Table 101: RMDS operating costs

RMDS	Unit	Costs
Cutter cost	R/t	90
Spares and Maintenance (MARC)	R/t	56
Electricity	R/t	14.3
Utilities	R/t	8.8
Stores cost	R/t	47
Total	**R/t**	**216.1**

Table 102: LHD operating costs

LHD	Unit	Value
Uptime hours	hrs.	17.5
Days per annum	days	23
Hours per annum	hrs.	4 830
Rate	R/hr	618.18
Total	**R/yr**	**2 985 809**

Table 103: Footwall Developed Scenarios Total Costs breakdown

	Revised Base Case	C02	C03	C04
Shaft Head Cost	32 767	32 773	29 803	24 068
Concentrator Cost	8 202	8 202	8 202	8 855
Indirect Cost	3 166	3 166	2 869	2 671
GCC	6 620	6 621	6 365	5 573
Total Cost	**50 754**	**50 761**	**47 238**	**41 166**

Table 104: Footwall Developed Scenarios Total Costs comparison

Scenario	Total Costs (Rm)	Reef Tonnes (Mt)	R/t
Revised Base Case	50 754	51.39	988
C02	50 761	51.39	988
C03	47 238	51.39	919
C04	41 166	51.39	801

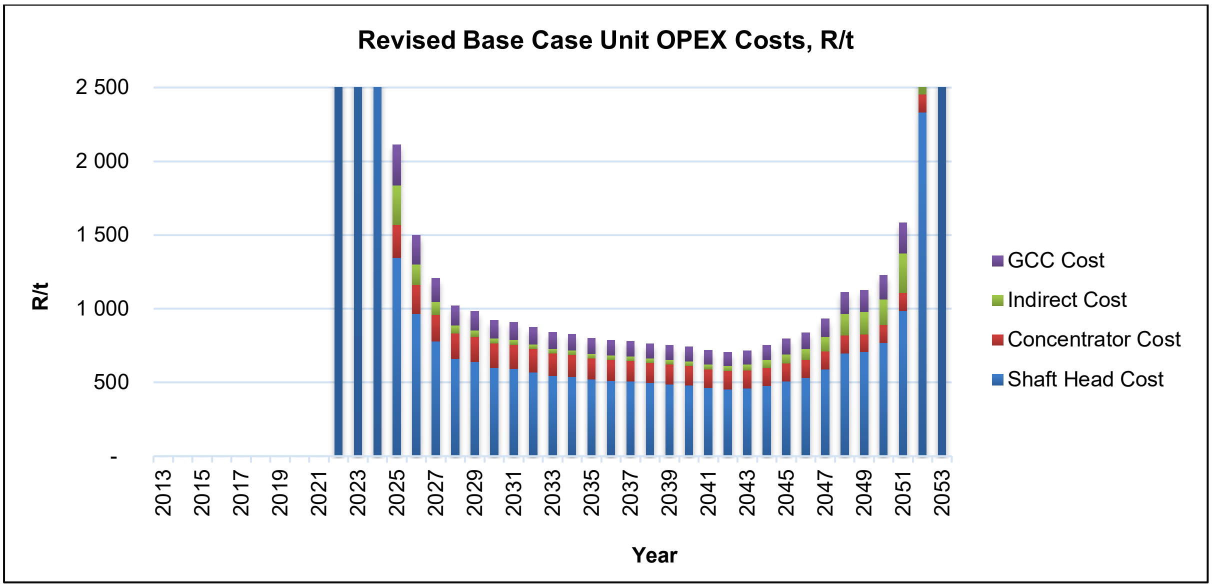

Figure 61: Base Case unit costs

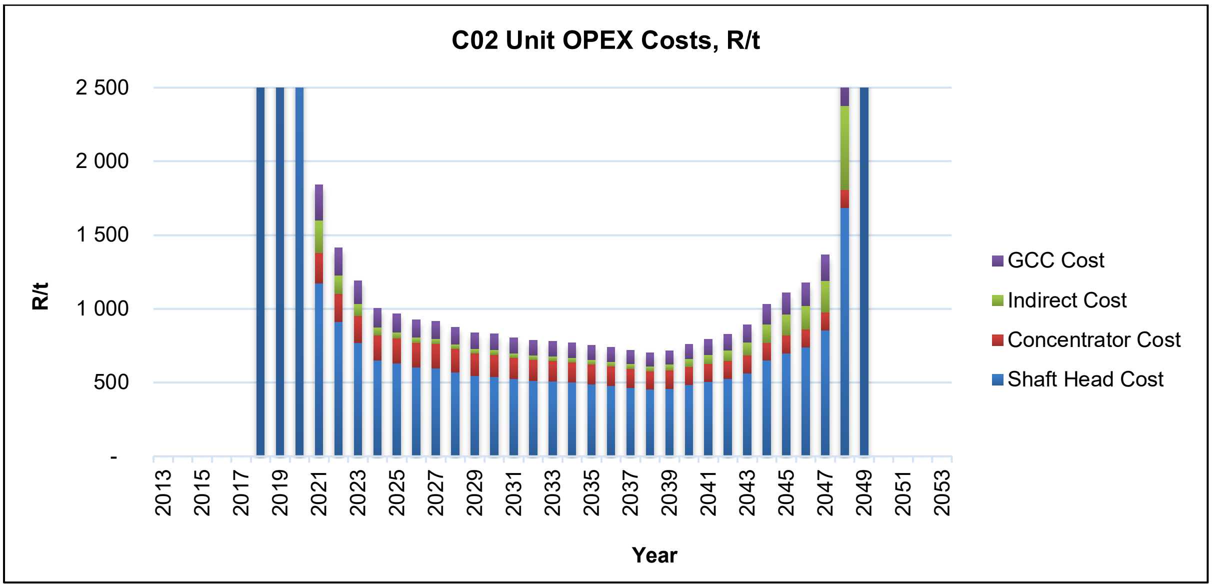

Figure 62: C02 unit costs

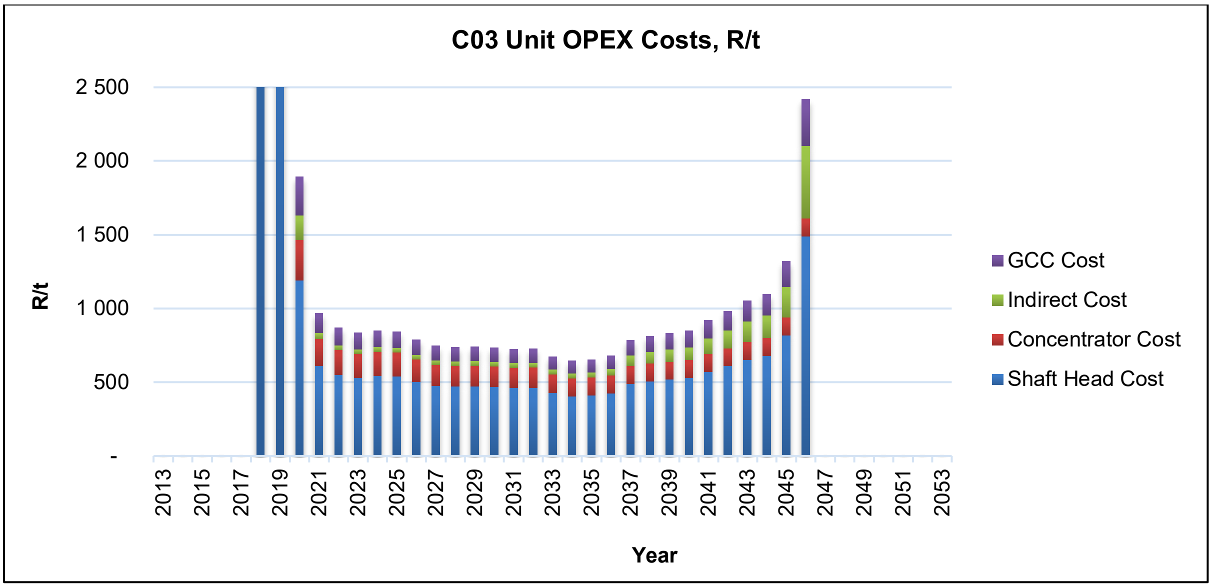

Figure 63: C03 unit costs

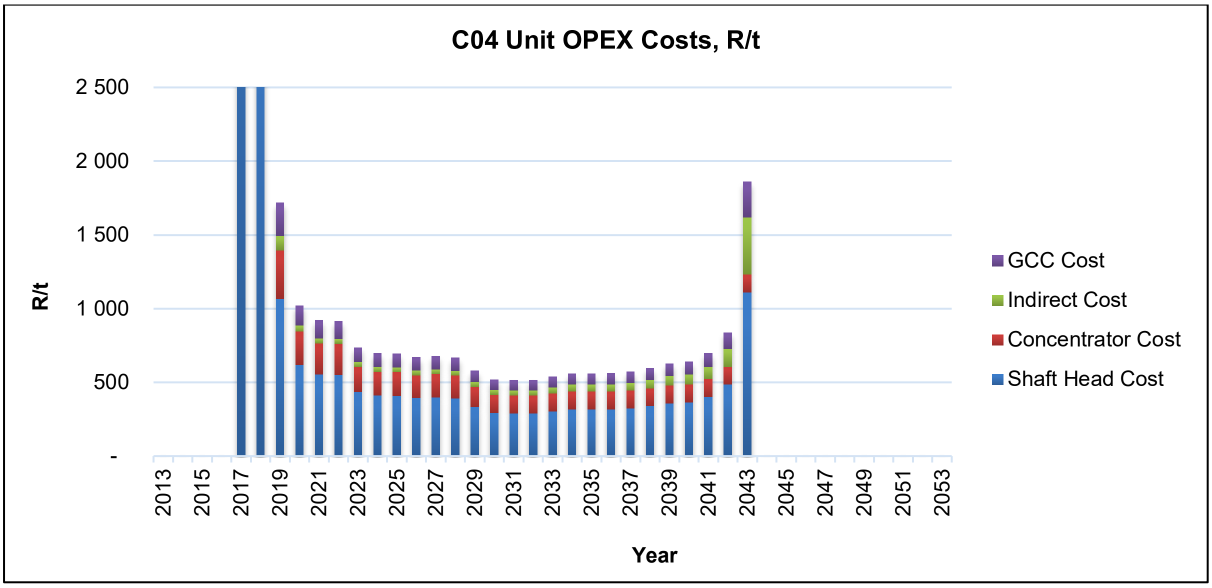

Figure 64: C04 unit costs

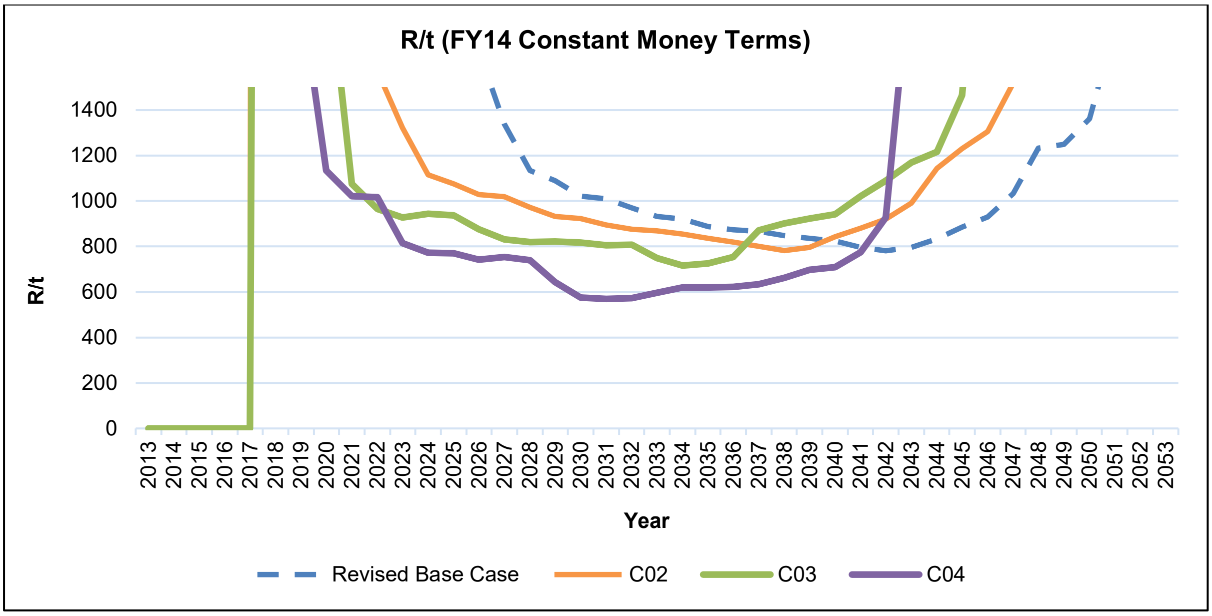

Figure 65: Footwall Developed Scenarios unit costs comparison

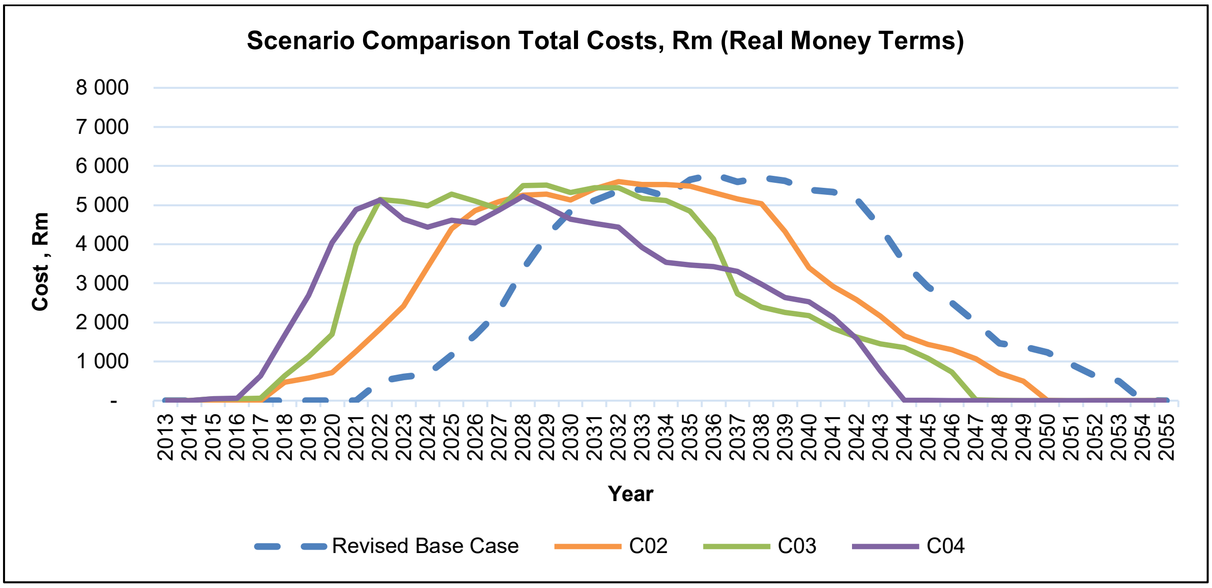

Figure 66: Footwall Developed Scenarios Total Cost comparison

13 On Reef Developed Scenarios

Table 105: On Reef Developed scenarios operating costs

	Unit	**Cost**
Utilities (Power & refrigeration)	R/t	48
Water	R/month	8 000
Stoping stores	R/m^2	251
Concentrator	R/t	115.08

Table 106: MTM operating costs

MTM	Unit	Costs
Roof support, consumables, energy & labour	$/m	2 000
Roof support, consumables, energy & labour	R/m	20 960

Table 107: TBM operating costs

TBM	Unit	Costs
Spares and maintenance	R/m	14 127
Power	R/m	687.7

Table 108: RMDS operating costs

RMDS	Unit	Costs
Cutter cost	R/t	90
Spares and Maintenance (MARC)	R/t	56
Electricity	R/t	14.3
Utilities	R/t	8.8
Stores cost	R/t	47
Total	**R/t**	**216.1**

Table 109: LHD operating costs

LHD	Unit	Value
Uptime hours	hrs.	17.5
Days per annum	days	23
Hours per annum	hrs.	4 830
Rate	R/hr	618.18
Total	**R/yr**	**2 985 809**

Table 110: Conveyor belt operating costs

Conveyor belt	Unit	Costs
Operating	R/t	10
Belt moves	R/m	10 000

Table 111: Tip operating costs

Tips	Unit	Costs
Tip moves	R/tip	500 000

Table 112: Slot boring machine operating costs

Slot Boring Machine	Unit	Costs
Cutter cost	R/t	105
Spares and Maintenance	R/t	26.36
Electricity	R/t	17.40
Utilities	R/t	19.11
Stores cost	R/t	8.43
Sundry	R/t	0.60
Total	**R/t**	**176.90**

Table 113: ULP Fleet operating costs

ULP Fleet	Unit	Costs
Stoping Stores	R/t	89
Stoping machine maintenance stores	R/t	63
Logistics HL belt stores	R/t	30
Total	**R/t**	**182**

Table 114: Sandvik MN 220 operating costs

Slot Boring Machine	Unit	Costs
Cutter cost	R/t	105
Spares and Maintenance	R/t	27
Electricity	R/t	5.50
Utilities	R/t	9
Stores cost	R/t	9
Sundry	R/t	1.40
Total	**R/t**	**157**

Table 115: On Reef Developed Scenarios Total Costs breakdown

	M01	M01b	M03	M04	M05	M06
Shaft Head Cost (Rm)	18 968	18 984	18 763	18 714	18 445	17 180
Concentrator Cost (Rm)	6 997	6 977	6 997	6 704	4 658	5 115
Indirect Cost (Rm)	1 229	1 229	1 229	1 371	1 374	1 164
GCC (Rm)	4 552	4 511	4 407	5 524	4 143	3 950
Total Cost (Rm)	**31 746**	**31 701**	**31 396**	**32 313**	**28 620**	**27 409**

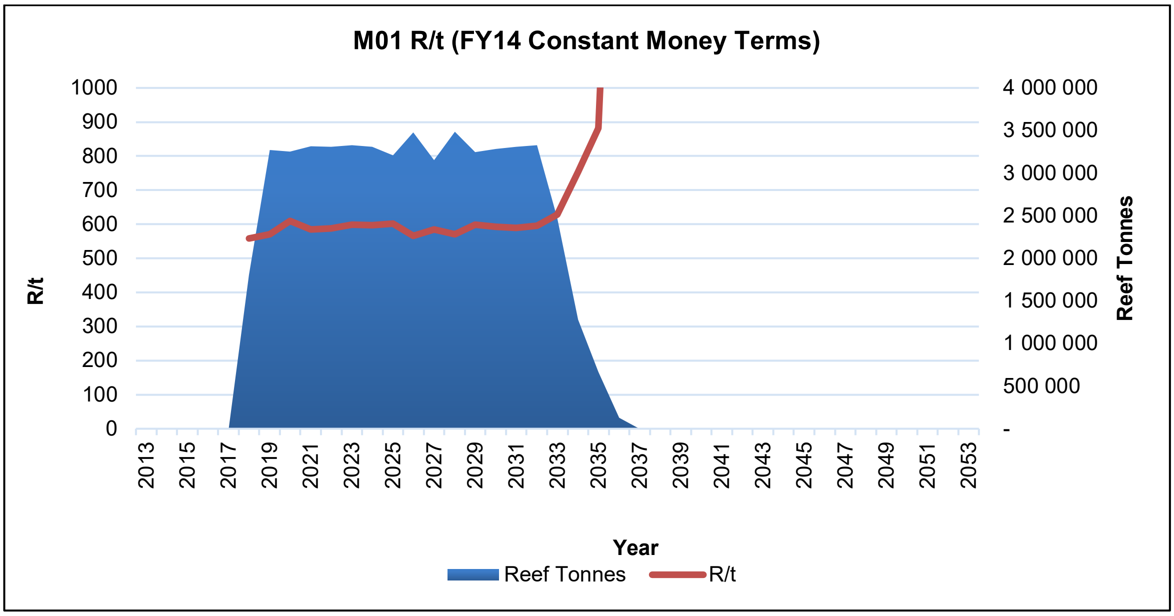

Figure 67: M01 tonnage profile and unit costs

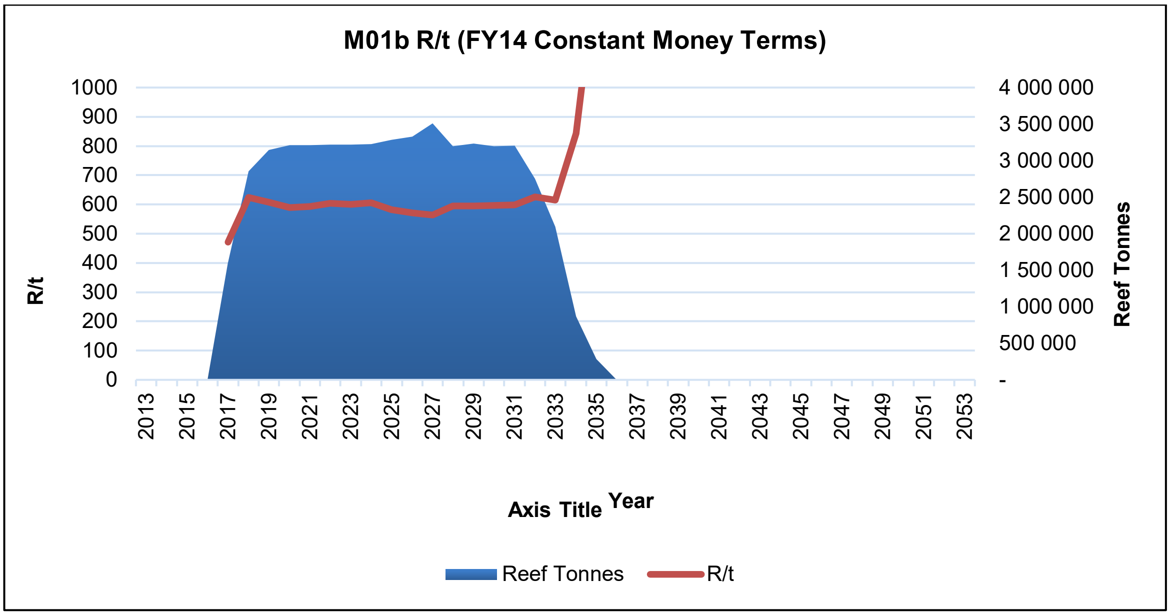

Figure 68: M01b tonnage profile and unit costs

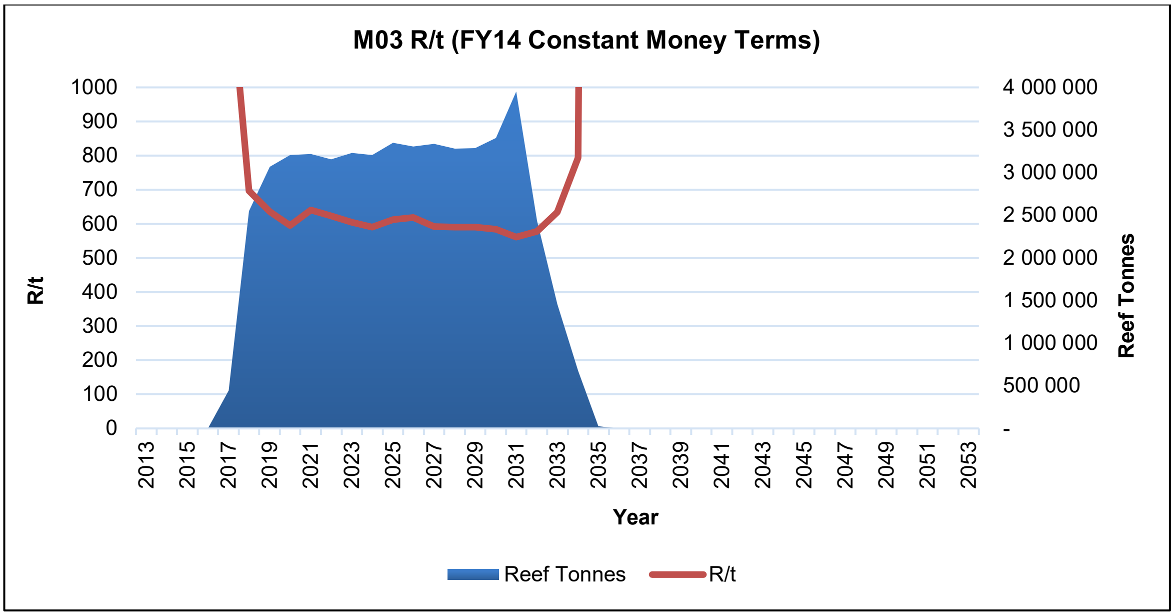

Figure 69: M03 tonnage profile and unit costs

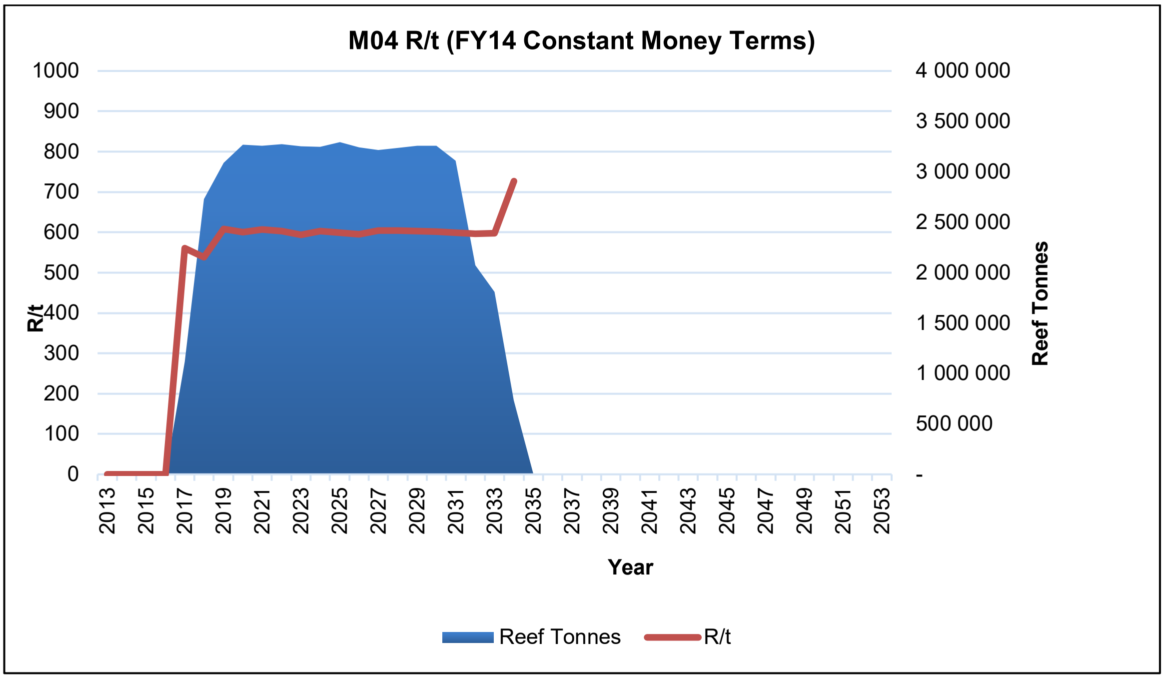

Figure 70: M04 tonnage profile and unit costs

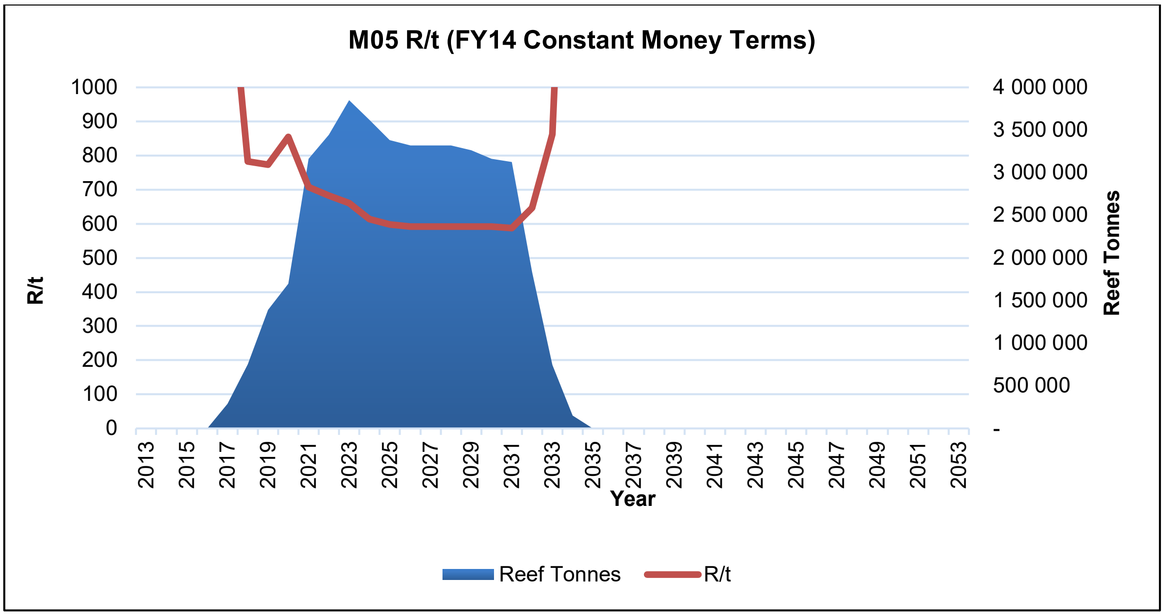

Figure 71: M05 tonnage profile and unit costs

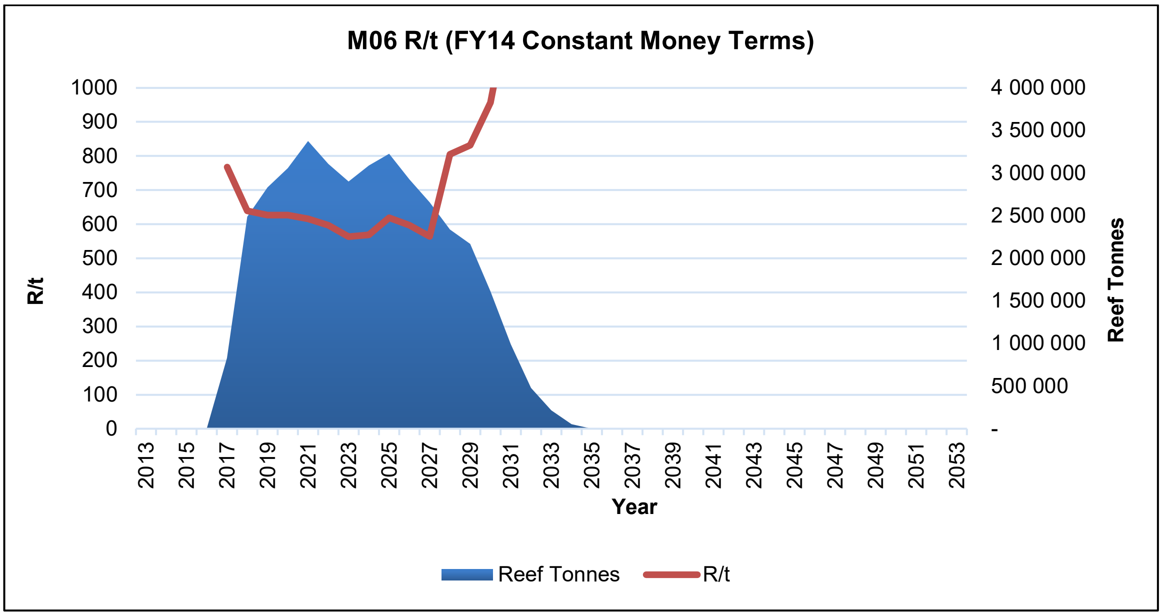

Figure 72: M06 tonnage profile and unit costs

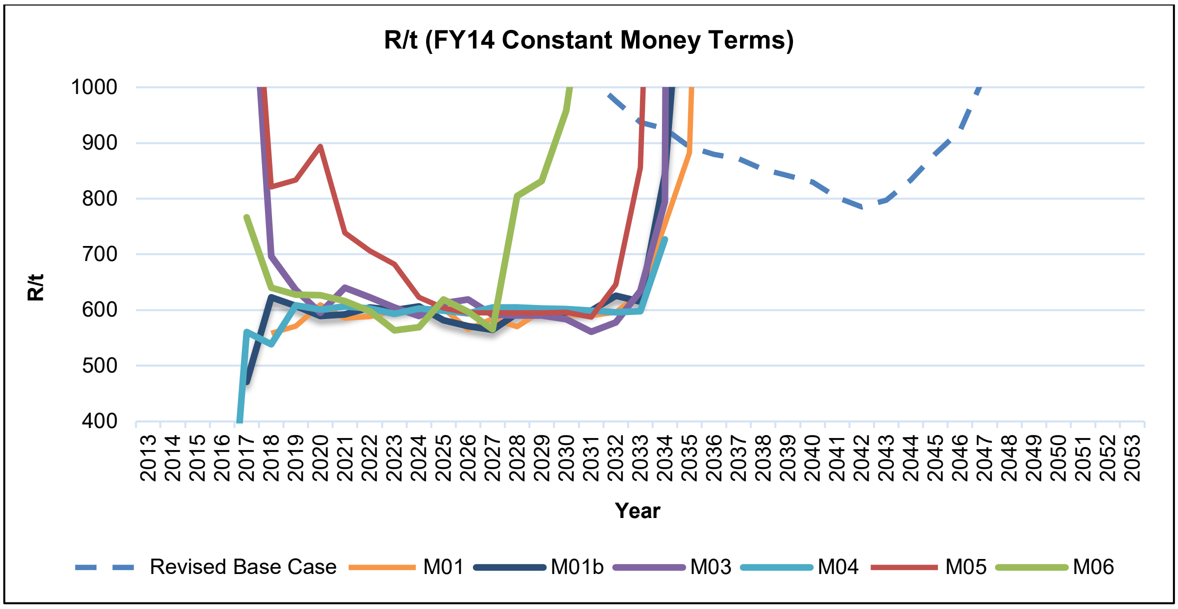

Figure 73: On Reef Developed Scenarios unit costs comparison

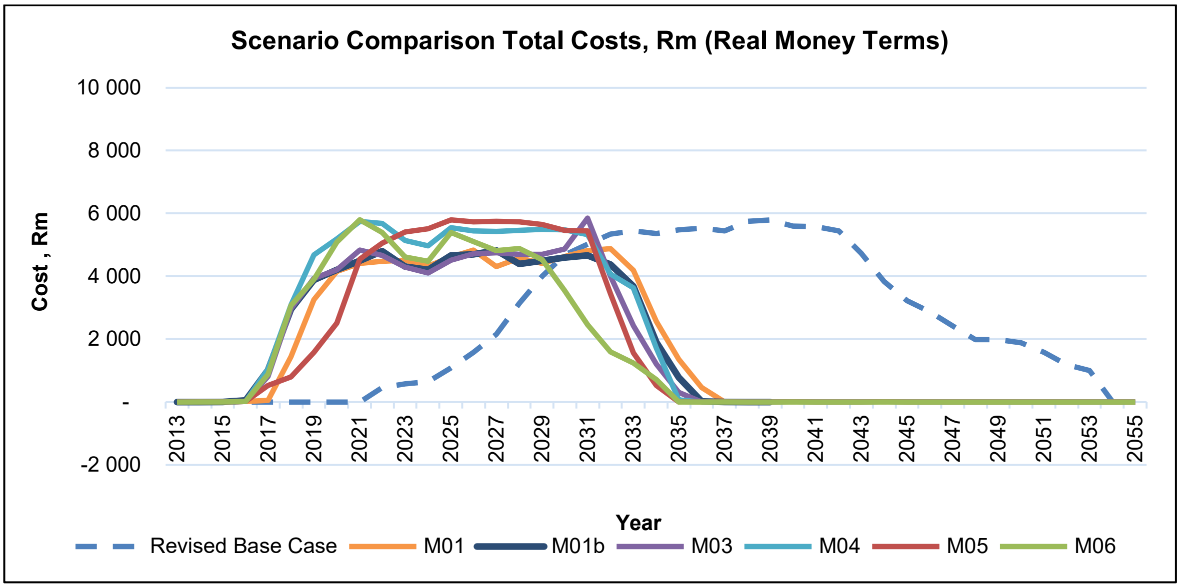

Figure 74: On Reef Developed Scenarios Total Cost comparison

Table 116: Footwall Developed Scenarios Labour count comparison

	Revised Base Case	C02	C03	C04
Total Employee Count	5 622	5 522	5 428	5 383
R/Employee	179 319	179 230	184 230	187 537
Total Labour Cost (Rm)	21 008	21 014	20 967	19 567

Table 117: On Reef Developed Scenarios Labour count comparison

	Revised Base Case	M01	M01b	M03	M04	M05	M06
Total Employee Count	5 622	1 436	1 310	1 343	1 746	4 718	1 616
R/Employee	179 319	297 174	286 839	270 994	244 324	192 713	231 549
Total Labour Cost (Rm)	21 008	6 496	7 718	6 915	7 267	11 598	5 332

15.0 Appendix 4 - Mine Layouts

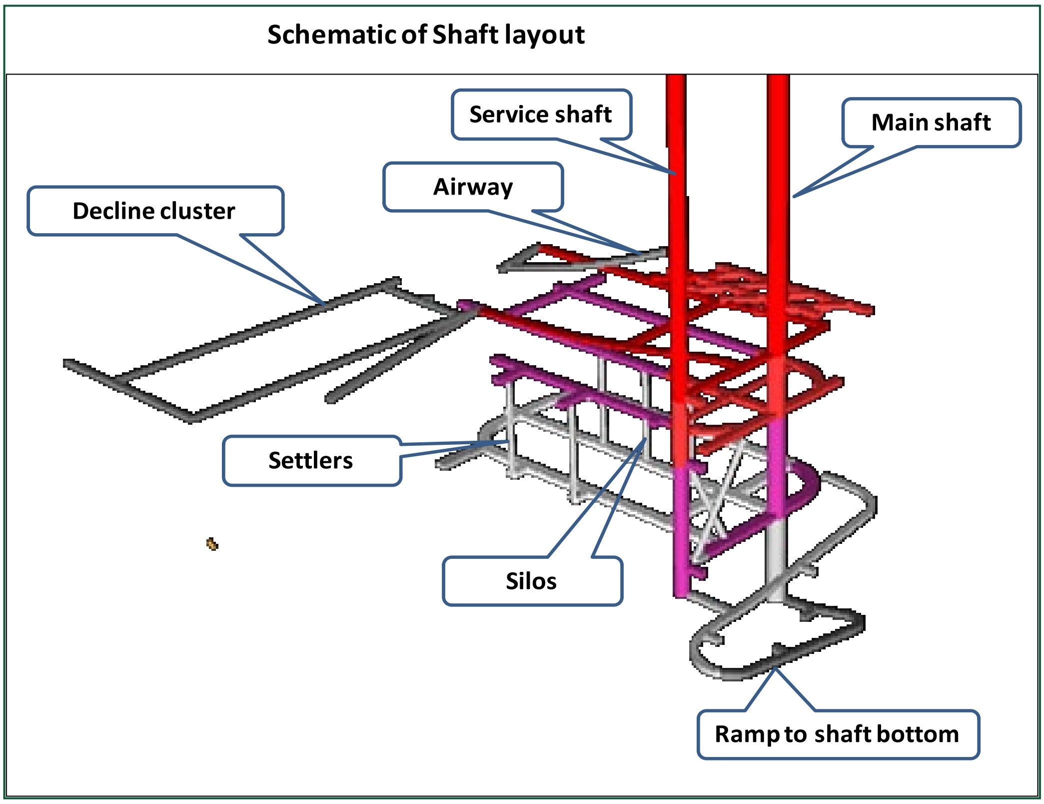

Figure 75- Mine Shaft Layout

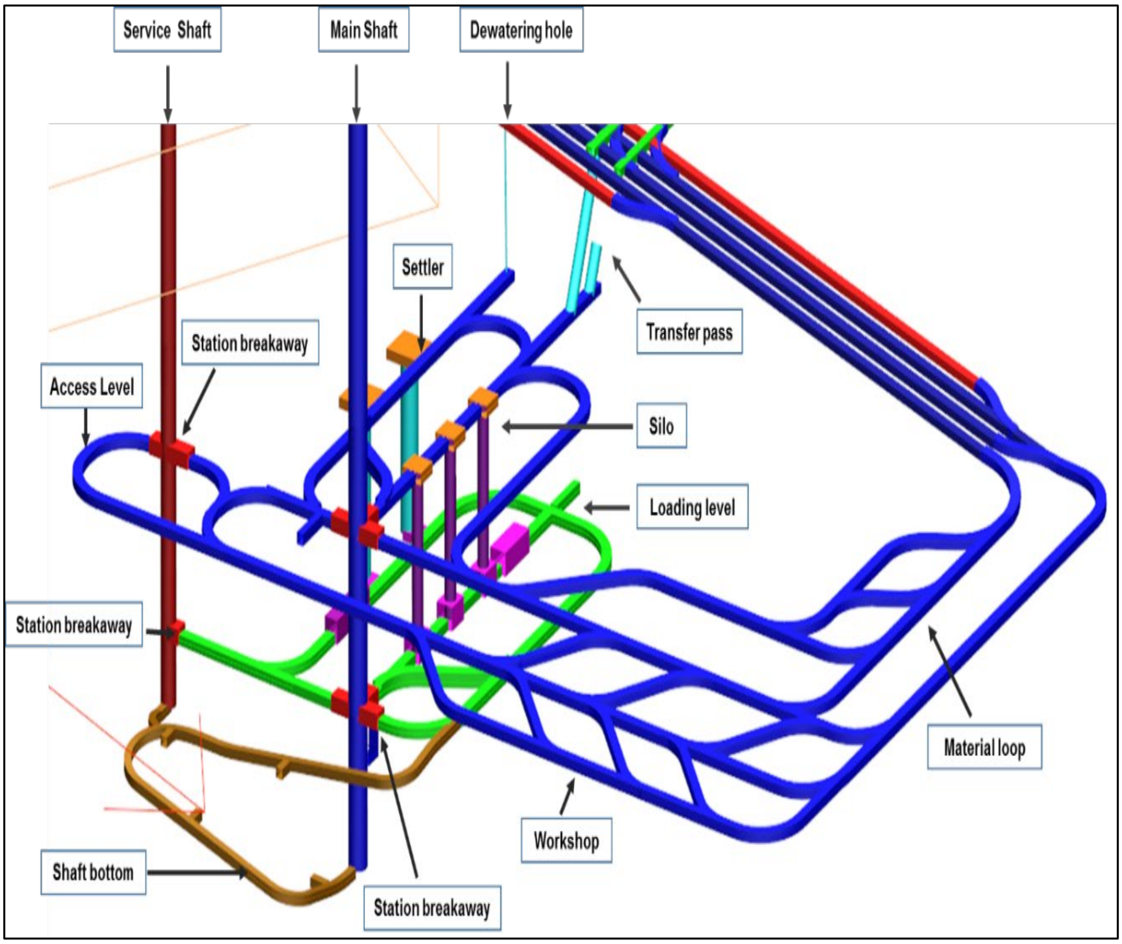

Figure 76: Station and Project Development Layout

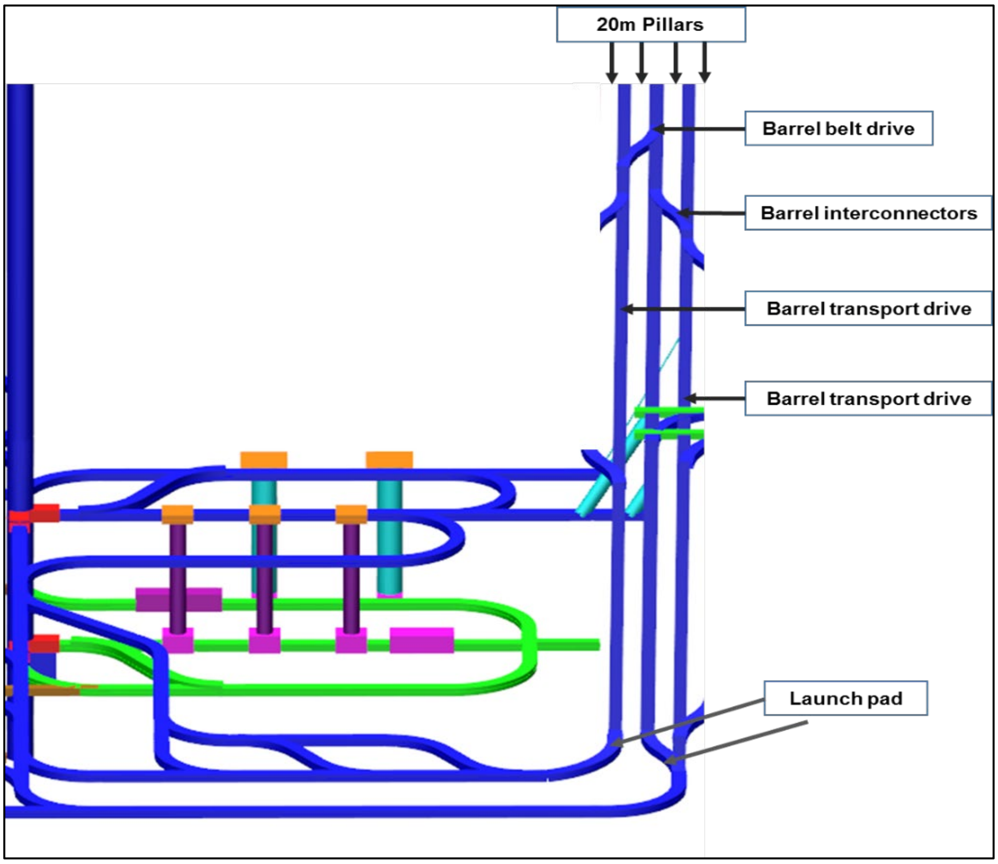

Figure 77: Decline Cluster System

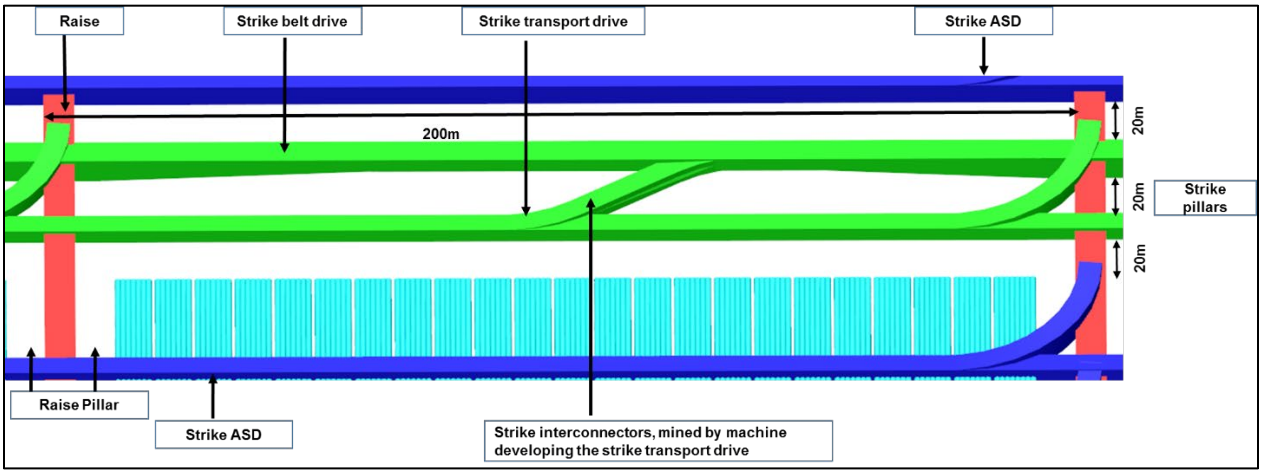

Figure 78: Working Cost Development design elements

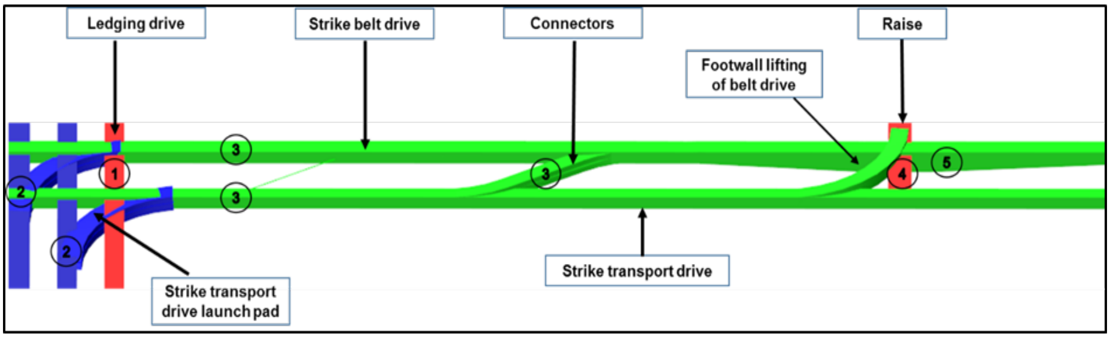

Figure 79: Strike belt and transport drive sequence

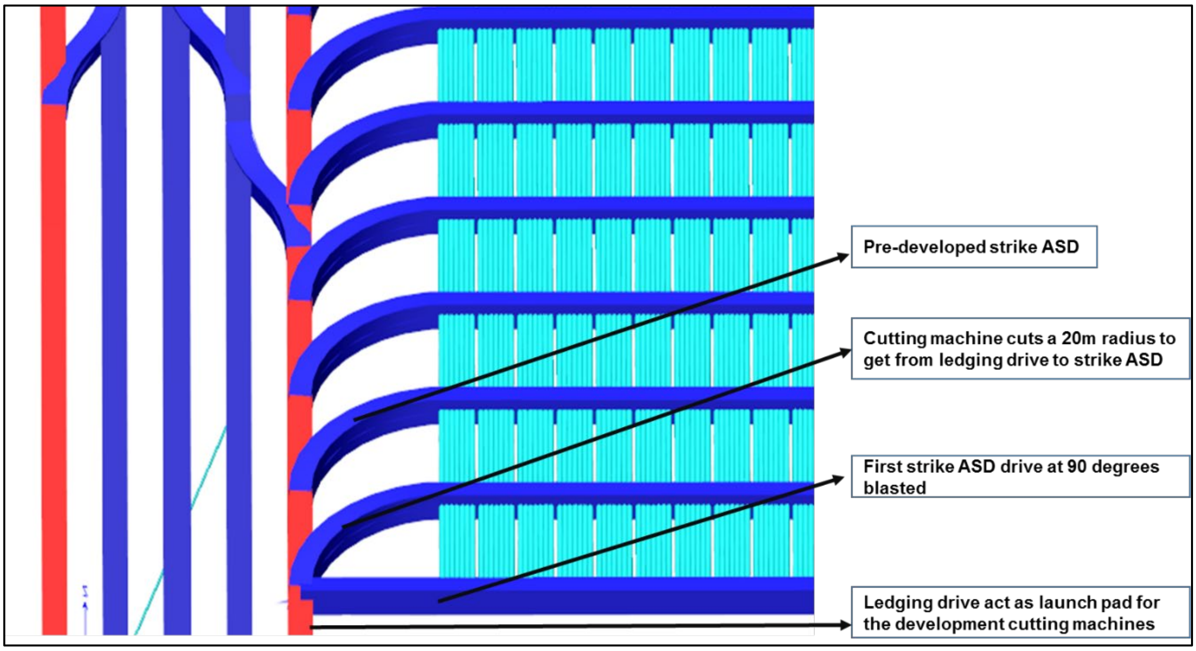

Figure 80: Launching of strike ASD

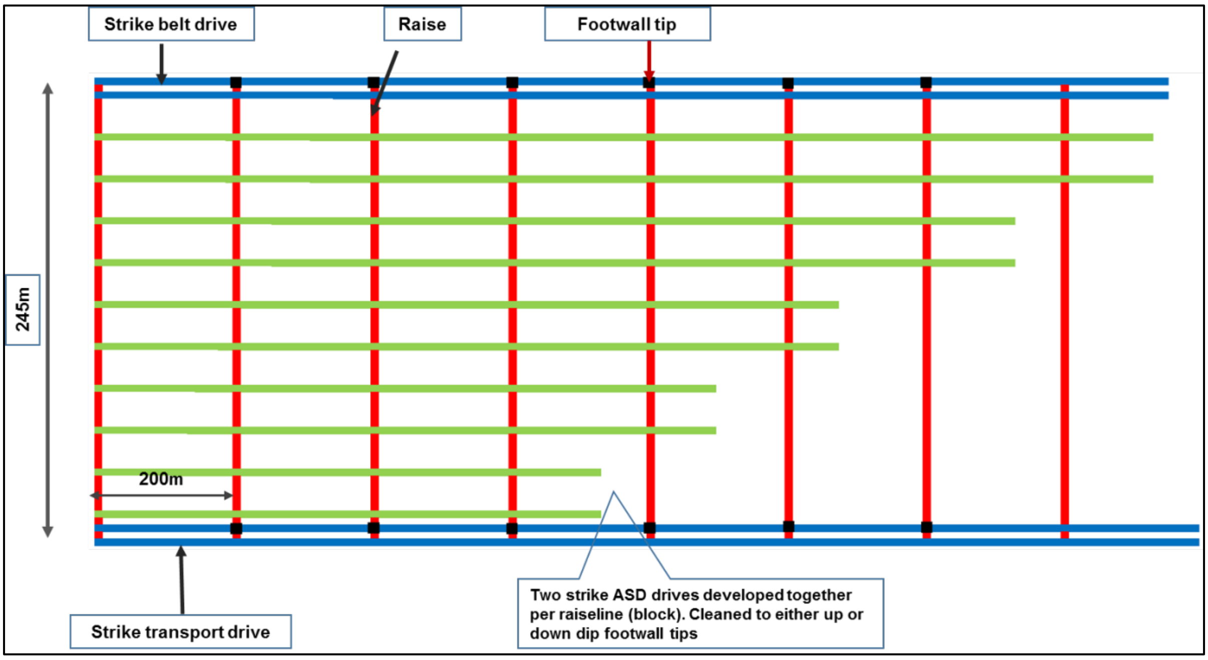

Figure 81: Half level working cost development sequencing

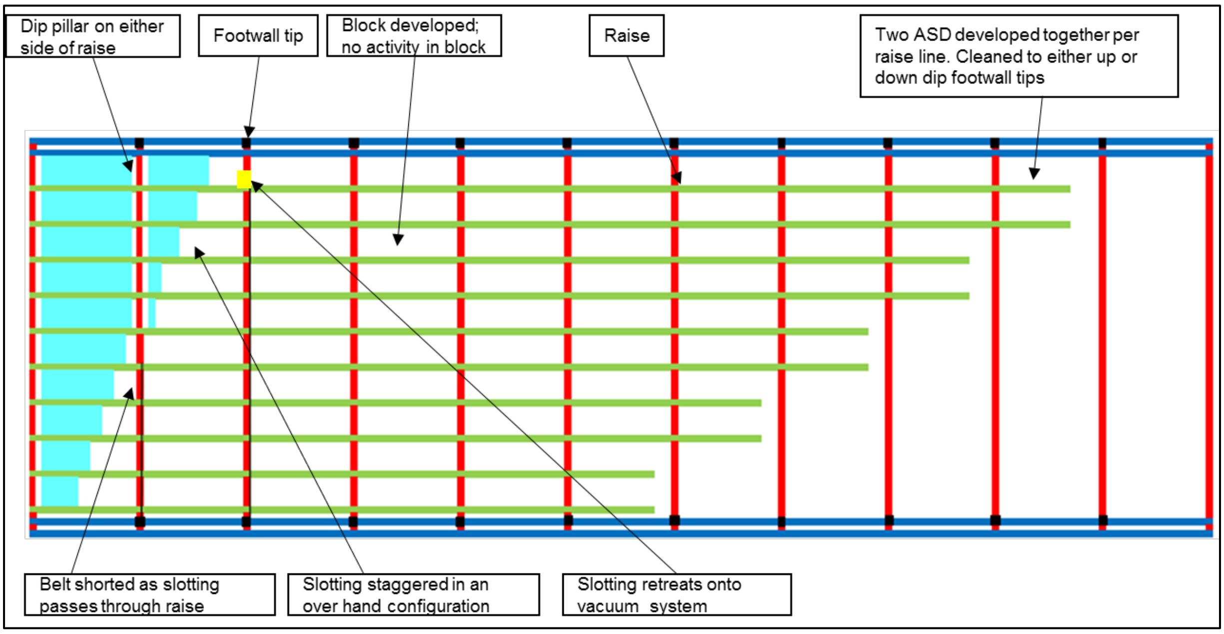

Figure 82: Half level development and slotting sequence

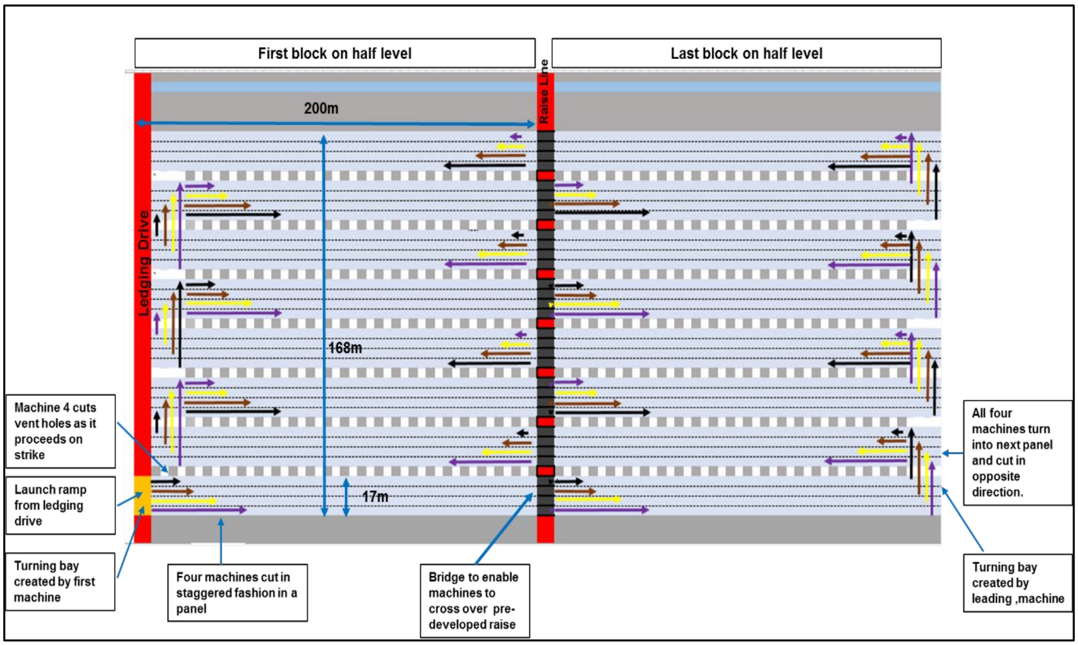

Figure 83: Sandvik MN 220 typical block layout

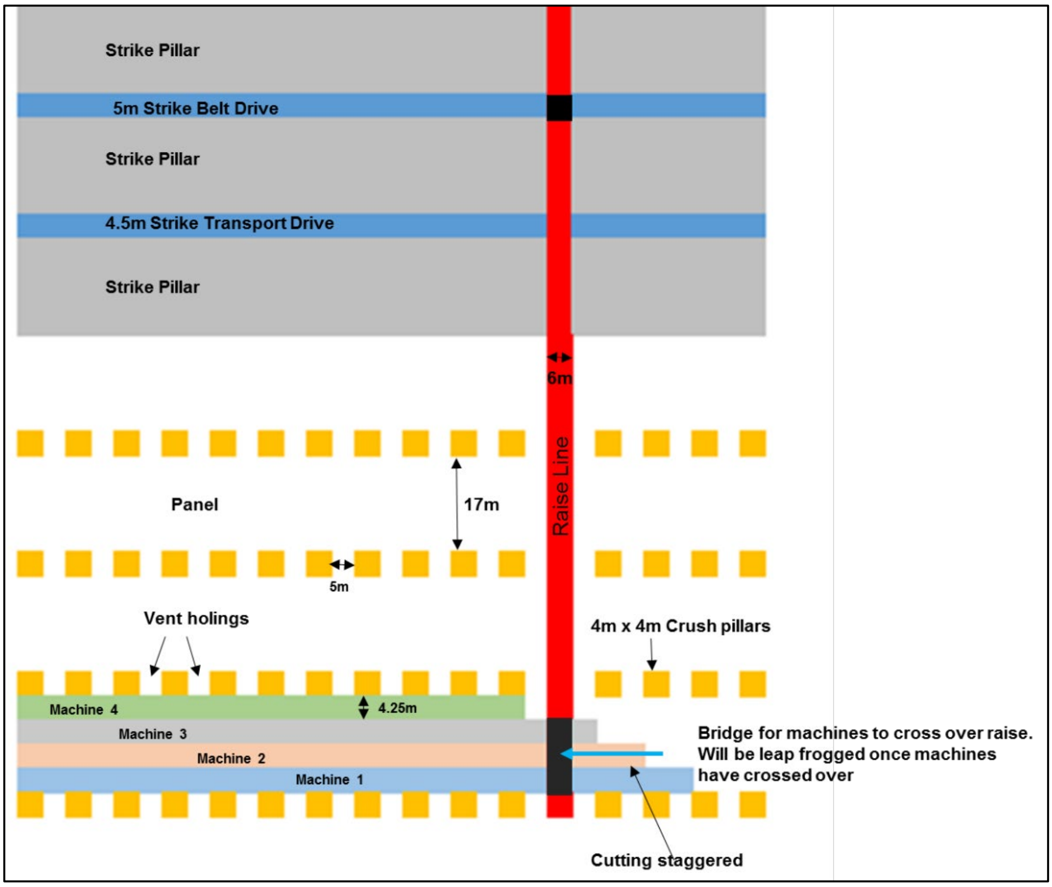

Figure 84: Sandvik MN 220 Panel layout

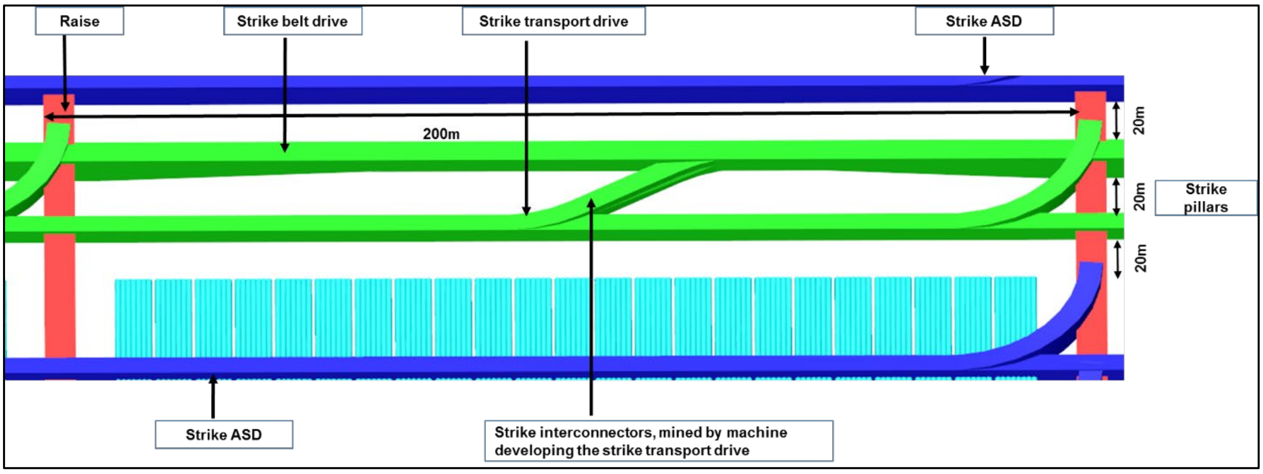

Figure 85: Strike pillar dimensions

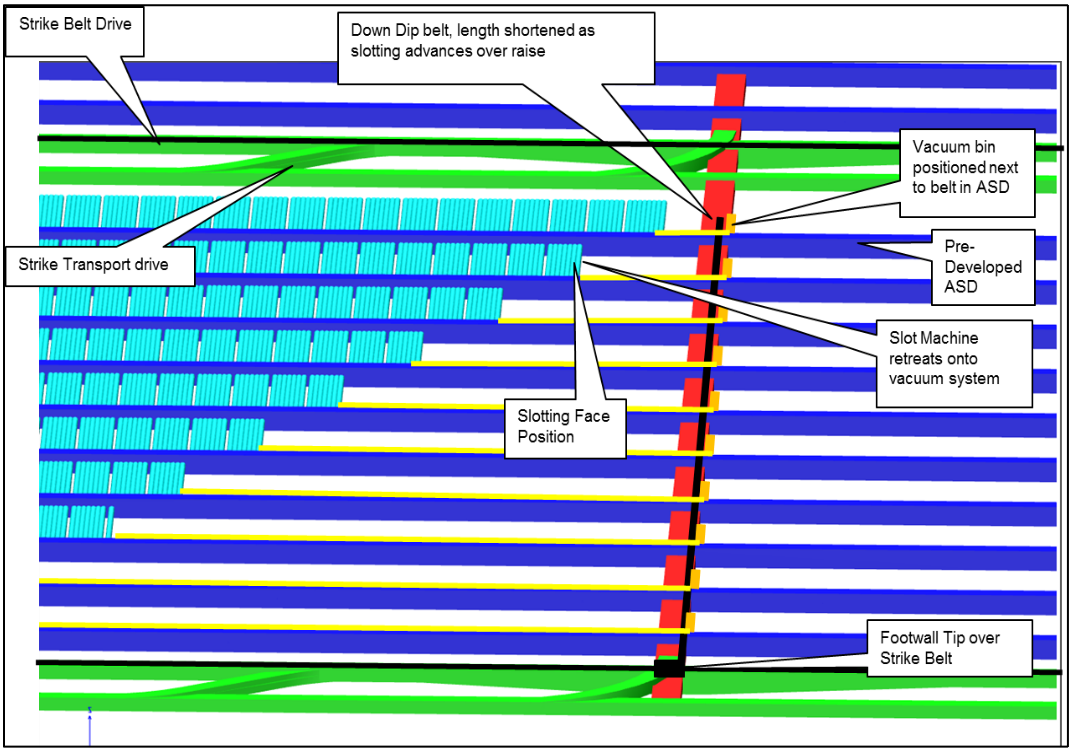

Figure 86: Slotting Layout

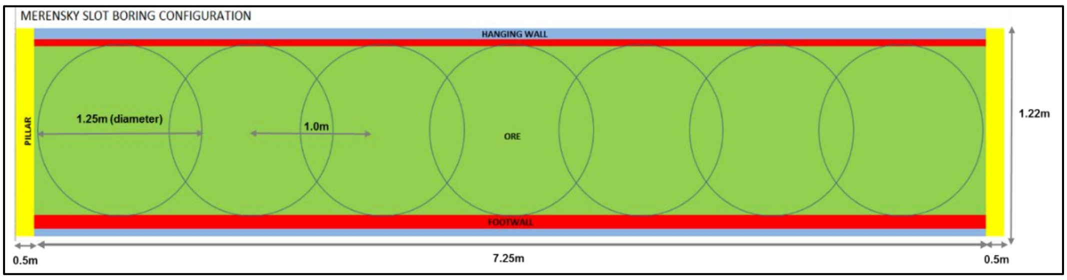

Figure 87: Slot boring configuration

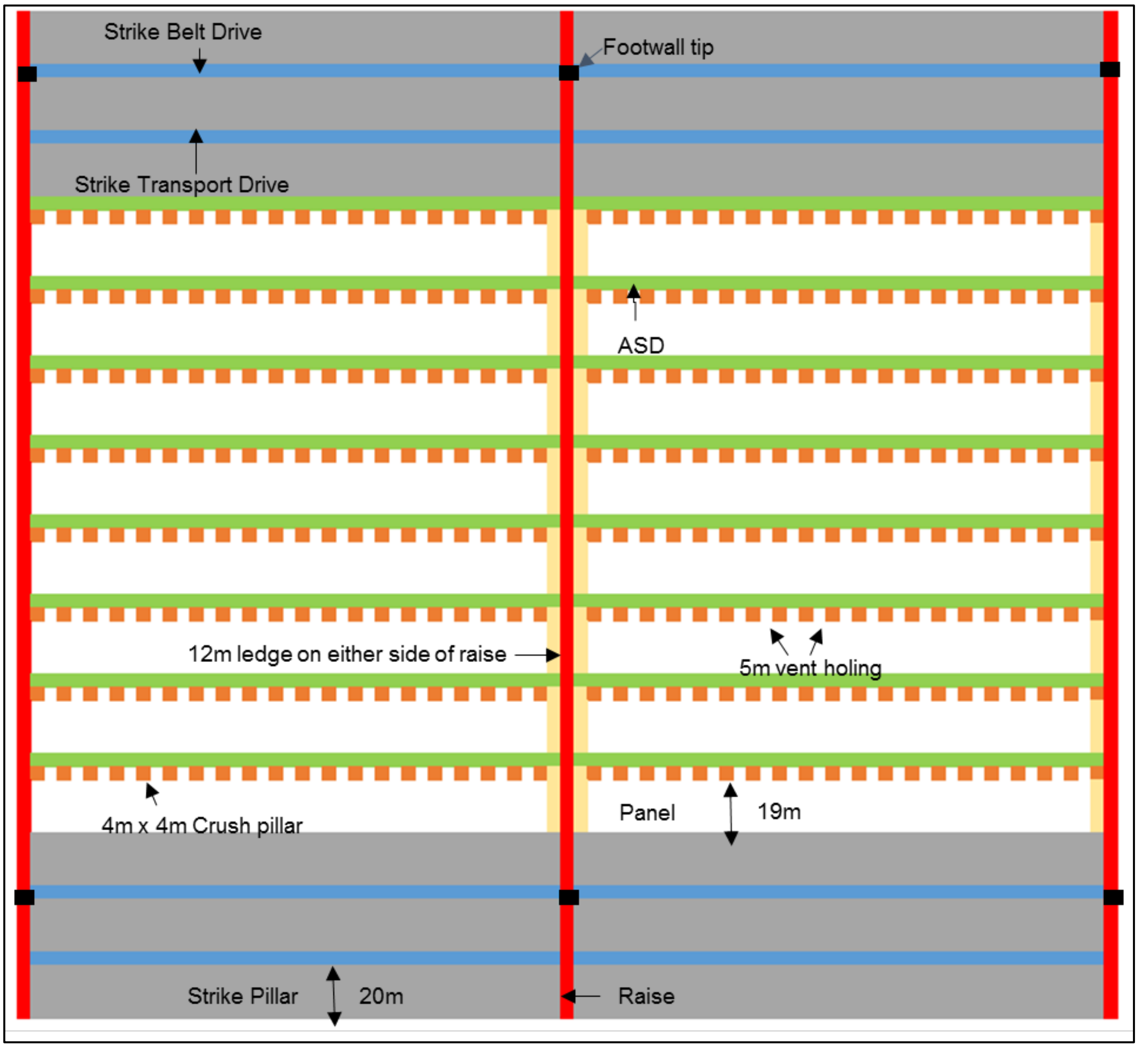

Figure 88: ULP Stoping Mining Layout

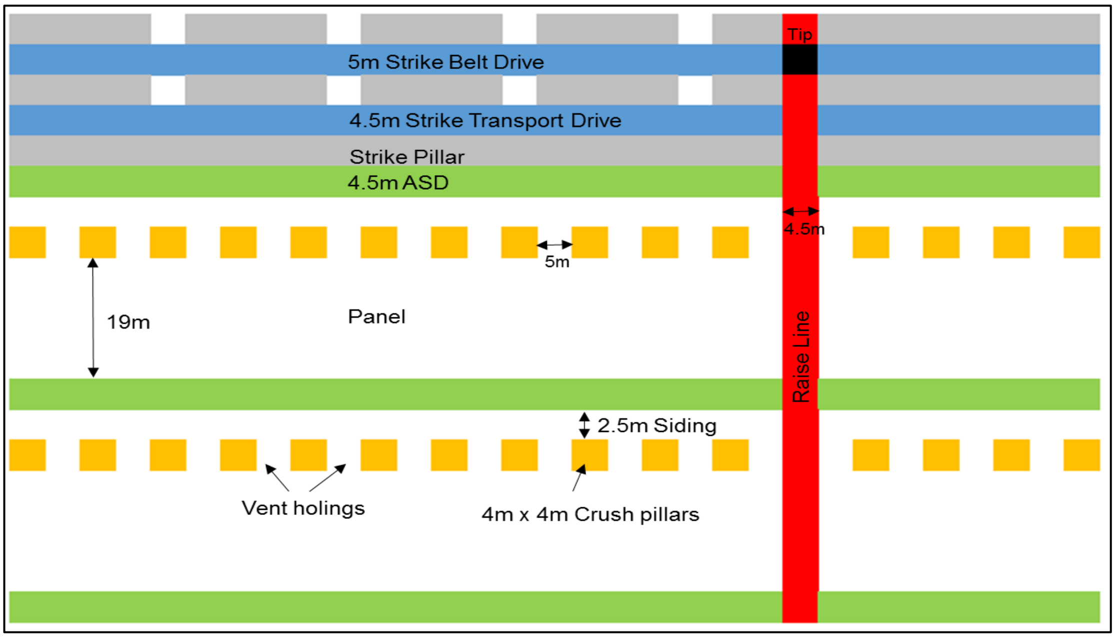

Figure 89: ULP Panel Layout

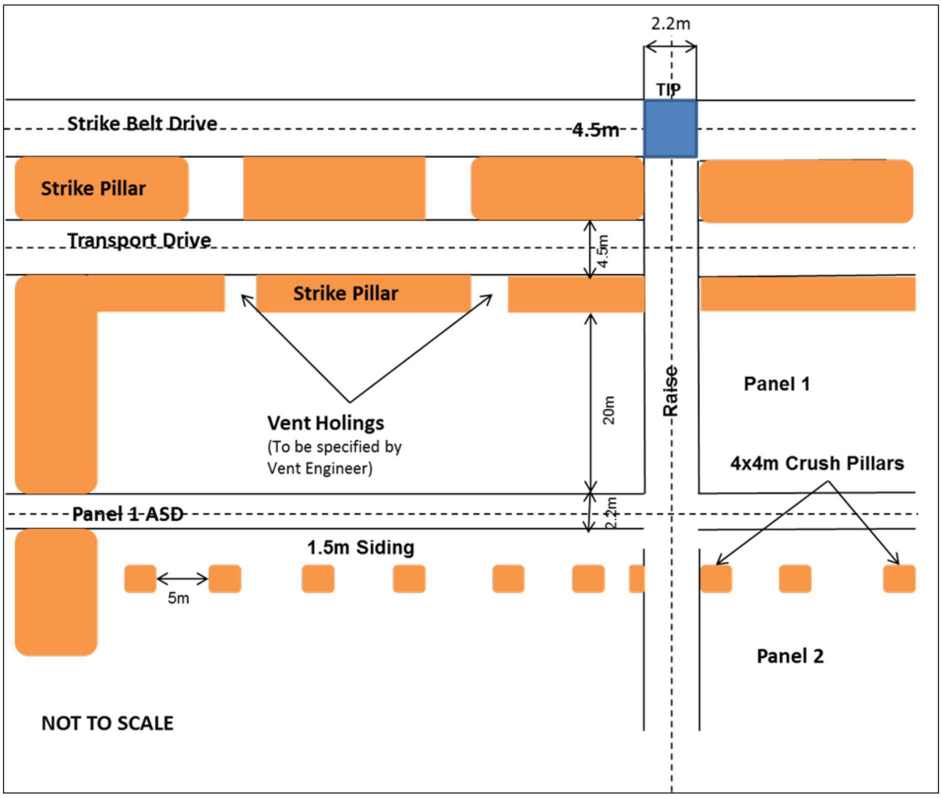

Figure 90: Conventional drill and blast stope layout