School of Mining Engineering

UNIVERSITY OF THE WITWATERSRAND, JOHANNESBURG

THE EFFECT ON NET PRESENT VALUE OF CUT–OFF GRADES APPLIED TO VARIOUS REEF TYPES USING DIFFERENT MINING METHODS

Eric Denzil Thomas Meredith

A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, in partial fulfilment of the requirements for the degree of Master of Science in Engineering.

DECLARATION

I declare that this report is my own, unaided work. I have read the University Policy on Plagiarism and hereby confirm that no plagiarism exists in this report. I also confirm that there is no copying nor is there any copyright infringement. I willingly submit to any investigation in this regard by the School of Mining Engineering and I undertake to abide by the decision of any such investigation.

____

E.D.T Meredith Date: 20 April 2021

ABSTRACT

Cut-off grade is influenced by economic factors including costs. Different mining methods have different costs associated with them. Most operations would only employ one mining method in the extraction of ore. In this research, several mining methods are used in the extraction process and by optimising these, the net present value is increased. The formulation and recording of costs lay the foundation for reliable cost information associated with each mining method. Without reliable cost information, the real cost of each mining method cannot be ascertained. The traditional approach to cut-off grade calculation used by the operation was compared with the new approach of a cut-off grade per mining method, and in so doing, there was a 3% improvement in cut-off grade leading to improved reserve tonnages and gold content. The results of the calculation were modelled per mining method in the comparison showing a 2% improvement in the net present value. The improved results were then simulated through the Monte Carlo simulator, @Risk and these were further enhanced. From the study, it was concluded that by the use of Activity-Based Costing and the analysing of costs per mining method, the cut-off grade could be reduced, and the net present value increased.

DEDICATION

All praise, honour and glory to my Lord Jesus Christ for His richest grace and mercy for the accomplishment of this project. To my wife Sandra and our children, Jayden and Daniella who had to sacrifice time and deal with my moods; this is dedicated to your love and kindness.

ACKNOWLEDGEMENTS

I would like to extend my gratitude to my colleagues at Gold One Group Limited for allowing me to complete this work. Not only for their financial assistance but the moral support that I received from them. A special thanks to Jiju Yuan, Jon Hericourt, Louis Bezuidenhout and all the members of the management team past and present.

Thank you to Izak Marais who dared to involve me in all aspects of the operation, which allowed my knowledge to grow. His ongoing support in my career has made it possible for me to achieve what I thought was impossible.

To Dr Le-Marlie Marais, thank you for your motivation and guidance throughout this project; for always being willing to assist with last-minute reviews and changes.

Evan Cook, who was a constant source of support and assistance particularly with the technical geological items in this research.

A special thank you for the support of all the staff at the University of the Witwatersrand School of Mining. A very special word of thanks to my supervisor Mr Clinton Birch for his amazing support not only with this project but throughout my academic journey with the Master's programme.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

LIST OF ABBREVIATIONS

ABC: Activity-Based Costing

AIC: All-in Cost

AISC: All-in Sustaining Cost

AOC: Adjusted Operating Cost

ASG: Advance Strike Gully

B&P: Bord and Pillar

BFS: Bankable Feasibility Study

BPLZ: Buckshot Pyrite Leader Zone

cm.g/t: Centimetre-grammes per tonne

cm: Centimetre

Dil: Dilution

FS: Feasibility Study

g/t: Grammes per tonne

GTC: Grade-Tonnage Curve

m: Metre

mbs: metre's below surface

MCF: Mine Call Factor

NPV: Net Present Value

PC: Process Costing

PRF: Plant Recovery Factor

R/t: Rand per tonne

SC: Standard Costing

UK9A: Reef within the Kimberley Reef

WGC: World Gold Council

1 INTRODUCTION

1.1 Overview of Chapter 1

Chapter 1 outlines the purpose of this study and highlights the key elements of the study namely, costs in mining and cut-off grades. The chapter also states the motivation for the research; to improve cut-off grades which leads to a better resource to reserve conversion and an improved net present value (NPV). The chapter also lays out the problem statement and in it, the focus of optimising the NPV whilst maintaining the life of mine. The sources of data and the structure of the report are discussed.

1.2 The Purpose of the Study

The purpose of this study was to ascertain the effect of combining different mining methods with different reef types to increase the NPV of the Modder East operation. While a fair amount of research has been done on cut-off grades, not much of it shows the impact on value through different mining methods. Each mining method has a different operating cost associated with it. Using a blend of mining methods to find the most cost-effective mining method for the operation increased the NPV.

1.3 Costs in Mining

While management accounting plays a vital role in mining today, the management accountants are generally relegated to the back office of the operation. The management of costs is fundamental to the management of an operation, not only in reducing costs but also in ensuring that costs are correctly allocated to the various cost centres. Management accounting can play a role in the strategic management of operations (Stewart, 2014). For a long time, costs were incorrectly reported in the public domain. The financial results could not be reconciled to the cash operating cost metric. This inconsistency led to the formulation of new reporting metrics spearheaded by the World Gold Council. All-in Sustaining Costs (AISC) and All-in Costs (AIC) define the elements that are fundamental to cost reporting (Yapo & Camm, 2017).

Costs must be reported accurately, but they also need to be allocated correctly to ensure that the correct business decisions are made. To this end, costing systems have come under the spotlight, and one such system is Activity Based Costing (ABC). The ABC system deviates from the traditional approach of Process Costing (PC) by allocating costs to activities rather than processes. This change in thinking has led to improvements in the NPV of operations using ABC (Lind, 2001).

1.4 Cut-off Grades

Cut-off grades describe the minimum amount of mineral contained in a deposit that can be mined and processed economically. It is defined not only by geological characteristics, but by the cost of mining and processing, and the selling price of the mineral (Taylor, 1972). While the definition for a cut-off grade is a simple one, the determination of a cut-off grade is complicated by the fact that orebodies are not alike. The orebody, in all instances, dictates the mining method to be used to extract the mineral. Traditionally, cut-off grades have been calculated per reef type only, disregarding the differences in geology and recoveries. In these instances, the traditional method of cut-off grade calculation has been the breakeven approach. With the recent advances in technology, more models have been postulated, which aim to optimise the cut-off grade and thereby the value of the operation (Githiria & Musingwini, 2018). The value optimisation approach frequently followed, leads to the cut-off grade dilemma. Whether to optimise the operation for life or value remains the cut-off grade dilemma faced by operations (Mugwagwa, 2017).

1.5 Research Motivation

This research study was motivated by the need to improve cut-off grades increasing the NPV of an operation. The research project demonstrated that due to the costs associated with various mining methods, the optimisation of the mining method mix leads to an improved NPV. The focus was on reducing the cut-off grade and thereby including more resource into the reserve. While there is a plethora of research done on cut-off grades, there appears to be little written on the optimisation of cut-off grades based on varying mining methods.

1.6 Problem Statement

The project investigated the effect of different mining methods with different costs structures on a variety of reefs with different recoveries. The application of different mining methods to an orebody, where possible, will allow the costs to be reduced, resulting in an improvement of the overall cut-off grade. The optimisation of extraction at the lowest cost will depend on the resource. The adage that "the resource dictates" is key to understanding how best to mine the resource. The initial Modder East Operation's bankable feasibility study (BFS) showed that the resource would be mined conventionally through conventional raise development and breast mining (Venmyn Deloitte, 2016). It became apparent relatively late in the life of the operation that this might not have been the only way to mine the orebody; with reefs being mined conventionally as a double-cut rather than mechanically, which comes at a much lower cost. The key focus is on demonstrating what the optimum strategy would be to mine economically to maximise NPV. This approach opposes that of the proponents of cut-off grade optimisation which comes at the expense of mine life.

Constraints in all areas of mining need to be defined and accounted for in the optimisation process. The effect of development on all reefs remains essential to the successful optimisation of the mining method mix. Simulations of historical mining based on existing knowledge will highlight the effect of different mining methods on the resultant NPV of the operation.

Given the falloff in grade, as anticipated by the BFS, the operation recently geared itself up to become a high-volume, low-cost gold producer. To this end, the metallurgical plant capacity was increased by approximately 33%. The increased plant capacity allowed for increased ore processing. The effect of this is also examined, although not considered lest it skews the results.

A pre-requisite of defining the cut-off grades according to reef type and associated mining method is that there needs to be a well-defined costing system structure in place to identify the costs associated with each respective mining method. ABC is key to a successful outcome. In the past, mining costs were applied generically and not per mining method. This research shows that the cut-off grades would be reduced by optimising (where possible) by mining method based on cost and higher volume throughput, keeping the plant producing at full capacity. The combination of lower cost and higher volumes will have the effect of reducing the cut-off grade and improving the resource to reserve conversion rate. Drawing on existing theories and research, this research demonstrates that operations' NPV could be improved through the optimisation of cut-off grade according to the most appropriate mining method associated with respective reef types.

1.7 Sources of Data

The data used in this report was collected from Gold One: Modder East Operation's internal documentation. The data is based on geological and geo-statistical data from the operation. The data includes financial data, data pertinent to the costing system used, as well as the costs themselves.

The actual data for the operation is modelled using the technical mining, geological and cost data to compare the outcomes of using one or more mining methods to extract the resource. Monte Carlo Simulation through @Risk software is used to optimise the available resources and the most cost-effective mining method mix in an attempt to maximise the net present value of the operation (Palisade, 1984).

1.8 Structure of the Report

The report begins with Chapter 1 which introduces the research. Chapter 2 lays the foundation for the research by reviewing pertinent scholarly material relevant to the topic in the literature review. This chapter is divided into the main themes of the research; namely, research on cut-off grade theory and practice, Monte Carlo Simulation theory and uses, and finally research on costs and cost analysis.

Chapter 3 leads the reader into the research by firstly describing the Modder East operation, the geology and mining characteristics, especially how the orebody was planned to be mined. After providing context, a technical discussion details how the cut-off grade for the operation was traditionally calculated, especially when there was only one mining method being used. The cut-off grade then determines the available resource in terms of tonnes and grade, which allows a simplistic model to be produced, costs established and an NPV to be calculated.

Chapter 4 focuses on costing, the costing methods used and how each mining method is costed using the Standard Costing (SC) and ABC approaches. It is necessary to explain the approaches to set up the next chapter which deals with costing of each of the different mining methods, how they are blended leading to a reduction in the operating cost and an improvement in the NPV.

Chapter 6 deals with the optimisation of the NPV through the optimal use of the available tonnes per mining method. The @Risk software was used to simulate the outcomes of blending the available tonnages produced by the various mining methods in an attempt to optimise the NPV of the operation. The optimisation objective used is on improving NPV without sacrificing the remaining life of the operation. Finally, conclusions and recommendations are made in Chapter 7.

2 LITERATURE REVIEW

2.1 Overview of Chapter 2

This chapter focuses on the scholarly research into the key aspects of this research study. Due to the volume of research into cut-off grades and cut-off grade optimisation, the focus is narrowed to the discussion around the various approaches to cut-off grade calculation and the effect that these have on NPV and the life-of-mine.

One of the key elements of the cut-off grade calculation is costs. There are numerous approaches to how costs are assimilated and measured. Costing systems and approaches lead to better information and improved decision making. Costs and the management thereof are crucial to improving value and resource to reserve conversion. In this study, various mining methods are used in the extraction process and these have different cost structures associated with them. To maximise value these mining methods are simulated to produce an optimum NPV. The simulation is done through Monte Carlo simulation. This chapter briefly researches Monte Carlo simulation and its use in enhancing NPV.

2.2 Cut-off Grades

One of the most critical decisions in mining is determining which proportion of the orebody can be mined profitably and which proportion is to be considered waste (Bascetin & Nieto, 2007). Cut-off grade is defined as: "the minimum grade required for a mineral or metal to be economically mined (or processed)" (Rendu, 2014). In this regard, there is a large body of research around cut-off grades and the application thereof. Within the body of knowledge, several applications and theories adapt cut-off grades based on the life cycle of an operation. The calculation of cut-off grades considers various technical and economic factors. Economic factors include product prices, mining and processing costs, and refining and selling costs. Technical factors include the geological data common to the orebody, including geological losses (Githiria & Musingwini, 2018).

While there is a consensus that the breakeven cut-off grade is the most popular approach in the mining industry, this approach is very rarely followed in its purest form. There are many adaptations, but it still draws criticism for excluding essential elements such as capacity constraints and sustaining capital (Poniewierski, 2016).

In their paper, Githiria and Musingwini (2018), compared the various approaches to cut-off grade models. They asserted that there are three approaches to cut-of grade models, namely, the breakeven cut-off grade approach, the heuristic cut-off grade approach and Lane's optimum cut-off grade approach. While the modern approaches show improved value, the trade-off was the life-of-mine, which was shortened significantly (Githiria & Musingwini, 2018).

2.2.1 Grade tonnage relationship

Based on the orebody characteristics, resource estimation is obtained from exploration drilling and sampling in mining operations. Resource estimation is modelled and grade distribution, ore tonnage and average grades are obtained. A three-dimensional geological block model is created, and from the data, the grade-tonnage relationship is captured in a grade-tonnage curve (Birch, 2019). Once the grade-tonnage curve or grade-tonnage table is obtained, selecting a cut-off grade below which no ore is mined will determine the ultimate size of the orebody available for mining. The relationship shows that as the cut-off grade is increased, the average grade of the material above cut-off increases and the portion of the orebody mined decreases (Yi & Sturgul, 1987).

The grade-tonnage curve (GTC) generated from the geological data best demonstrates the grade-tonnage relationship (refer to Figure 2.1). It is important to note that the curve expresses the cut-off grade in terms of centimetre-grammes per tonne (cm.g/t). The cm.g/t concept is important as it demonstrates the amount of the mineral contained within the mining width; the mining width in this example (Figure 2.1) is 100cm.

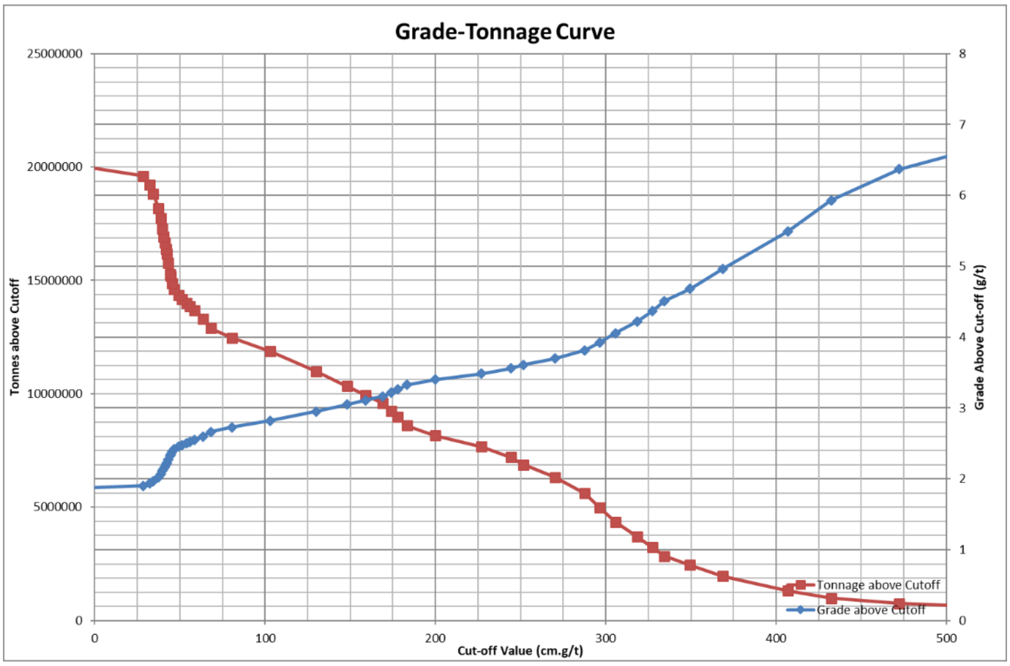

Figure 2.1: Grade-tonnage curve for UK9a Reef - Modder East Operation

2.2.2 Breakeven cut-off grade approach

The simplest form of the cut-off grade is the breakeven or marginal cut-off grade. Cut-off grades are based on the economics of extracting and processing ore to the final saleable product. The breakeven cut-off grade is the grade where the cost of mining, processing and selling the final product equals the value obtained for the product. Expressed differently, the breakeven grade is that grade where the cost of mining and processing a tonne of ore is equal to the revenue obtained from a tonne of ore. This can be demonstrated in Equation 1 (Rendu, 2014):

$$xc = \frac{M+P+O}{[r.(V-R)]} \qquad .... (1)$$

where: xc = Cut-off grade (grammes of product per tonne of ore),

M = Mining cost per tonne of ore to be processed,

P = Processing cost per tonne,

O = Overhead cost per tonne to be processed,

r = The portion of valuable material recovered from the ore,

V = Value or selling price of a unit of the product, and

R = Refining and selling cost per unit of the product.

The breakeven cut-off grade ignores the grade-tonnage distribution and production/processing constraints as well as the variation in economics around these constraints (Githiria & Musingwini, 2018). It also excludes any capital or sustaining capital; sustaining capital would be required to open new ground through development.

The breakeven cut-off grade approach leads to a “negatively geared resource” as a result of some inherent assumptions and errors contained in the breakeven cut-off grade calculation (Poniewierski, 2016). A negatively geared resource occurs when a portion of the reserve has a negative value and are in effect subsidised by more profitable portions. The implication is that value is lost if these negatively geared reserves are included, raising the question as to whether these negatively geared reserves should have been included in the first place. It is clear that there is not a homogenous distribution of material throughout the orebody, and some of the reserve blocks will be of marginal value. A small error in any of the components of the breakeven cut-off grade calculation could result in these marginal reserves being considered sub-economical (Poniewierski, 2016). Eliminating these portions would cause a reduction in the amount of ore, effectively reducing the life of mine. In an underground mining environment, it is more difficult to exclude ore that is considered negatively geared. A degree of uncertainty in grade values of underground deposits remains, and it is only when mining the orebody that the real grade is determined. Statistically, there are two types of errors, namely, type I and type II errors. A type I error occurs when a hypothesis that should have been rejected is accepted, while a type II error occurs when a hypothesis is rejected when it should have been accepted. Concerning an ore deposit, type I errors occur when a block is classified as ore, but when it is mined, the values are lower than expected. Type II errors occur when the material is classified as being below cut-off grade and is regarded as waste when in effect it is above cut-off grade, not mined and the value of this material is lost (Birch, 2017).

2.2.3 Heuristic cut-off grade approach

The Webster Dictionary (2020) defines heuristic as: "of or relating to exploratory problem-solving techniques that utilise self-educating techniques (such as the evaluation of feedback) to improve performance" (Merriam-Webster Inc., 2020). Heuristics techniques are used to make quick decisions using complex data. The outcomes of heuristic approaches are not necessarily optimal (Chen, 2019).

Heuristic cut-off grade models are value optimisation models that use higher cut-off grades at the beginning of the operation and lower or breakeven cut-off grade models during steady-state and towards the end of the life of mine. The heuristic cut-off grade approach would partially address the cut-off grade dilemma by improving recovery of capital in the early years and then optimising for life in the later years of the life of mine (Githiria & Musingwini, 2018). It has also been suggested that cut-off grades should be increased in times when the commodity prices are high and reduced when they are lower. Whatever the motivation, the approach suggests that there should not be an overall cut-off grade policy for the entire life-of-mine (Rendu, 2014). A criticism of the heuristic approach is that only direct operating costs are considered in its calculation, ignoring capacity constraints and opportunity costs; a similar criticism of the breakeven approach (Githiria & Musingwini, 2018).

Conversely, there are many proponents of the heuristic approach, each with a different method of determining the cut-off grade at various stages. Most often, computer programs are used to establish the cut-off grade at the different stages. One such study uses an iterative approach that develops an optimisation factor (Bascetin & Nieto, 2007). The optimisation factor effectively increases the cut-off grade in the early stages of the operation and through the iterative process, taking into account the mined ore, reduces the cut-off grade each year until the end of the life of mine (Bascetin & Nieto, 2007). Another body of work suggested the use of a mixed-integer programming model to establish which blocks and in which sequence blocks would be mined. In this model, the optimiser determines the best cut-off grade for each area to maximise the NPV of the operation (Martinelli et al, 2019).

There are manual iterations that can be employed in the heuristic approach, namely, the recovery of the cost of initial capital over a predetermined timeframe, for example, the first five years. By adding this factor to the traditional breakeven cut-off grade calculation, the cut-off grade will be higher in the first five years of the operation and then reduces after that. This would pose a subjective view and would not necessarily optimise the value of the operation. Computerised mine planning models do this more efficiently such as those of Bascetin and Nieto (2007), and Martinelli et al (2019) mentioned above.

To demonstrate the manual calculation of the heuristic cut-off grade approach, Githiria and Musingwini (2018) used the following example of the calculation. An operation with an estimated life of mine of 18 years decides that it needs a minimum profit per tonne for the first five years for it to pay back the initial investment. It has also decided that it will include depreciation of equipment to provide for replacement after the initial 10 years. Accordingly, there are three stages in this example: Stage 1 (years 1 – 5), which accounts for the initial profit required and the first five years of depreciation; Stage 2 (years 6 – 10), which excludes the profit loaded for Stage 1, but includes the second period of depreciation; Stage 3 (years 11 – 18), which excludes both the profit and the depreciation of the previous stages. A cut-off grade is calculated for each of the three stages as follows:

Stage 1:

For years 1 to 5 of an operation, where it is important to pay back the initial investment, a profit factor is introduced. In addition to the profit factor, the authors have also incorporated the depreciation of the initial capital equipment over 10 years to account for replacement (Githiria & Musingwini, 2018). The formula for stage 1 is shown in Equation 2 as follows:

$$hc = \frac{M+O+pt+d}{[r.(V-R)]} \qquad \text{.......... (2)}$$

where: *hc* = Heuristic cut-off grade (grammes of product per tonne of ore),

M = Milling cost per tonne of ore to be processed,

O = Overhead cost per tonne to be processed,

pt = minimum profit per tonne required,

d = depreciation of initial capital equipment,

r = The portion of valuable material recovered from the ore,

V = Value or selling price of a unit of the product, and

R = Refining and selling cost per unit of the product.

Stage 2:

The next period from year 6 to 10 excludes the profit factor *pt* but still includes the second 5 years of depreciation for the capital equipment. The cut-off grade for stage 2 is calculated using Equation 3:

$$hc = \frac{M+O+d}{[r.(V-R)]} \qquad \text{.......... (3)}$$

Stage 3:

In the final stage, depreciation is also ignored, and the formula becomes similar to the breakeven cut-off grade formula as calculated in Equation 4:

$$hc = \frac{M+O}{[r.(V-R)]} \qquad \text{.......... (4)}$$

The inputs into the formula in Equation 4 are subjective and depend on the particular strategy of the operation concerned. The computer model removes the subjectivity but do not enhance the initial years with a minimum profit expectation. The computer models demonstrated in the paper employed a set of algorithms managing varying cut-off grades and the sequence of extraction while others managed the processing flows leading to optimal cut-off grades throughout the life of mine (Githiria & Musingwini, 2018). The cut-off grade varies each year, being higher at the beginning of the life of mine and lower towards the end (Githiria & Musingwini, 2018). Heuristic cut-off grades calculated through computerised models, appear to deliver the optimum value of an orebody, but this inevitably leads to a reduction in the life of the mine and could also result in the sterilisation of some of the resource. A comparison presented in section 2.1.6 shows how large the effect is on the life of the mine.

2.2.4 Lane’s limiting cut-off grade approach

Lane is widely recognised as the pioneer of cut-off grade theory. His theories still form the basis of modern cut-off grade theory and policy. From the discussion on the heuristic cut-off grade approach, it emerges that Lane’s theory is also based on heuristics. The aspect that sets it apart is the introduction of capacity constraints. Lane (1988) found that cut-off grade was not only dependent on grade distribution through the orebody, but it also depended on capacities through the various extraction stage gates, namely, shaft infrastructure, tramming capabilities, rates of mining, and process constraints, to name but a few. He also postulated that by taking the capacity constraints into account, the operation should choose to process only the most profitable material. Once capacities are accounted for, the theory converges with other heuristic approaches and optimises cut-off grade through the maximisation of NPV (Lane, 1988).

Lane’s model is based on preparing a cut-off grade for each of the three stages in the mining/marketing process. Each stage has its associated costs and limitations and should be accounted for separately rather than as part of a global cost of mining. Lane (1988) defined the three stages and the capacities associated with them as mine limiting; treatment-limiting; and market limiting. Each stage would also have sub-stages with associated capacity constraints.

The mine limiting stage is defined most often as the development or mining rate. Mining capacity can be considered as a function of the availabilities of the basic mining elements such as access, face-length, labour and infrastructure (Minnitt, 2004). The mining limiting grade is calculated using Equation 5 (Lane, 1988):

$$gm = \frac{h}{((p-k)y)} \qquad \text{.......... } (5)$$

where: gm = mining limiting grade (grammes of product per tonne of ore),

h = treatment variable cost,

p = price of a unit of product,

k = marketing costs per unit of product sold, and

y = recovery of product from ore as a percentage.

Hence according to Lane (1988): "mineralised material should be classified as ore for as long as its implicit value exceeds the cost of further processing" (Lane, 1988). It should be noted that time and mining cost are irrelevant in the mine limiting stage. Similarly, if a mine is limited by capacity, then it should be operated tactically rather than strategically (Minnitt, 2004).

Treatment constraint is a function of design and ore handling. Ore handling would include tramming from the mine to the processing plant. There is a design capacity inherent in the processing plant, which is designed to process a certain volume of ore through its various stages. Also, there is a time factor now associated with capacity and cost figures. The time factor represents the opportunity cost of processing ore later in the mine's life. The opportunity cost is investor-related in that it is the cost to the investor of having tied up their investment in the present operation (Minnitt, 2004). The process limiting grade is calculated using Equation 6 (Lane, 1988):

$$gh = \frac{(h+\frac{f+F}{H})}{((p-k)y)} \qquad \text{.......... (6)}$$

where: gh = treatment-limiting grade (grammes of product per tonne of ore),

h = treatment variable cost,

f = fixed cost per annum,

F = opportunity cost,

H = treatment capacity,

p = price of a unit of product,

k = marketing costs per unit of product or product sold, and

y = recovery of product from ore as a percentage.

Equation 6 shows that the cut-off grade declines through time; the older the operation gets, the smaller *F* will be (Minnitt, 2004).

The marketing constraint relates to refining capacities and exclusive sales contracts. According to Minnit (2004), this would not pose a restriction on precious metals in the short to medium term but has the potential to be limiting in other commodity markets. Lane (1988) has, for this reason, identified the potential for market limiting. The market capacity is delineated by k units per year demonstrated by Equation 7 (Lane, 1988):

$$gk = \frac{h}{(p-k)y-y(f+F)/K} \qquad \text{.......... (7)}$$

where: gk = market limiting grade (grammes of product per tonne of ore),

h = treatment variable cost,

p = price of a unit of product,

k = marketing costs per unit of product sold,

y = recovery of product from ore as a percentage,

f = fixed cost per annum,

F = opportunity cost, and

K = marketing capacity per annum.

2.2.5 Cut-off grade and net present value

Irrespective of the approach used to calculate cut-off grades, net present value (NPV) is the common optimisation criterion. When declaring a reserve or preparing a life-of-mine plan, a detailed cash flow by year is developed. The detailed cash flow will consider fluctuations in technical factors such as geology, and economic factors such as costs and price. The cash flows are then discounted back to the present value using a discount rate based on the operation's cost of capital (Minnitt, 2004).

The optimal cut-off grade is established by the NPV. The optimal cut-off grade is the one that is deemed to yield the highest value. Lane established that value was a function of both the size of the remaining reserve (S) and the rate of extraction (q). Hence, the larger the rate of extraction, the shorter the life of mine (T) will be (Minnitt, 2004). Lane (1988) NPV algorithm is stated in Equation 8 as:

$$NPV = P/T\,[((1+d)T - 1)/(d(1+d)T)] \qquad \text{.......... (8)}$$

where: P = total cash flow using Equation 9 below,

T = mine life in years,

d = discount rate.

$$P = (s-r)[(g)(y)(Qc)] - (m)(Qm) - (c)(Qc) - (f)(T) \qquad \text{.......... (9)}$$

where: P = total cash flow,

s = commodity price,

r = refining, transportation, of the saleable product,

g = average grade (g/ton) of material sent for treatment,

y = recovery of product from ore as a percentage,

Qc = quantity of ore sent for treatment,

m = mining variable cost,

Qm = quantity of material mined (both ore and waste),

c = treatment variable cost,

f = fixed cost per annum, and

T = mine life in years (Stewart, 2014).

2.2.6 Cut-off grade dilemma

Most research studies proposed to maximise value, thereby increasing the cut-off grade and reducing the life of the operation. While this reduction in the life of mine may not be of concern to investors, it would be a concern for other stakeholders, creating a dilemma. The dilemma is particularly relevant in South Africa where the "social responsibility" stakeholders, namely, labour and communities, are highly dependent on a long life of mine. Profit-driven stakeholders, such as management and shareholders, would opt to optimise on NPV, given that mining rights are highly dependent on the social license to operate (Mugwagwa, 2017). Governments also have an interest in ensuring that resources are not wasted to maximise the revenue obtained from it. The real options theory presented by Thomson and Barr (2014) used real prices to predict the future value in an attempt to appease all the stakeholders .

The decision to increase the cut-off grade, thereby decreasing the life of the project has an opportunity cost associated with the mineral left behind. The choice of mining method exacerbates this. In a surface mining environment, the scale mining can be changed to take advantage of improved market conditions; cut-off grades reduced and NPV increased. Conversely, if market conditions weaken, cut-off grades can be increased to maintain the NPV. On the contrary, in underground mining, the cut-off grade must increase to take advantage of higher market prices, especially if the NPV is to be maintained. The paradox exists because in open pit mining all the material is mined, but in underground mining, it is possible to mine more selectively (Yi & Sturgul, 1987).

In a comparative study done by Githria and Musingwini (2018), they compared the cut-off grades using the same mining and economic data but using the different approaches mentioned above namely, the breakeven approach, the heuristic approach and Lane's approach. The approach that yielded the optimal NPV of $435.52 million was the heuristic approach. The breakeven approach yielded the lowest NPV of $218.50 million. The life of mine using the heuristic approach was 10 years and the breakeven approach produced a life of mine of 35 years. In the two extremes, the cut-off grade that yielded the best NPV mined only 2,121,000 ounces, while the breakeven model produced 3,365,900 ounces. Therefore, approximately 1,2 million ounces were left behind over the 25 years (Githiria & Musingwini, 2018).

In essence, it depends on the outlook of the prices for the commodity being mined, demonstrating the trade-off between current prices and prices in future. If prices in the future are expected to decline, then the strategy should be to extract more at a higher value and therefore increase cut-off grades (Thomson & Barr, 2014).

2.3 Monte Carlo Simulation

The Monte Carlo simulation, also known as multiple probability simulations, is used to model the probability of different outcomes. Variables are stipulated and changed in several iterations to produce a range of possible outcomes. Mining is an inherently risky venture, not only from a health and safety point of view but also due to the economic and geological uncertainties associated with it. There are many variables to consider, including commodity prices, costs, exchange rates, geological factors and production rates. It would be preferable to assess the risks based on the uncertainties before the venture is undertaken, or at suitable intervals after that. Due to advances in computer technologies, simulation processes that took hours to complete in the 1970s can now be completed in a matter of seconds (Heuberger, 2005). Mining models associated with feasibility studies (FS) and even reserve models for life-of-mine planning can be evaluated using Monte Carlo simulation software. The model must be detailed with all known factors affecting the operation accounted for as well as a clear stipulation of any included assumptions.

2.4 Cost and Cost Analysis

Operating costs are an essential part of the mining industry. Several measures and indices are quoted and misquoted based on costs and cost elements. Economic uncertainty is a critical factor associated with project evaluation. Economic factors include (but are not limited to) market prices and, very importantly, the associated cost of extracting and processing the commodity. Costs are as crucial in the calculation of value as revenue is (Dehghani & Ataee-pour, 2012). The importance of costs relates not only to the costs themselves but how these costs are associated with the various mining activities and how they are recorded. Management accounting has, for a long time, been underplayed in the mining industry. Errors in cost estimation are relatively frequent and incorrect cost allocations could mean that incorrect mining decisions are made (Poniewierski, 2016).

2.4.1 All-in sustaining costs and all-in cost metrics

One of the biggest challenges in the mining industry is to reflect costs accurately. Until 2013, almost every mining group or operation used a different classification of operating costs. Leading gold producers, through the World Gold Council (WGC), developed a more representative metric of costs to be included in their reporting, namely, AISC and AIC. The WGC has guided the cost elements reported under those two cost definitions and in so doing, they have provided consistency and transparency into the costs associated with producing one ounce of gold (Yapo & Camm, 2017).

The WGC, in collaboration with leading gold producers, defined the two standard reporting metrics, namely, AISC and AIC. The purpose was to provide stakeholders with “comparable metrics that reflect, as close as possible, the full cost of producing and selling an ounce of gold” (Harris, 2013). The introduction of AISC and AIC ensures that all mining companies report consistently on the costs and their respective associated cost elements. The grouping and association of cost elements (attached as Annexure A) indicate that there is a sub-classification within AISC called “Adjusted Operating Costs” (AOC). The various cost elements contained in AOC resemble what most companies term “operating costs” or “cash operating costs”. The WGC guidelines define sustaining costs as those costs necessary to maintain production and carry out the current production plan (Yapo & Camm, 2017).

For some time, traditional cost reporting only focussed on mining and processing costs to produce gold, ignoring critical elements such as sustaining capital (Yapo & Camm, 2017). Similarly, the traditional calculations of cut-off grade ignore the role of sustaining capital in the cut-off grade calculation (Poniewierski, 2016). With the introduction of the new cost metrics came the realisation that the traditional “Cash Costs” reported only represented the “visible” part of the cost iceberg. The new metrics are built on the traditional cost reporting framework employed in the mining industry (refer to Figure 2.2).

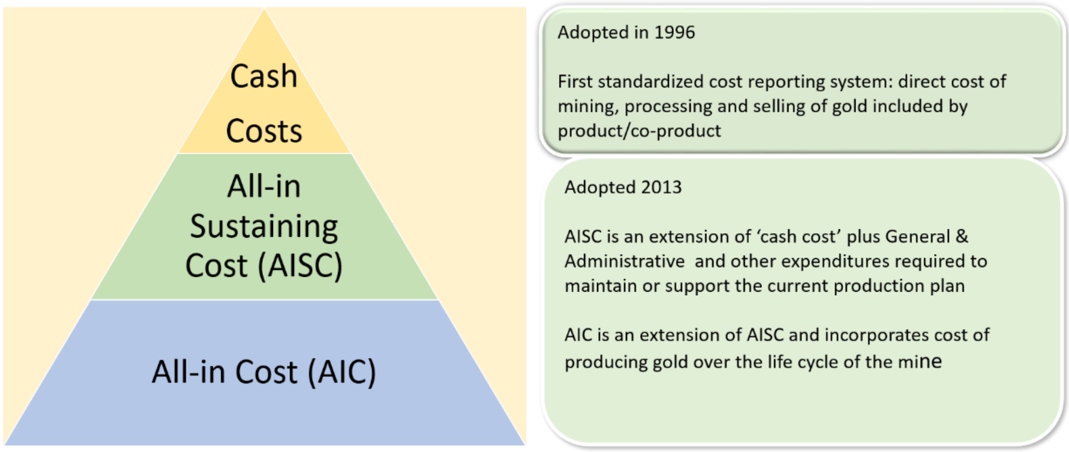

Figure 2.2: Iceberg of gold mine costs (Yapo & Camm, 2017)

2.4.2 Activity-based cost system for mining

ABC system assign costs based on cost drivers rather than the volume of output. The system ensures that costs relating directly to the activity are captured together so that the activity can be effectively managed (Noreen, 1991).

Lind (2001) found that ABC was more effective in costing underground coal mining systems than other traditional approaches. The ABC method can be applied to all mining-related activities and, regarding this research, it is essential to delineating costs per mining method. Lind (2001) also found that ABC produced better investment decisions than when any other costing system was used. Traditional costing methods assign costs based on the product rather than the activity. Also, overhead costs or fixed costs are generally assigned by the volume driver and often cross-subsidise other products that have less volume. It was further found that the traditional costing approach used in coal mining was Process Costing (PC). In this approach, costs are accumulated by process (Lind, 2001). One of the shortcomings of the PC approach is that processes are costed rather than products or groups of products, making it challenging to account for multiple products. Traditional systems base the allocation of overheads and indirect expenses on labour or machine hours. In contrast, ABC segregates the overheads and indirect expenses by activity, allowing for the cost to be assigned based on the drivers associated with those activities (Cooper & Kaplan, 1991).

Within ABC and an integral part of it, is the process of Standard Costing (SC). SC refers to the costing of a process, activity or product to ascertain the cost of producing a single unit of the product or activity. The standard will include all direct costs associated with the production of one unit. These costs would include direct labour, direct materials and direct overheads. These standards are used for budgeting purposes, and actual costs are then compared against these to determine variances. Both the actual and the standard costs are reported in the accounting system, which assists with the control of costs by addressing the variances (Swamidass, 2000).

A weakness of ABC is found in the collection of accurate data per activity, which, similarly, also has costs associated with it. The system is difficult to maintain and requires constant review of cost allocations. Potential errors in the system include other errors such as the basket of costs or bill of materials that make up an activity, the cost driver of the activity and the allocation of costs. These errors are not unique to ABC, as all costing systems have similar errors and are as reliable as the information fed into them (Labro, 2019).

A review of existing literature revealed that very little is written about ABC in mining. Lind(2001) demonstrated the difference between the costing approaches in coal mining. He applied the PC and ABC approaches to two different mining methods, namely, mining using a continuous miner and longwall mining. When comparing the two different costing techniques for the two approaches, findings indicated that the PC approach overstated the costs, leading to a reduction in NPV. An evaluation of the NPV for the respective approaches indicated a significant difference in value (Lind, 2001).

2.5 Summary of Chapter 2

The common aspect of Chapter 2, the reviewed literature, is the influence of costs on the cut-off grade and the resultant value of the operation. While researchers hold differing views on the role of cost (most notably which costs should be included), there was a consensus that costs play a significant role in the formulation of cut-off grade approaches. The allocation of costs and ensuring that all costs are included in the calculation is highlighted by Poniewierski (2016), who demonstrated the effect of errors in the calculation of elements of the cut-off grade that could lead to a “negatively geared ore reserve.”

The use of an optimal costing system was also highlighted in the literature review (Lind, 2001). Therefore, this research is built on the ABC system, which increased the reliability of the outcome.

The many approaches to cut-off grade calculation and the effect that they have on the life of the project highlighted a vital focus of this research. This research has demonstrated that the NPV of the operation could be improved through a reduction in cut-off grade by the various mining methods, thus increasing the mineral reserve. Capacity constraints mentioned by Lane (1988) will be included through the limiting of ore trammed and processed through the metallurgical plant.

Due to the different mining methods used at the Modder East Operation and the various costs associated with them, the optimal mining approach is examined by using the Monte Carlo simulation. The simulation is expected to optimise based on NPV, but which may not be the maximisation of the lowest cost method due to the volume-grade relationship.

3 OPERATIONAL OVERVIEW AND CUT-OFF GRADE DETERMINATION

3.1 Overview of Chapter 3

This study is based on data obtained from Gold One Limited's New Kleinfontein Goldmine (NKGM). Chapter 3 gives insight into the operation, its geology and infrastructure. The mining methods proposed by the BFS are also highlighted giving insight into the impact of changes in mining methods on NPV and the life of the operation.

The chapter gives insight into the traditional approach to cut-off grade calculation at the operation as well as the economic and recovery factors used in the calculation. The cost factors associated with the calculation are highlighted. The GTCs for the various reefs are then demonstrated obtaining the tonnages and average grades from the cut-off grade calculation for each reef type. With all the data in place, the NPV is calculated for the operation.

3.2 Operational Overview

The operation that was used as a case study is (NKGM), otherwise known as Gold One: Modder East Operations. The operation is the northernmost operational underground mine in the East Rand Goldfield and is situated directly north of the town of Springs, 30km east of Johannesburg and north-northeast and directly adjacent to the historic Consolidated Modderfontein Mine (refer to Figure 3.1).

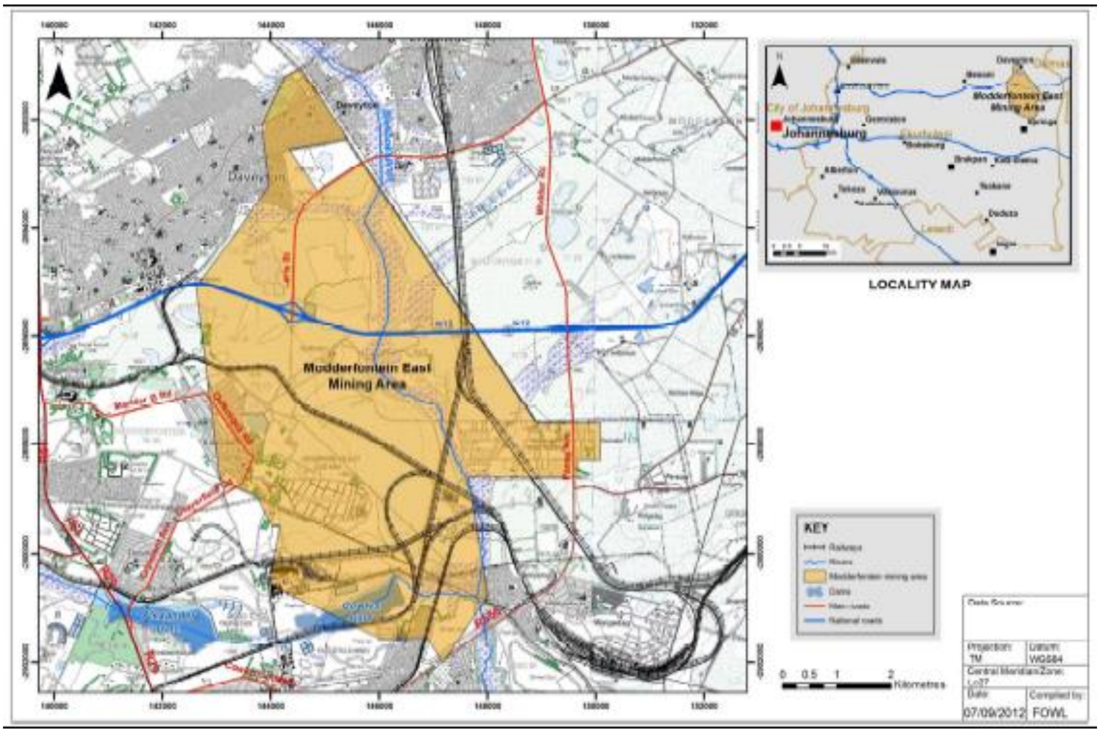

Figure 3.1: Modder East Operations Location Map (SRK Consulting (South Africa) (Pty) Ltd. , 2013)

The operation comprises a shallow underground mine extracting the Witwatersrand Basin placer gold deposits which are stratiform, gold enriched reefs within the East Rand Goldfield and the younger overlying Transvaal Supergroup. The target Witwatersrand Supergroup sequence (see Figure 3.2) is the UK9A horizon within the Kimberley Reefs, as well as the Black Reef formation, also known as the Buckshot Pyrite Leader Zone (BPLZ), a basal unit in the overlying Transvaal Supergroup (Ogilvy, 2019).

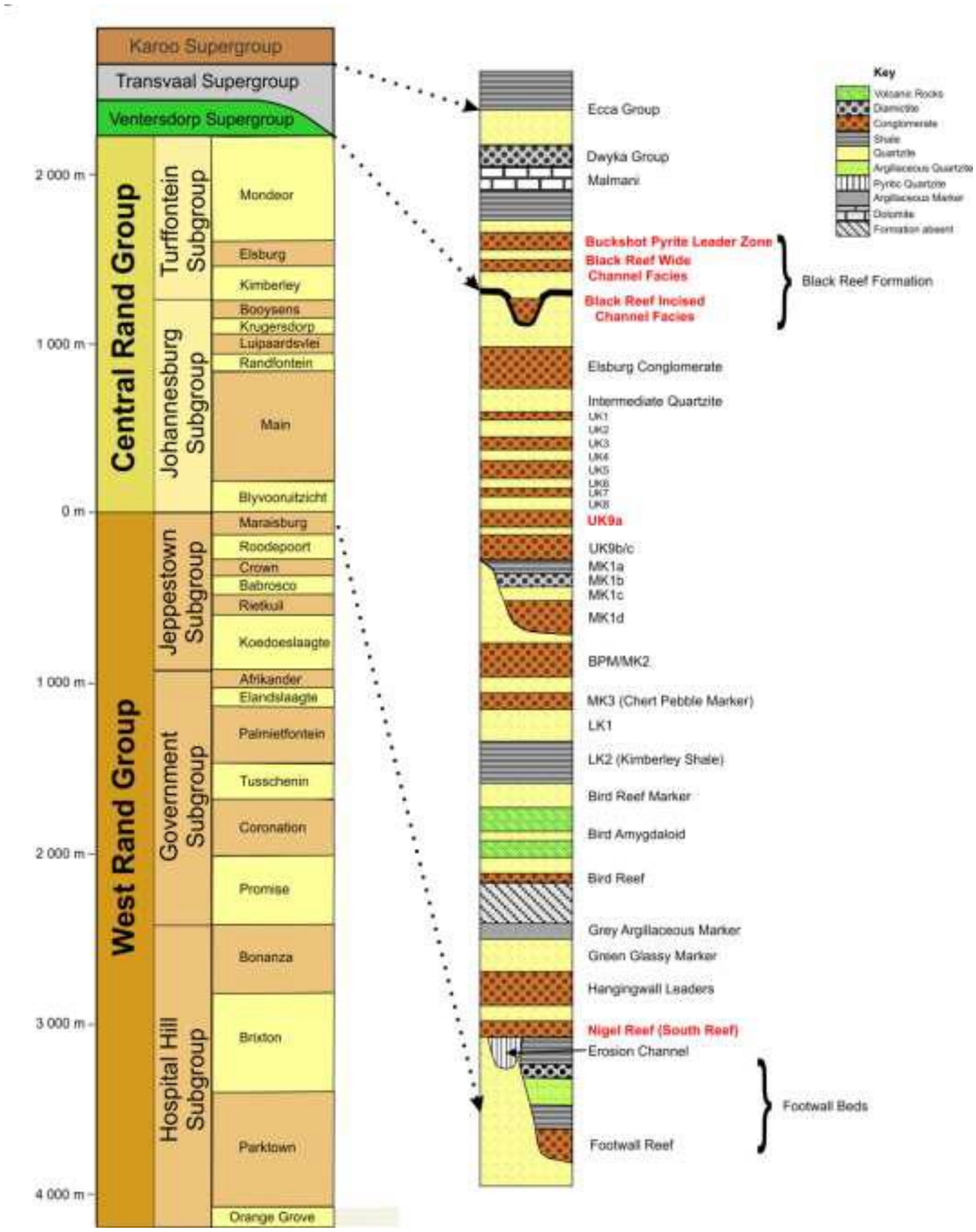

Figure 3.2: Schematic stratigraphic column for the East Rand Goldfields (Ogilvy, 2019)

The BPLZ is approximately 300 metres below the surface (mbs) and the inclination varies between flat and shallow dipping to 4 degrees. The reef is relatively undisturbed with minimal geological structures encountered during exploration. The gold is concentrated along a depositional shoreline on the northern edge of the orebody with grades dropping off as the distance increases from this shoreline in a southerly direction. The BPLZ is underlain by Blanket Facies and Channel Facies (Pooley , 2006).

The UK9A unconformably underlies the Black Reef Formation and sub-crops against the base of the Black Reef. The sub-crop is approximately 300 mbs and dips at approximately 6 to 12 degrees to the south. The UK9A has been evaluated to a depth of 530 mbs on the property (Pooley, 2006).

The underground workings are serviced by a trackless decline from the surface accessing the footwall of both reef horizons. The decline is used as a roadway for vehicles to transport rock and materials in and out of the mine. Additionally, there is a vertical shaft that is used to transport personnel in and out of the mine and to provide additional ventilation.

In terms of the original FS, the mining of the two reef horizons would be undertaken by employing conventional narrow reef breast mining as is commonly practised in South African gold mines. Support of the development and stopes is mainly based on a rigid pillar system, which protects the surface from any impact from mining. Due to changes in mining technology, it is now possible to mine the pillars and replace this support with cement grout packs.

To support mining operations, various transport systems are utilised:

- Personnel are transported in and out of the mine in conveyances that are hoisted in the vertical shaft by a single drum hoist sited on the surface. Additionally, people can travel through a decline in suitable vehicles when required.
- Materials are transported into the mine using materials transport vehicles, which operate in the decline.
- Rock is transported out of the mine in articulated dump trucks operating between the underground silos and the surface stockpile.

Mined ore is delivered to a tipping point on the surface from where it is delivered to the metallurgical plant by belt conveyors. The original processing capacity of the plant was 90,000 tonnes per month. The processing capacity was increased during 2018 to 120,000 tonnes per month to reduce the impact of falling grades transforming the operation into a high volume, low-grade operation (Pooley , 2006).

The original BFS suggested that the deposit should be mined conventionally. The original mine design criteria include (Pooley, 2006):

- 1.2m stoping widths,
- panel widths of 26m,
- pillar sizes of 6m length by 4m width,
- advance strike gully (ASG) of 1.5m wide and 2m high,
- remaining 23.5m of the panel mined at stope width of 1.0m,
- tramming width of 1.08m, which is considered low,
- face advance per stoping team 14m/month,
- raise development 40m/month,
- extraction recovery of 89.7%, and
- safety factors of 1.9 for the BPLZ and 2.3 for the UK9A.

3.1 Cut-off Grade Determination at Modder East

The calculation of the cut-off grade is based on the traditional approach used by the Mineral Resource Management department at Modder East from inception to determine the cut-off grade per reef type. For continuity, this calculation will be used both in the basic determination and the determination of cut-off grade per mining method. The calculation includes sustaining capital and the milling limiting factor was intrinsically included.

To determine the cut-off grades, various economic data need to be analysed. This analysis will include a view on the gold price for the coming period and on the expected operating and sustainable capital required to mine the orebody.

Additionally, various geological parameters need to be considered, including the recovery and dilution factors.

3.2.1 Economic and recovery factors used in cut-off grade calculation

At Modder East, the budgeted gold price and the exchange rate is communicated via the corporate strategy in the planning and budget stages. Also, the mine call factor, plant recovery factor and dilution were determined based on the geological characteristics of the reef types and orebody. These characteristics determine the modifying factors contained in Table 1:

Table 1: Cut-off Grade modifying factors

		Reef Type		
		Black Reef		
Parameter	Unit	BPLZ	Basal Reef	UK9A
Gold Price	US$/oz	1 230	1 230	1 230
Gold Price	R/g	560	560	560
Exchange Rate	R:US$	14.15	14.15	14.15
Mine Call Factor	%	93	93	93
Plant Recovery Factor	%	94	94	95
Dilution	%	3	3	5

The WGC conventions are used at the mine to establish the classification of the various costs. These classifications are AOC and AISC, where AOC form the basis of the AISC and AIC. These costs can be equated to the Cash Operating Costs quoted in financial results of most precious metals companies (Harris, 2013).

The Modder East operation has a plan to treat 90,000 tonnes of ore per month through its metallurgical plant, equating to 1 080 000 tonnes treated per annum. Total tonnes treated is applied to the total costs (fixed and variable) to arrive at a total cost per tonne treated. The milling capacity is also the limiting factor used for the cut-off grade calculation.

The budgeted AOC for the operation is classified into fixed and variable costs. Variable costs are determined on a Rand per tonne treated (R/t) basis and then multiplied by the number of tonnes treated, after which the fixed costs are added. These are then analysed to produce a total cost per tonne treated. The detailed determination of fixed and variable cost associated with the mining operation is established (see Annexure B) and are summarised in Table 2.

Table 2: Current Cost Breakdown (Modder East, 2019)

Direct Costs	Proportion	Tonnage	Cost (R'000)	R/t
Conventional Stoping	96%	1 034 053	259 867	251
Trackless Reef Development	3%	32 942	5 291	161
Conventional Reef Development	1%	13 006	10 222	786
Total Direct Costs	100%	1 080 000	275 380	255
Indirect Production Costs				
Mining (Waste development & other)			73 190	68
Engineering and logistics			389 276	360
Total Indirect Costs			462 465	428
Treatment Costs		1 080 000	136 428	126
Overheads				
Management			23 955	22
Mining (Fixed costs)			200 751	186
Engineering (Fixed costs)			186 081	172
SHEQ			53 416	49
Mineral Resource Management			32 117	30
Metallurgical Plant (Fixed costs)			69 152	64
Human Resources			91 395	85
Finance & Administration			57 416	53
Corporate Costs			2 550	2
Capitalized Mine Development			(262 649)	(243)
Total Overhead Costs			454 184	421
Total Adjusted Operating Costs			**1 328 457**	**1 230**
Sustaining capex			**359 340**	**333**

3.2.2 Cut-off grade calculation at Modder East Operation

Following the establishment of the cost classifications, the breakeven cut-off grade was calculated using all the economic and recovery data (Section 3.1.1). The breakeven point is the point where the revenue is equal to the costs of producing the revenue. The breakeven cut-off grade would then be the *in-situ* grade before any mining takes place.

At breakeven, revenue equals cost, therefore:

Volume x grade x recovery x price = (variable costs x tonnes + fixed costs + sustaining capital expenditure). Utilising the formula for the breakeven cut-off grade (Equation 1 mentioned in Chapter 2), the cut-off grade is calculated as follows (Rendu, 2014):

$$xc = \frac{M+P+O+C}{[r.(V-R)]}$$

Where: xc = Cut-off grade (grammes of product per tonne of ore),

M = Mining cost per tonne of ore to be processed,

P = Processing cost per tonne,

O = Overhead cost per tonne to be processed,

C = Sustaining capital per tonne processed,

r = The portion of valuable material recovered from the ore,

V = Value or selling price of a unit of the product, and

R = Refining and selling cost per unit of the product.

The recovery factor (*r*) was calculated per reef type based on the parameters for mine call factor, plant recovery factor and dilution to arrive at the overall recovery factor for the respective reef types (Table 3).

Table 3: Overall Recovery Calculation

Factors	Unit	Reef Type		
		Black Reef	Kimberley Reef	Basal Reef
Mine Call Factor	MCF	93%	93%	93%
Plant Recovery Factor	PRF	94%	95%	94%
Dilution	Dil	3%	5%	3%
Overall recovery[1]	%	85%	84%	85%

1 Overall recovery = MCF x PRF x (100-Dil)

For the calculation, the direct costs and indirect costs were considered mining costs (M). Using the information from Table 2 and Table 3, the breakeven cut-off grade per reef type was calculated using Equation 1 as follows:

(1) BPLZ and Basal Reef:

$$xc = \frac{\text{R255/t+R428/t+R126+R421/t+R333/t}}{[85\%\ (560\text{-}3)]}$$

$$= 3.30 \text{ g/t}$$

(2) UK9A Reef:

$$xc = \frac{\text{R255/t+R428/t+R126+R421/t+R333/t}}{[(84\%\ (560\text{-}3)]}$$

$$= 3.34 \text{ g/t}$$

An assumption included in this cut-off grade calculation is that each reef is mined exclusively and that the fixed costs applied for the total mine assume that only that reef is mined. In reality, a combination of the reefs is mined in proportion as determined by the life of mine schedule (Table 4). When the respective reefs are mined conventionally the cost of mining is the same allowing for the combination of these costs after accounting for the proportion of each reef type mined per year. Each respective reef type has a GTC (Figures 3.3, 3.4 and 3.5). The GTC shows the cut-off grade in cm.g/t. It is a product of the channel width (cm) and the contained gold (g/t). This is the grade over the reef width.

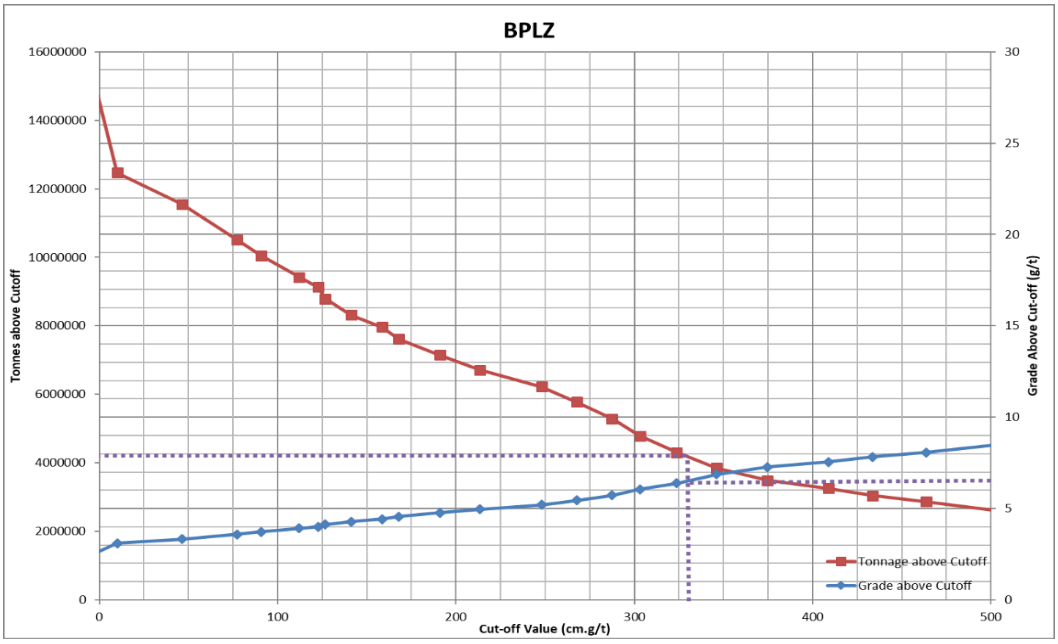

Figure 3.3: GTC for BPLZ

For BPLZ, the reef width is 100cm. From the GTC for BPLZ (Figure 3.3), it emerges that at a cut-off grade of 3.30 g/t the total tonnage above cut-off that can be mined is 4,2 million tonnes and the average grade is 6.50 g/t.

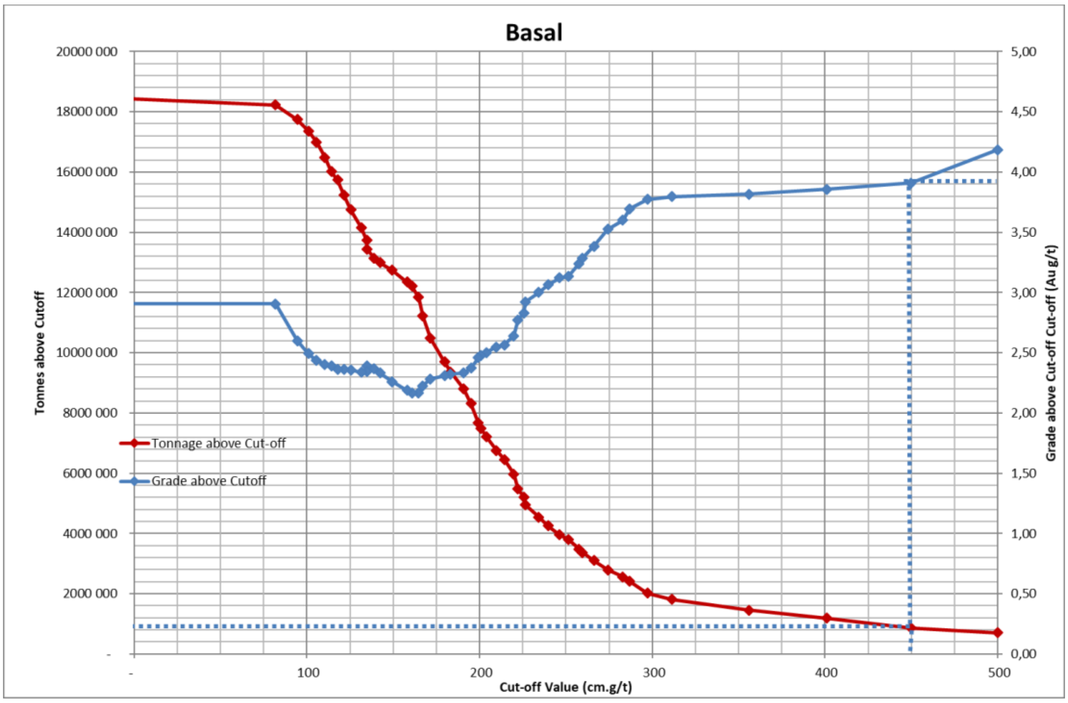

Figure 3.4: GTC – Basal

For Basal Reef (Figure 3.4), the GTC demonstrates that at a cut-off grade of 3.30 g/t, the total tonnage above cut-off that can be mined is 865,000 tonnes and the average grade is 3.91 g/t. The Basal Reef has a wider reef channel so at a cut-off grade of 3.30 g/t, the cut-off grade is 450 cm.g/t. The reef width of BPLZ is 100 cm, whereas the reef width for Basal Reef is 136cm

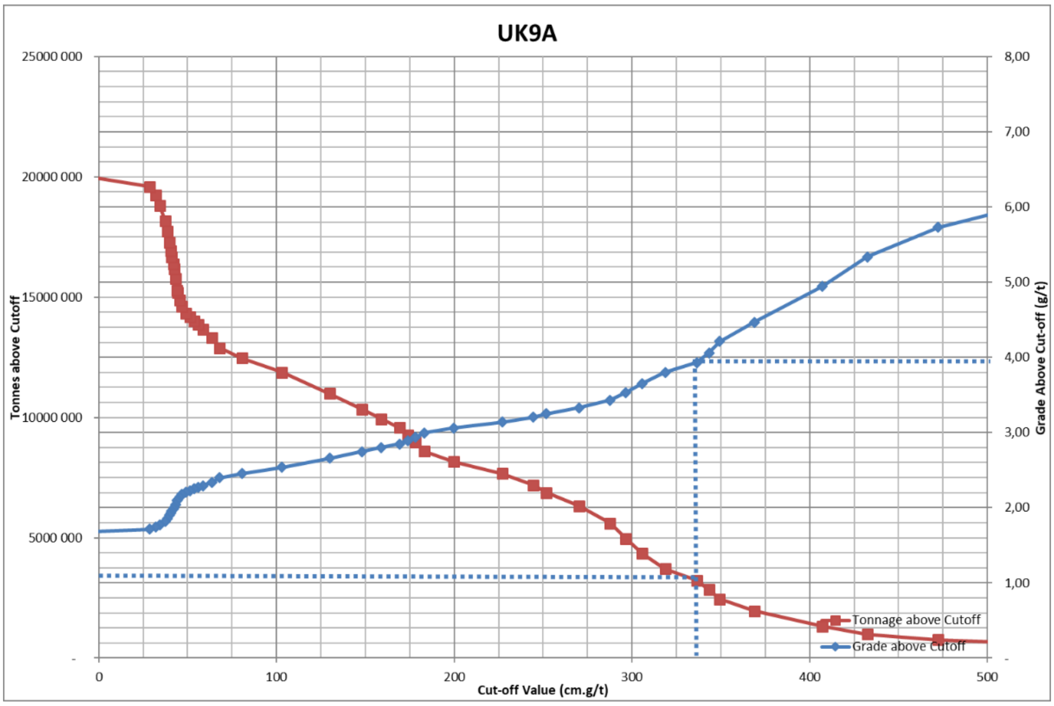

Figure 3.5: GTC - UK9A

The GTC for UK9A Reef (Figure 3.5) shows that at a cut-off grade of 3.34 g/t the total tonnage above cut-off that can be mined is 3,2 million tonnes and the average grade is 3.90 g/t.

3.3 Net Present Value Calculation

Based on the calculated cut-off grades (Section 3.1.2), the mine planner can proceed with the preparation of the life-of-mine plan to incorporate the tonnages and grades per mining block in detail. Planning and scheduling are required because not all the tonnage on all reef types will be available to be mined and development needs to be scheduled to ensure access to the various reef blocks associated with each reef type. The UK9A Reefs are located below the BPLZ and Basal Reef, as illustrated in the stratigraphic column (Figure 3.2). The mine was designed to mine the BPLZ and Basal Reef first while developing towards the UK9A Reefs. By way of illustration, the schedule (Table 4) is a simplistic life of mine model based on the proportion of each reef type mined.

Estimating the NPV based on the tonnage and grade above the cut-off allows for a cash flow model (Table 5) based on the maximum tonnage through the metallurgical plant to be established. The estimated NPV at a discount rate of 12% is R4,42 billion over the remaining life of mine of 8 years.

Table 4: Life-of-Mine Schedule per Reef Type

		2019	2020	2021	2022	2023	2024	2025	2026
Black Reef									
Tonnage Mined	*'000 t*	677	642	602	583	490	469	443	273
Grade Mined	*g/t*	6,50	6,50	6,50	6,50	6,50	6,50	6,50	6,50
Grade Recovered	*g/t*	5,53	5,53	5,53	5,53	5,53	5,53	5,53	5,53
Gold Recovered	*'000 kg*	3,7	3,6	3,3	3,2	2,7	2,6	2,4	1,5
Basal									
Tonnage Mined	*'000 t*	137	133	126	123	121	120	105	
Grade Mined	*g/t*	3,91	3,91	3,91	3,91	3,91	3,91	3,91	
Grade Recovered	*g/t*	3,32	3,32	3,32	3,32	3,32	3,32	3,32	
Gold Recovered	*'000 kg*	0,5	0,4	0,4	0,4	0,4	0,4	0,3	
Kimberley									
Tonnage Mined	*'000 t*	267	305	352	373	468	491	533	400
Grade Mined	*g/t*	3,90	3,90	3,90	3,90	3,90	3,90	3,90	3,90
Grade Recovered	*g/t*	3,28	3,28	3,28	3,28	3,28	3,28	3,28	3,28
Gold Recovered	*'000 kg*	1	1	1	1	2	2	2	1
Total Mined									
Tonnage Mined	*'000 t*	1 080	1 080	1 080	1 080	1 080	1 080	1 080	673
Grade Mined	*g/t*	5,53	5,45	5,35	5,31	5,08	5,03	4,97	4,96
Grade Recovered	*g/t*	4,69	4,62	4,54	4,50	4,30	4,26	4,20	4,19
Gold Recovered	*'000 kg*	5	5	5	5	5	5	5	3

Table 5: NPV Estimation

		2019	2020	2021	2022	2023	2024	2025	2026
Gold Price	*$/oz.*	1 230	1 230	1 230	1 230	1 230	1 230	1 230	1 230
Exchange Rate	*R/USD*	14,15	14,15	14,15	14,15	14,15	14,15	14,15	14,15
Gold Price	*R/kg.*	559 568	559 568	559 568	559 568	559 568	559 568	559 568	559 568
Revenue	***R mil***	**2 836**	**2 793**	**2 741**	**2 719**	**2 601**	**2 574**	**2 541**	**1 579**
Adjusted Operating Costs	***R mil***	**1 435**	**1 431**	**1 427**	**1 425**	**1 415**	**1 412**	**1 410**	**878**
Mining & treatement costs	*R mil*	1 326	1 326	1 326	1 326	1 326	1 326	1 326	826
Corporate costs	*R mil*	3	3	3	3	3	3	3	2
Royalty tax	*R mil*	106	103	98	96	86	84	81	50
Sustaining Capital	***R mil***	**359**	**359**	**359**	**359**	**359**	**359**	**359**	**224**
Cash Flow (Pre-tax)	***R mil***	**1 042**	**1 003**	**955**	**935**	**827**	**802**	**772**	**477**
NPV (R mil's)	***12%***	***4 421***							

3.4 Summary of Chapter 3

The various reef types and an overview of the Modder East Operation gives credence to the costs of the operation. The mine is relatively shallow in the context of gold mining in the Witwatersrand Basin. In defining the reef type, the overall recoveries were calculated to feed into the cut-off grade calculation. Mining is limited by the processing capacity of the metallurgical plant, which constrains the production per annum. This statement assumes that there are no other mining constraints such as development. In the history of the operation, the processing plant has not been impacted by the delivery of ore from underground. Coupled with this constraint, is the proportion of each reef mined, which is established based on the mine schedule of the operation. The overall cost analysis was described together with the rationale for establishing the various costs associated with the cut-off grade calculation. The WGC definitions of AISC were used for this purpose.

When calculating the cut-off grade, the breakeven cut-off grade approach was used. Once the cut-off grade was established, the overall tonnage and average grade above cut-off were calculated. A simplistic mine schedule was presented showing the life of the operation. Eventually, an NPV model was created, giving an overall NPV of the operation based on the proportion of reef type mined.

4 COST OVERVIEW

4.1 Overview of Chapter 4

Costs and costing are an integral part of any operation. Chapter 4 demonstrates the methodology used at Modder East to assimilate costs into the various cost centres which are associated with the various activities on the mine. It demonstrates how the ABC system is used at the operation with the formulation of the cost centres. In addition, the chapter shows how direct costs are identified for the use in formulation of costing standards. The standards then form the basis of cost analysis in ABC.

The weaknesses identified in Section 2.3.2 are addressed in this chapter as they pertain to the Modder East Operation. It was through the use of ABC that costs associated with the various mining methods were properly analysed leading to the reduction in the cut-off grade and an improvement of NPV of the operation.

4.2 Costing System and Structure at Modder East

The costing system used by Modder East is based on the ABC approach. Within the costing system, there is a need to apply SC to each of the mining activities. SC, is defined in Chapter 2, refer to Section 2.3.2

The structure of the chart of accounts at Modder East is multi-layered and each cost is represented by a 15-digit cost code (Figure 4.1). The first four digits represent the cost element, such as payroll, consumables, contractors (not shown in Figure 4.1). The next three digits represent the entity, in this instance, Modder East, followed by the next two digits representing the department such as mining, engineering, processing. The next digit identifies the cost centre owner, followed by two digits for the supervisors reporting to him. The last three digits represent the activity, such as trackless development and stoping. The multi-layered system allows the cost data to be collected by element for each activity.

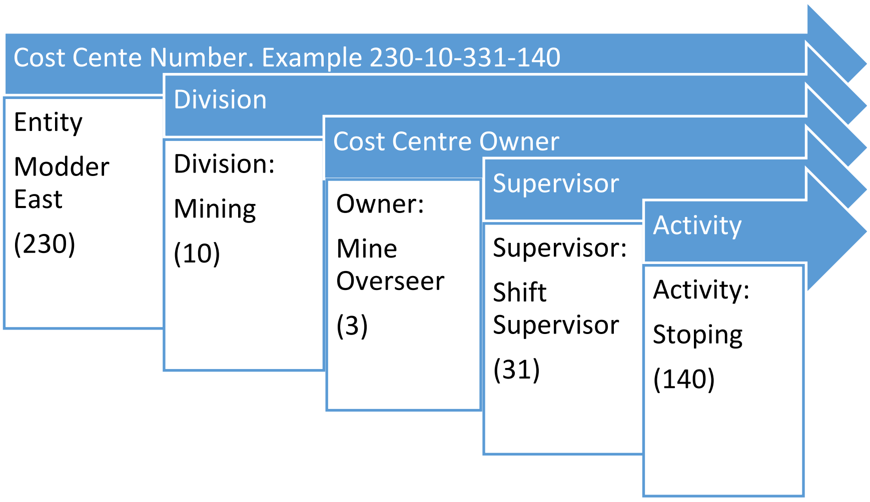

Figure 4.1: Modder East Cost Centre Configuration

There is a 4-digit cost code that precedes the cost centre number and designating the cost element, namely, payroll, consumables, consultants, contractors and services (Table 6). Each of these has further details relating to the cost. For instance, the cost element consumables will have further details, such as explosive costs, reagents and fuel. In the Modder East system, 477 element codes form part of the account number.

Table 6: Sample of Cost Element Codes (Modder East, 2009)

Cost Element Sample		
Payroll Sample		
Labour - Payroll 1	1110	Payroll 1 - Basic Pay
Labour - Payroll 1	1114	Payroll 1 - Annual Bonus
Labour - Payroll 1	1115	Payroll 1 - Production Bonus
Labour - Payroll 2	1210	Payroll 2 - Basic Pay
Labour - Payroll 2	1211	Payroll 2 - Overtime
Labour - Payroll 2	1215	Payroll 2 - Production Bonus
Labour - Payroll 3	1310	Payroll 3 - Basic Pay
Labour - Payroll 3	1311	Payroll 3 - Overtime
Labour - Payroll 3	1312	Payroll 3 - Public Holiday Pay
Labour - Payroll 3	1313	Payroll 3 - Allowances
Labour - Payroll 3	1314	Payroll 3 - Annual Bonus
Labour - Payroll 3	1315	Payroll 3 - Production Bonus
Consumables Sample		
Stores	2100	Assay consumables
Stores	2103	Air conditioners
Stores	2110	Bearings
Stores	2120	Blasting consumables
Stores	2210	Drill Steel & bits
Stores	2220	Drilling equipment - Non Hydropower
Stores	2230	Electric cabling & consumables
Stores	2240	Electric motor spares
Stores	2600	Reagents - Activated Carbon
Stores	2606	Reagents - Oxygen
Contractors, Consultants and Services Sample		
Consultants	3250	Consultants - General
Consultants	3260	Consultants - Geological Reporting
Contractors	3300	Contractors - Cover Drilling
Contractors	3310	Contractors - Other Services
Contractors	3311	Contractors - Drop Raises
Services	3664	Secretarial Services
Services	3670	Security services
Services	3680	Seismic Services

In the cost code example 2120-230-10-331-140, the cost centre represents explosives for Shift Supervisor 31 under Mine Overseer 3 for one of his stoping panels. From the exemplar, it emerges that the costing system allows each cost element to be accurately costed to each activity and sub-activity, for example, an individual stope panel.

One of the criticisms of ABC is that it needs an accurate allocation of costs for it to be meaningful and that this cost allocation is time-consuming (Labro, 2019). In the instance of Modder East, a management decision was taken that production personnel would not be responsible for accurate allocations as their primary focus should be on gold production. To ensure that costs were allocated correctly, mining cost clerks were employed to capture requisitions into the correct cost centres. These requisitions were then electronically approved by the cost centre owners based on an approval limits matrix. A weekly review of costs against budget highlighted overspending and possible misallocations.

4.3 Direct Costs and Standard Cost Analysis

With the cost model in place, it is necessary to undertake SC to cost each activity accurately. Direct costs are calculated by consulting and observing each activity to ascertain the quantity of each unit of costs applicable to each task. This information is used for benchmarking and budgeting purposes and for controlling costs (Swamidass, 2000). The parameters for the activity (for example, conventional stoping) are derived from the observations and consultation relating to conventional stoping at Modder East (Table 7). The parameters include the actual dimensions of the panel as they are currently mined. These dimensions will then dictate the drilling pattern required as well as the number of holes required to break the rock sufficiently; including the depth of the hole to achieve the required face advance. Using these parameters, a determination of how many drill bits are required per one square metre is made. The direct unit costs associated with a standard stope panel in a conventional stope (Table 7) are then calculated.

Table 7: Items used for costing in a standard stoping panel (Modder East, 2019)

Standard Panel	Unit	Standard
FACE LENGTH	m	23
FACE LENGTH excl. GULLY	m	21.20
STOPE WIDTH	m	1.20
GULLEY WIDTH	m	1.60
GULLEY HEIGHT	m	2.70
ADVANCE PER BLAST	m	0.80
M^2 PER BLAST	m^2	18.40
HOLES PER PANEL	no.	92.00
HOLES PER GULLY	no.	27.00
HOLES PER VENT HOLING	no.	8.50
TOTAL HOLES	no.	127.50
DEPTH PER HOLE	m	1.00
Bits per Blast	no.	4.11
Meters per Bit	m	31.00
Holes per Bit	no.	31.00
Ground Density	g/dm3	2.78
Ton per Square Meter	t	3.34

The costs associated with each of the major stoping activities, namely, drilling, blasting, support and cleaning are calculated using the input parameters obtained from the consultation and observation process (Table 8) and are expressed in unit costs per square metre (m^2) and per tonne mined. All costs were based on the values used for 2019. The labour costs are established based on the productivity measure of 27 m^2 per stope employee and using the wage rate per employee, including production bonuses.

Table 8: Costs per m^2 and per tonne for a standard stoping panel (Modder East, 2019)

Stores & Material	R/m^2	R/t
Stores: Blasting Consumables	**168.30**	**50.45**
Stores: Drill Steel & Bits	**45.39**	**13.61**
Stores: Hydropower	**28.26**	**8.47**
Stores: Scrapers & Winches	**55.04**	**16.50**
Stores: Support	**41.05**	**12.31**
Stores: Stores - Other	**19.32**	**5.79**
Total Stores & Material	**357.37**	**107.12**

Labour	R/m^2	R/t
@ 27m2/stope employee; therefore 0.04 emp/m2	481.00	**144.18**
TOTAL	**838.37**	**251.31**

The same process followed for the conventional stoping example is observed for each mining activity, and the direct costs associated with each activity are collated and measured in the same way. The SC for all the mining methods used at Modder East is included in Annexure C.

Indirect costs for production refer to development and other mining costs required to support mining. All engineering costs related to the operations are included under indirect costs. The consultation and observation process used to determine direct costs is also applied to activities resulting in indirect costs to ensure accurate cost allocation and calculation.

Modder East is a one shaft operation; all overhead costs support mining at the operation, meaning that there is no need to apportion costs. Costs that are considered as overhead costs are those not directly related to production or processing. Overhead costs include the administration and finance, human resources, and mineral resource management departments.

4.4 Summary of Chapter 4

To adequately account for costs per activity, it is necessary to produce SC per mining activity, which feeds into the ABC system. The structure of the costing system at Modder East was described in detail, including how each activity is accounted for within the system. The formulation of the SC was illustrated with all the mining methods standard costs shown in Annexure C. Due to the one shaft nature of the operation, all indirect and overhead costs incurred was to support mining activities. These overhead and indirect costs were described and shown in the context of the operation.

5 CUT-OFF GRADE CALCULATION PER REEF TYPE AND MINING METHOD

5.1 Overview of Chapter 5

In determining the cut-off grade per mining method, all costs associated with each mining method were established as described in Chapter 4. The same economic criteria associated with cut-off grade (as per Chapter 3) were used to ensure comparability between the conventional calculation of cut-off grade at the operation and the practice suggested by this study. Further, the direct costs per mining method were calculated and explained, as well as the detail around indirect and overhead costs.

To give credence to the research, it is crucial to explain the mining methods used, starting with the conventional stope method and then leading onto the mechanical stoping or Bord and Pillar (B&P) method. Developments in the industry have led to the potential removal of the rigid pillars left behind as support. The extraction of these pillars form part of the on-going mining at Modder East and are considered as a mining method for this study. Once the costs associated with the respective mining methods were established, the overall cut-off grade was calculated on the same basis and compared with the conventional cut-off grade calculated in Chapter 3. For comparison purposes, the calculation methodology applied is the same. The chapter concludes with an NPV calculation and a comparison between the different approaches used.

5.2 Mining Methods Used at Modder East

The 2019 life of mine plan incorporates the following mining methods:

- Trackless development both on-reef and off-reef,
- Conventional raise development,
- Conventional stoping,
- Mechanical stoping (bord and pillar), and
- Pillar mining.

The initial Modder East FS advised that mining would take place conventionally, using conventional narrow reef breast mining, which is commonly practised in South African gold mines. Support of the development and stopes is mainly based on a rigid pillar system, which protects the surface from any impact from mining. Though a trackless footwall infrastructure services the mining operations, all access to the reef horizon and all on-reef excavations are being mined conventionally (Pooley , 2006).

5.2.1 Conventional stoping and raise development

A typical stope panel layout (Figure 5.1) which consists of a stope panel is 29 metres wide in the reef dip direction and includes a 4-metre wide rigid pillar providing a maximum span of 25 metres. The strike gulleys are 1.5 metres wide and 2 metres high. The remaining 23.5 metres of stope was planned to be mined at 0.9 meters wide, excluding dilution.

The layout of a stope (Figure 5.2) consists of 16 such stope panels, 8 to the east of a central ore pass, and 8 to the west of it. Each group of 8 panels is divided in two, with four panels on either side of a central raise. The central raise is developed conventionally, 2 metres wide by 2 metres high. Each raise is equipped with a mono-winch installation to assist with the transportation of materials, and it also carries services required for the mining. One strike gully per raise is developed to hole into the adjacent stope to provide ventilation before stoping activities in that stope begin.

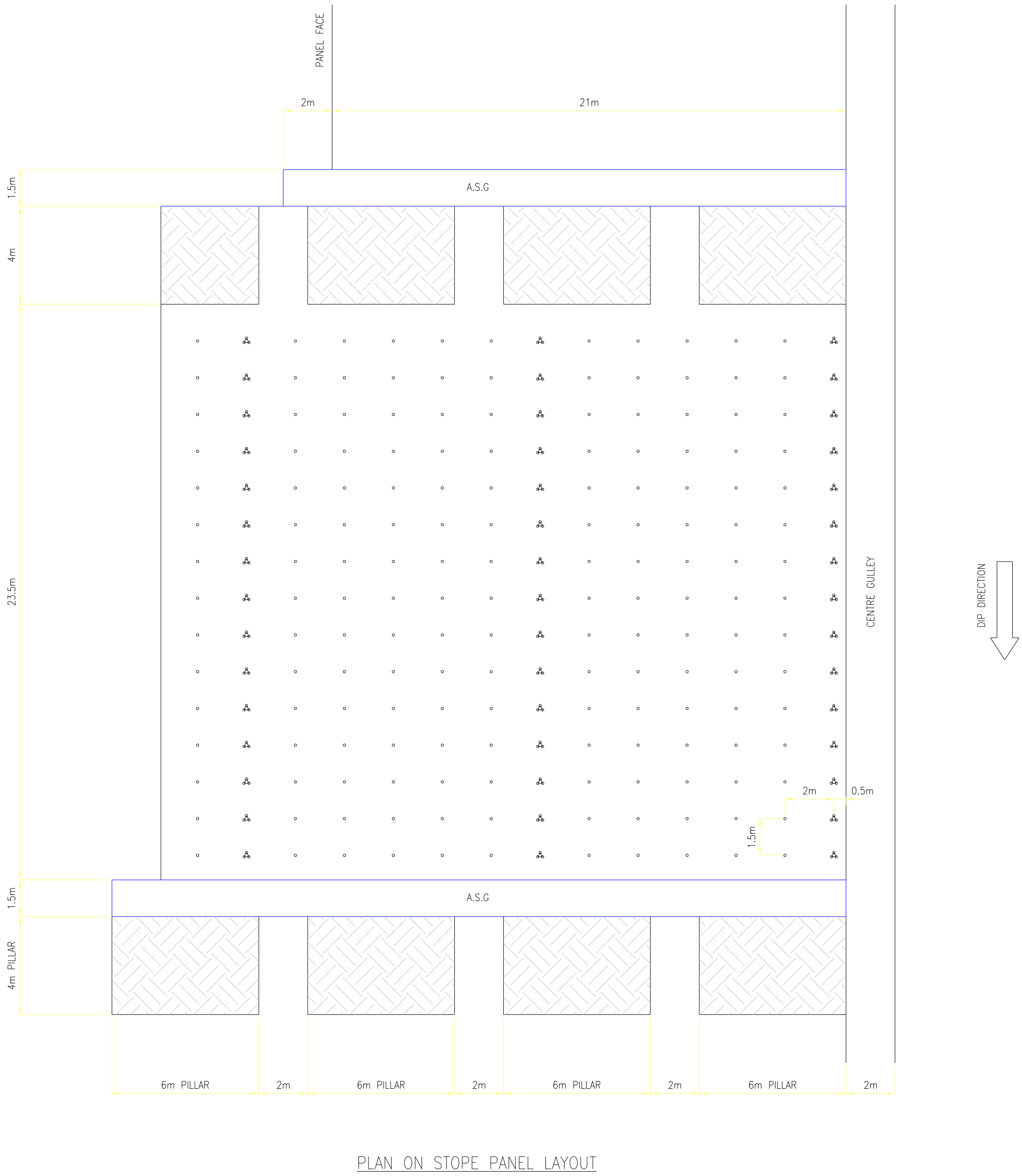

Figure 5.1: Typical conventional stope panel layout at Modder East Operation (Pooley , 2006)

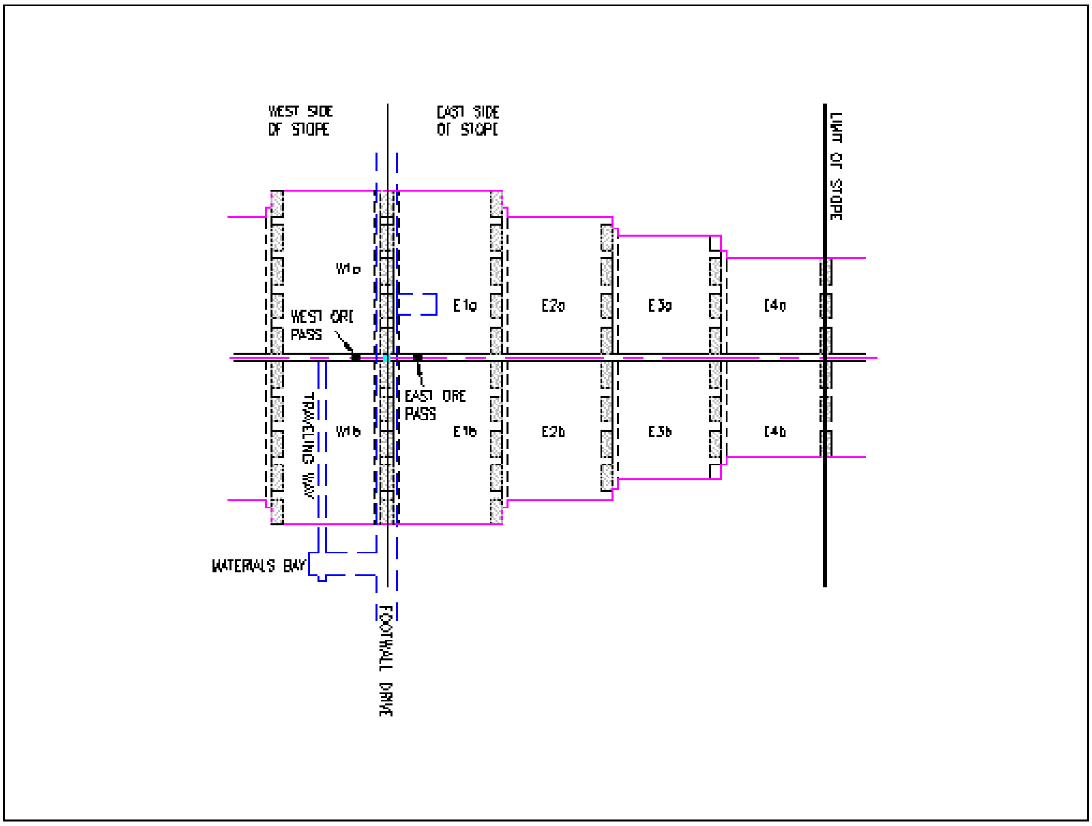

Figure 5.2: Typical conventional breast mining stope configuration at Modder East Operation (Pooley , 2006)

5.2.2 Bord and pillar mining

While mining the orebody, particularly the BPLZ, it was found that some of the geological blocks could be mined over a wider cut. This finding led to the implementation of B&P mining in 2018. The reef channel was much thicker in these blocks due to the grade distribution over the earlier and much wider Basal Unit of the Black Reef Formation. The grade distribution is more even over the entire channel width, but at a lower overall grade. This occurrence was initially identified in previous studies but based on the geological interpretation at the time, the compositing of BPLZ and Basal Unit or BPLZ and Black Reef Channel Facies, these were not converted into the reserve. This was exacerbated by the higher cut-off grade at the time (Cook, 2020).

The B&P sections have higher production outputs because of the large excavations. The geology associated with the cost of mining dictates the mining method, which accounts for potential dilution and waste. The typical layout of a B&P section at Modder East (Figure 5.3) has bord dimensions of 15 metres wide by 5.5 metres high (maximum regardless of reef thickness). The pillars are 10 metres by 10metres by 5.5metres, with a 5-metre gallery between pillars to allow for ventilation.

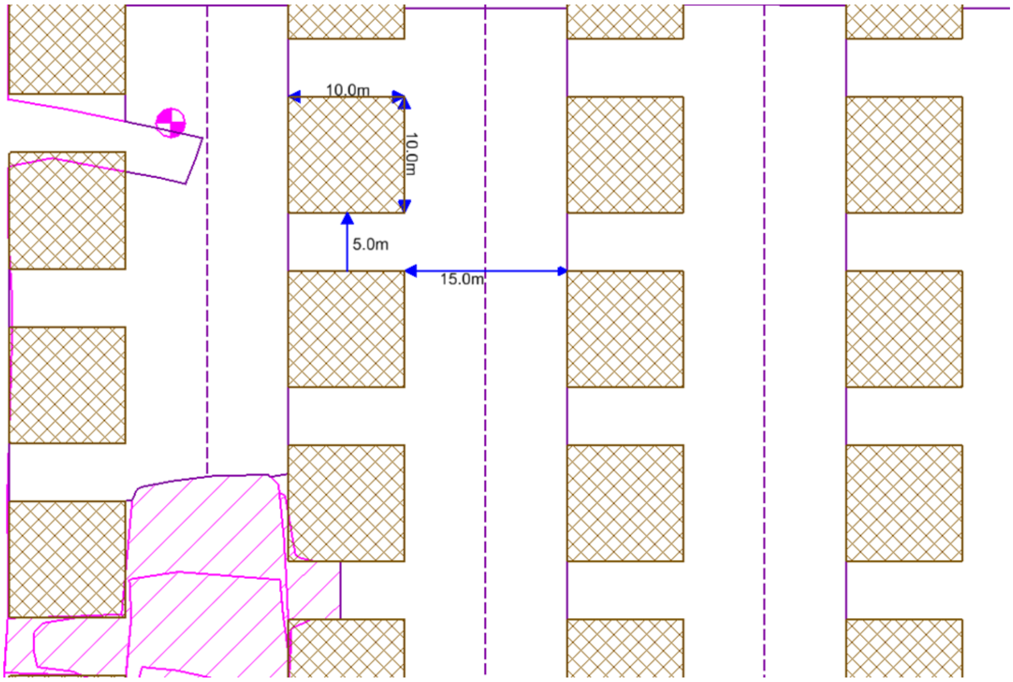

Figure 5.3: Typical Bord and Pillar layout at Modder East (Martin, R, 2019)

5.2.3 Rigid pillar mining

Rigid support pillars were incorporated into the stope layout (Figure 5.1) of the original mine design. In 2018, around the same time the operation started B&P mining, it was found that artificial support could be implemented in the form of grout pumped into large bags. These could replace the need for the rigid pillars which would allow for the removal of the pillars. Removal of the rigid pillars led to the incorporation of the pillar resources into the life-of-mine plan for the operation in 2019.

5.3 Direct Costs Per Mining Method

The costing system accounts for all costs in the above mining methods using the ABC method. The costs extracted in conjunction with the applicable gold price were used to determine the marginal cut-off grades associated with each mining method. In calculating the cut-off grade per mining method, the direct costs associated with the mining method was calculated by the process of SC (discussed in Section 4.1). All the direct costs associated with the respective mining methods were collated and all of these are stated in Annexure C. These costs do not include indirect costs or costs that are necessarily associated with any mining method but with the operation as a whole. Based on the mine design, the mine planner plans the development and mining of each block and each mining method.

5.3.1 Direct cost comparison per mining method

The direct costs associated with the respective mining methods are comparable (Table 9); all these mining methods produce reef with various dilutions. The on-reef development ore is highly diluted as the sole purpose for the development is to support reef production in the form of stoping, mechanical stoping and pillar mining.

Table 9: Mining method direct cost comparison

	Labour R/t	Consumables R/t	Other R/t	Total R/t
Conventional stoping	144	107		251
Bord & Pillar mining	31	24		55
Pillar mining	97	106	744	947
Trackless reef development	121	39		161
Conventional reef development	649	137		786

Of significance is the comparison of conventional stoping and mechanised stoping (B&P mining). The latter mining method is characterised by higher production output per square metre and less labour. In addition to the lower labour numbers, the mechanised nature of the B&P mining is more efficient than conventional mining. Labour efficiency on conventional stoping is 27m^2 per stope employee or

89 tonnes per stope employee, while the B&P mining efficiency is 30m^2 per employee or 417 tonnes per stope employee.

The other noticeable figure is that for pillar mining, which is expensive, mainly due to the cost of grout and the associated pumping costs of the material underground. However, the investment case is sound because these pillars are usually of higher grade, most of them coming from the early years of the operation when the grade was much higher. Compared to mechanised mining, labour associated with the mining of pillars is less efficient due to the slow nature of mining the pillars because of the re-equipping of workplaces and the placing and pumping of the support pillars.

5.4 Indirect Mining Costs and Overheads

In addition to direct mining costs, there are also indirect mining costs that are associated with mining activities. Indirect mining costs include waste development, mining logistics and equipping crews costs. Modder East's indirect production costs include indirect mining and engineering costs associated with its underground operation (Table 10).

Table 10: Indirect production costs

Indirect Production Costs	Annual Budget (R million)	R/t Treated
Mining (Development and other)	73	68
Engineering and logistics	389	360

Overheads would include the cost of service departments such as finance, administration and human resources (Table 11) and are represented within the mining method cost. Similarly, all costs that form part of the AOC metric are represented under each mining method. The entire operation is geared towards gold production, supporting the philosophy that all service departments are in place to support mining in whatever form. Given that one mining method may have less staff than another, it, therefore, should use fewer resources from the Human Resource and Payroll departments. For this study, the mine's entire overhead, which is a fixed cost, was calculated and divided by the total amount of reef tonnes treated (Table 11). This overhead cost was then applied to each mining method based on the tonnes produced. Capitalised mine development, which is the cost of waste development, is shown as a credit (Table 11). The credit is part of the sustaining capital cost, which is accounted for under the AISC.

Table 11: Operational Overheads per Annum

Overheads	Annual Budget (R million)	R/t Treated
Management	24	22
Mining (Fixed costs)	201	186
Engineering (Fixed costs)	186	172
SHEQ	53	49
Mineral Resource Management	32	30
Metallurgical Plant (Fixed costs)	69	64
Human Resources	91	85
Finance & Administration	57	53
Corporate costs	3	2
Capitalized mine development	(263)	(243)

The indirect costs and overheads will be absorbed per mining method based on the tonnages mined per mining method (Table 12). Based on the tonnage apportionment per mining method (Table 12), the costs were calculated for use in the cut-off grade calculation. The direct costs for Modder East are calculated based on the life-of-mine proportion of tonnes per mining method and are expressed as a percentage of total tonnes (Table 13). Once the direct costs were established, the indirect costs, overhead costs and treatment costs were added to give a total AOC per year. The AOC per year assumes that mining will be done in the proportions allocated to the various mining methods.

Table 12: Reef tonnes per mining method (thousand tonnes)

Mining Method	2019	2020	2021	2022	2023	2024	2025	2026	TOTALS	%
Conventional stoping	499	394	388	370	371	380	386	309	3 098	38%
TMM Stoping	454	533	540	564	567	560	550	283	4 051	49%
Pillars mining	57	95	99	101	108	107	114	77	758	9%
TMM Development	50	42	40	34	23	22	21	4	236	3%
Conventional Development	20	16	12	11	11	10	9	1	90	1%
TOTAL	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**673**	**8 233**	**100%**

5.5 Cut-off Grade Calculation

The breakeven approach is the approach that was used in this study to calculate the cut-off grade, which is the same as that used in Chapter 3. The reason for using the breakeven approach in both cases is discussed in Chapter 2 (section 2.1.6), which explained the cut-off grade dilemma. The dilemma arises when deciding whether to optimise an orebody based on the life of mine or economic value. When the optimization objective is to maximise economic value, the cut-off grade is raised, and a large portion of the orebody is excluded from the resource to reserve conversion. Also, the life of the operation is shortened. Given the social impact of mining on the country, it is essential to optimise profitably for life. Many of the stakeholders, such as labour and communities, are highly dependent on the operation for their well-being (Mugwagwa, 2017).

The economic factors (Table 1) and the recovery factors (Table 3) discussed in Chapter 3 remain relevant. The change, however, is concerning the AOC recalculated by mining method (Table 13). The recalculated AOC demonstrates the effect on costs of the respective mining methods shown under direct costs.

Table 13: Adjusted Operating Costs recalculated by mining method

Direct Costs	Proportion	Tonnage	Cost (R'000)	R/t
Conventional Stoping	38%	406 705	102 209	251
Bord & Pillar Mining	50%	538 958	29 624	55
Pillar Mining	8%	91 772	86 891	947
Trackless Reef Development	3%	30 569	4 910	161
Conventional Reef Development	1%	11 996	9 428	786
Total direct costs	100%	1 080 000	233 062	216
Indirect Production Costs				
Mining (Waste development & other)			73 190	68
Engineering and logistics			389 276	360
Total indirect costs			462 465	428
Treatment costs		1 080 000	136 428	126
Overheads				
Management			23 955	22
Mining (Fixed costs)			200 751	186
Engineering (Fixed costs)			186 081	172
SHEQ			53 416	49
Mineral Resource Management			32 117	30
Metallurgical Plant (Fixed costs)			69 152	64
Human Resources			91 395	85
Finance & Administration			57 416	53
Corporate Costs			2 550	2
Capitalized Mine Ddevelopment			(262 649)	(243)
Total overhead costs			454 184	421
Total Adjusted Operating Costs			**1 286 139**	**1 191**
Sustaining capex			**359 340**	**333**

Comparing the recalculated AOC (Table 13) with the initial AOC (Table 2), shows that the recalculated AOC has come down by R39/t and this is only related to the direct costs. The effect of this reduction is shown in the following cut-off grade calculation, using Equation 1:

$$xc = \frac{M + P + O + C}{[r.(V - R)]}$$

Where: xc = Cut-off grade (grammes of product per tonne of ore),

M = Mining cost per tonne of ore to be processed,

P = Processing cost per tonne,

O = Overhead cost per tonne to be processed,

C = Sustaining capital per tonne processed,

r = The portion of valuable material recovered from the ore,

V = Value or selling price of a unit of the product, and

R = Refining and selling cost per unit of the product

Applying the economic factors, recovery factors and the cost factors, the cut-off grade for each reef type is calculated as follows:

(1) BPLZ and Basal Reef:

$$xc = \frac{\text{R216/t+R428/t+R126/t+R421/t+R333/t}}{[85\%\ (560\text{-}3)]}$$

$$= 3.21 \text{ g/t}$$

(2) UK9A Reef:

$$xc = \frac{\text{R216/t+R428/t+R126+R421/t+R333/t}}{[84\%\ (560\text{-}3)]}$$

$$= 3.24 \text{ g/t}$$

Reading off GTC for BPLZ (Figure 3.3), at the cut-off grade of 3.21 g/t, the tonnage above cut-off becomes 4,4 million tonnes and the average grade becomes 6.33 g/t. On the GTC for Basal Reef (Figure 3.4), at the cut-off grade of 3.21 g/t, the tonnage above the cut-off grade becomes 1,0 million tonnes and the average grade becomes 3.88 g/t. The GTC for UK9A Reef (Figure 3.5) shows that at a cut-off grade of 3.24 g/t, the tonnage above cut-off becomes 3,6 million tonnes and the average grade above cut-off becomes 3.83 g/t. In all cases, the cut-off grade has dropped leading to an increase in tonnage above cut-off and a slight decrease in the average grade above cut-off

5.6 Net Present Value Calculation

A simplistic mining schedule based on the mining method mix (Table 14) demonstrates the revised cut-off grade calculations (Section 5.4) as well as the available tonnage per reef type and mining method.

Table 14: Life-of-mine schedule per reef type based on mining method split

		2019	2020	2021	2022	2023	2024	2025	2026	2027
Black Reef										
Tonnage Mined	'000 t	697	662	620	549	480	459	443	386	73
Grade Mined	g/t	6,33	6,33	6,33	6,33	6,33	6,33	6,33	6,33	6,33
Grade Recovered	g/t	5,38	5,38	5,38	5,38	5,38	5,38	5,38	5,38	5,38
Gold Recovered	'000 kg	4	4	3	3	3	2	2	2	0
Basal										
Tonnage Mined	'000 t	224	205	153	147	143	81	41	-	-
Grade Mined	g/t	3,88	3,88	3,88	3,88	3,88	3,88	3,88		
Grade Recovered	g/t	3,30	3,30	3,30	3,30	3,30	3,30	3,30		
Gold Recovered	'000 kg	1	1	1	0	0	0	0		
Kimberley										
Tonnage Mined	'000 t	159	213	308	384	457	541	596	694	208
Grade Mined	g/t	3,83	3,83	3,83	3,83	3,83	3,83	3,83	3,83	3,83
Grade Recovered	g/t	3,22	3,22	3,22	3,22	3,22	3,22	3,22	3,22	3,22
Gold Recovered	'000 kg	1	1	1	1	1	2	2	2	1
Total Mined										
Tonnage Mined	'000 t	1 080	1 080	1 080	1 080	1 080	1 080	1 080	1 080	281
Grade Mined	g/t	5,45	5,37	5,27	5,11	4,95	4,90	4,86	4,72	4,48
Grade Recovered	g/t	4,63	4,56	4,47	4,33	4,19	4,14	4,11	3,99	3,78
Gold Recovered	'000 kg	5	5	5	5	5	4	4	4	1

The revised life of mine schedule (Table 14) informs a revised mining method split, which is based on the higher tonnages from the reduced cut-off grades (Table 15), expressed as reef tonnes per mining method.

Table 15: Reef tonnes per mining method (thousand tonnes)

Mining Method	2019	2020	2021	2022	2023	2024	2025	2026	2027	TOTALS	%
Conventional stoping	499	373	343	355	356	366	376	427	262	3 359	38%
TMM Stoping	454	552	590	589	577	570	570	550	-	4 452	50%
Pillars mining	57	95	90	91	114	113	104	74	20	758	8%
TMM Development	50	42	40	34	23	22	21	21	-	253	3%
Conventional Development	20	17	16	11	9	8	9	8	-	99	1%
TOTAL	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**1 080**	**281**	**8 921**	**100%**

The costs associated with mining the orebody and the associated revenue received allows for the modelling and calculation of a revised NPV (Table 16) based on the proposed mining method mix. The resultant NPV (Table 16) is higher when compared to when the orebody was mined conventionally only.

Table 16: NPV based on mining method mix

		2019	2020	2021	2022	2023	2024	2025	2026	2027
Gold Price	*$/oz.*	1 230	1 230	1 230	1 230	1 230	1 230	1 230	1 230	1 230
Exchange Rate	*R/USD*	14,15	14,15	14,15	14,15	14,15	14,15	14,15	14,15	14,15
Gold Price	*R/kg.*	559 568	559 568	559 568	559 568	559 568	559 568	559 568	559 568	559 568
Revenue	***R mil***	**2 797,517**	**2 754,094**	**2 700,871**	**2 615,707**	**2 531,915**	**2 502,891**	**2 481,718**	**2 411,711**	**594,737**
Adjusted Operating Costs	***R mil***	**1 392,624**	**1 388,933**	**1 384,409**	**1 377,170**	**1 370,047**	**1 367,580**	**1 365,781**	**1 359,830**	**351,256**
Mining & treatement costs	*R mil*	1 283,730	1 283,730	1 283,730	1 283,730	1 283,730	1 283,730	1 283,730	1 283,730	334,266
Corporate costs	*R mil*	2,550	2,550	2,550	2,550	2,550	2,550	2,550	2,550	0,664
Royalty tax	*R mil*	106,344	102,653	98,129	90,890	83,767	81,300	79,501	73,550	16,326
Sustaining Capital	***R mil***	**359,338**	**359,338**	**359,338**	**359,338**	**359,338**	**359,338**	**359,338**	**359,338**	**93,567**
Cash Flow (Pre-tax)	***R mil***	**1 045,556**	**1 005,824**	**957,124**	**879,199**	**802,530**	**775,973**	**756,600**	**692,544**	**149,914**
Discount Rate	***12%***									
NPV (R mils)		***4 500***								

When comparing the conventional and mining method mix output (Table 17), a drop in cut-off grade (3%) resulted in a R79 million increase in NPV. Another outcome was that the increased tonnages added an additional year to the life of mine. This demonstrates the effect of the lower cut-off grade on NPV, and even with the resultant lower recovered grade, the NPV was improved by 2%.

Table 17: Comparison between conventional mining and mining method mix

	Unit	Conventional Mining	Mining Method Mix	% Variance
Cut-off grade - Black Reef	g/t	3,30	3,21	-3%
Cut-off grade - Basal	g/t	3,30	3,21	-3%
Cut-off grade - UK9A Reef	g/t	3,34	3,24	-3%
Tonnage	*'000 t*	8 233	8 921	8%
Recovered grade	*g/t*	4,42	4,29	-3%
Gold recovered	*'000 kg.'s*	36	38	5%
NPV	*R mil's*	4 421	4 500	2%

5.7 Summary of Chapter 5

This chapter discussed the rationale for changing from conventional mining only to a mix of mining mix methodology. The mining methods were described and the direct costs, the only real change to the cost analysis, was also described. A direct cost comparison was given and the difference in the costs of each mining method explained. The other costs associated with mining the orebody were also described and calculated. Other costs include indirect mining costs and operational overheads, which all form part of the overall AOC.

Once the costs were established, a revised cut-off grade was calculated. The calculation showed a 3% change in cut-off grade for all reef types, an 8% increase in the available tonnages and a 5% increase in gold recovered, increasing the life-of-mine in the model. The final stage of the exercise was to produce a discounted cash flow for the updated model and a resultant NPV. The comparison showed that the NPV had increased by 2%. In the next chapter, the mining methods are simulated through a Monte Carlo Simulation package called @Risk attempting to improve the NPV by optimising the mining method mix.

6 NET PRESENT VALUE SIMULATION

6.1 Overview of Chapter 6

Chapter 6 demonstrates how the @Risk software simulates the results obtained in Chapter 5 and then optimises those results based on the set criteria. Parameters of the simulation are changed and the results obtained are analysed. The conclusions are highlighted showing the optimised NPV obtained.

The study has researched cut-off grades and found several theories on the optimisation of NPV, which is based on economic value optimisation while sacrificing the life of the operation. In this study, the determination of cut-off grade has been based on the breakeven cut-off grade approach, which is the one approach that protects the life of the operation rather than shortens it, as do so many of the other approaches (Githiria & Musingwini, 2018). The cut-off grade calculated in Chapter 5, utilised the possible (not optimal) mining method mix, which would reduce overall costs thereby lowering the cut-off grade bringing more of the resource into the reserve and ultimately improving NPV (refer to Table 17).

While the development, both on and off-reef, has remained the same to support stoping in all its forms, there is room to improve by utilizing more B&P mining and less conventional stoping. Due to the difference in the cost associated with the respective methods, the optimal mining method mix choice would be where the most cost-effective method is maximised and the least cost-effective is minimised. The objective of this chapter is to demonstrate how a computer-aided simulation program, @Risk, would optimise the NPV, by simulating the optimum mining method mix, without changing the life-of-mine. The programme was introduced in Chapter 2 and employed Monte Carlo Simulation optimising its output based on several variables.

6.2 Simulation Inputs

The first step in the optimisation process is to apply @Risk to the model generated in Chapter 5. In this instance, the main driver of NPV is the costs associated with the model, specifically, direct costs. The direct costs per mining method have been expanded from the Chapter 5 model for ease of input (Table 18).

Table 18: Direct costs per mining method

Direct Costs (R million)	2019	2020	2021	2022	2023	2024	2025	2026	2027	TOTALS
Conventional stoping	611	457	420	435	436	449	461	523	320	4 112
TMM Stoping	466	568	606	605	593	586	586	565	-	4 576
Pillars mining	109	183	174	174	219	217	200	141	38	1 455
TMM Development	57	47	45	39	26	25	24	24	-	286
Conventional Development	35	31	29	19	16	14	15	15	-	174
TOTAL	**1 279**	**1 285**	**1 274**	**1 272**	**1 291**	**1 291**	**1 285**	**1 268**	**358**	**10 603**
R/t	*1 184*	*1 190*	*1 180*	*1 178*	*1 195*	*1 195*	*1 190*	*1 174*	*1 273*	*1 189*

The variances in costs are based on the tonnages associated with each mining method. It is this variability that was modelled using @Risk attempting to optimise NPV. The original tonnages are those shown in Table 15.

When inputting the variables into @Risk, ranges for each variable needs to be given, such as a minimum and maximum. The author used 10% either way of the most likely or original tonnages. To ensure that the maximum tonnage available in each year was not exceeded, a total tonnage of 1,080,000 per annum was used as a constraint. The requires the number of iterations to be set. The number of iterations was set at 10,000. During simulation, a profitability distribution for each variable must be selected. For this study, the Pert distribution was selected, which is a continuous bounded distribution. It was chosen because it is the one most commonly used in project management and cost analysis and because of the narrow range selected.

6.3 Net Present Value Simulation Outputs

The @Risk Optimizer tool was used with AOC and NPV for the conventional mining and the mining method mix serving as inputs (Table 19). The optimised output expressed as optimised NPV (Table 19) showed a marginal increase in the NPV.

Table 19: Cost and NPV comparison

	Unit	Conventional Mining	Mining Method Mix	Optimized NPV
Adjusted Operating Costs	*R'million*	10 832	11 358	11 321
NPV	*R'million*	4 421	4 500	4 521

Using a Pert distribution for all relevant variables (Section 6.1), the simulation was done and produced the distribution for optimised NPV (Figure 6.1). The distribution for optimised NPV shows that the original mining method mix follows a normal distribution. From this distribution, there is a 50% chance of exceeding the optimised NPV.

The distribution was obtained by not changing any of the parameters in the model but given that these changes occur later in the life of mine, this would have a minimal impact on NPV. The graphic output generated by @Risk (Figure 6.2) demonstrates the contribution per mining method to the change in the optimised NPV.

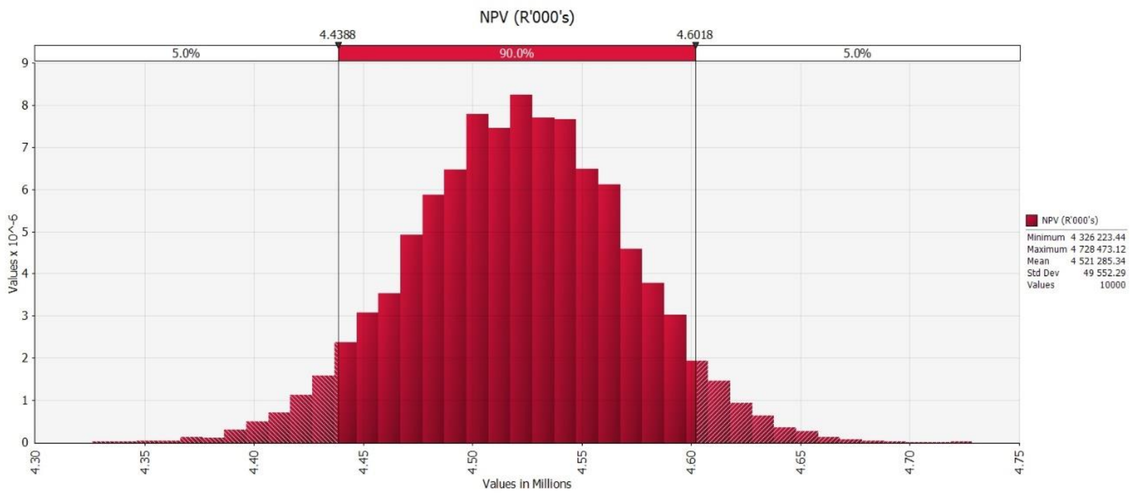

Figure 6.1: Distribution for optimised NPV

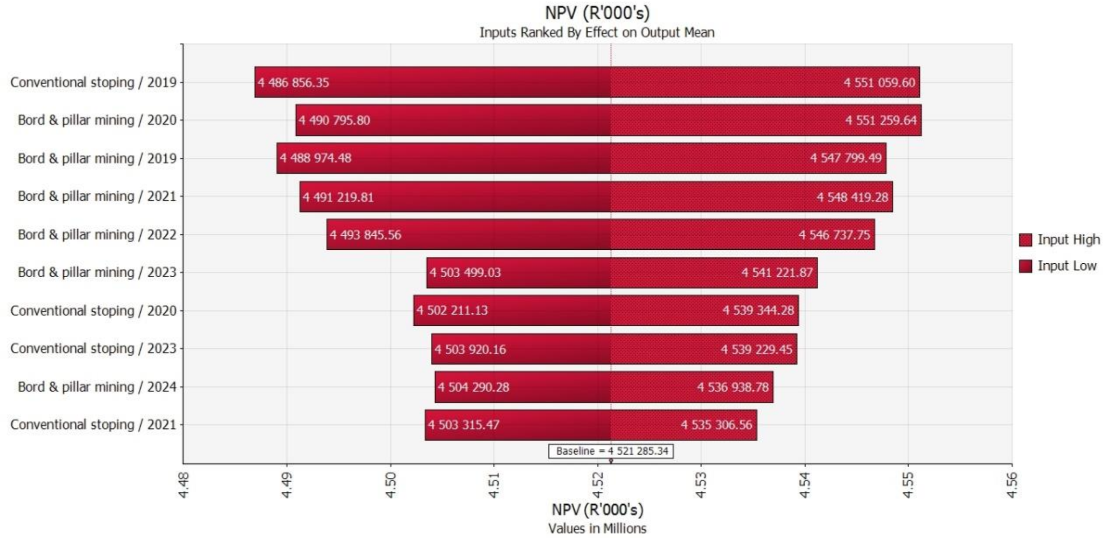

Figure 6.2: Contribution per mining method to the change in NPV

6.4 Net Present Value Based on Higher Parameters

Given that the narrow input range of 10% either way of the most likely scenario (Section 6.1) produced a relatively small (2%) change in NPV, a trial at 30% was done. The change to the input range (Table 20), shows a more significant difference in the NPV.

Table 20: Optimised NPV based on parameters of +-30% of the most likely scenario

	Unit	Conventional Mining	Mining Method Mix	Optimized NPV
Adjusted Operating Costs	*R'million*	10 832	11 358	11 213
NPV	*R'million*	4 421	4 500	4 585

The change allowed the simulator to make more aggressive use of the lower cost B&P mining and less of the most expensive pillar mining. Subsequently, the distribution (Figure 6.3) indicates that the chance of exceeding the optimised NPV (Table 20) is still 50%. In this scenario, the original NPV for the mining mix falls outside one standard deviation of the mean. Applying the change to range generates a tornado graph demonstrating the effect of the mining methods on the change in NPV (Figure 6.4).

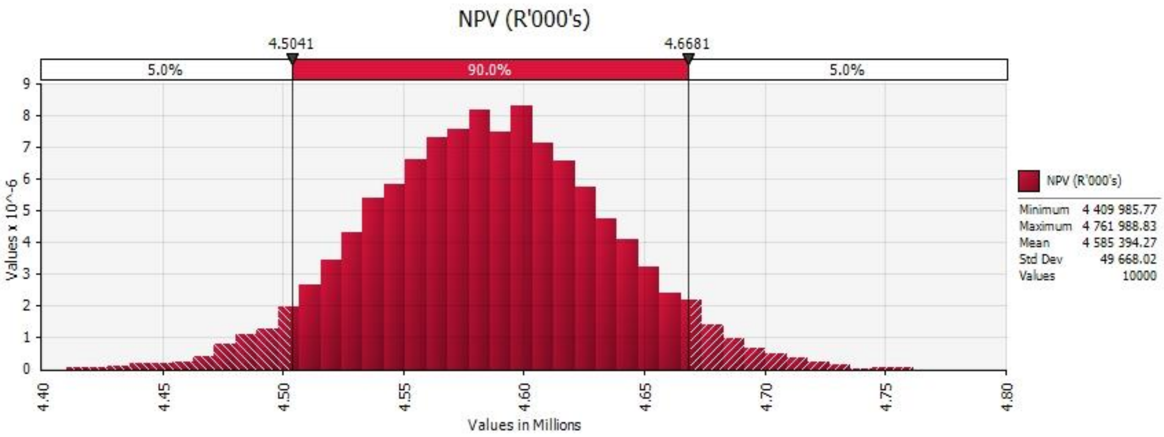

Figure 6.3: Distribution for optimised NPV with amended parameters (+- 30%)

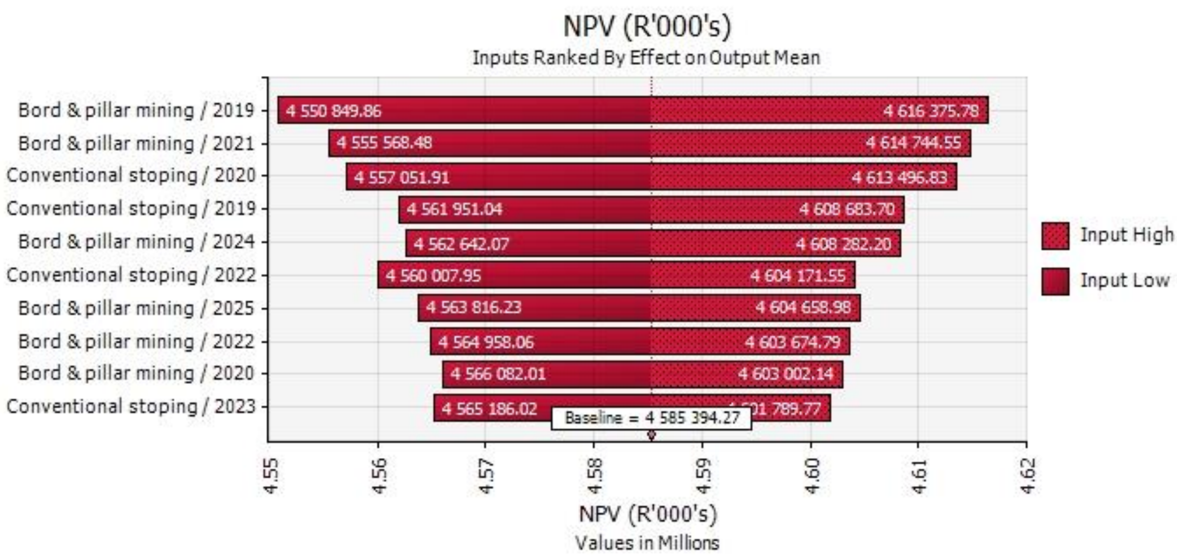

Figure 6.4: Tornado graph of the effect of the mining methods on the change in the NPV

6.5 Summary of Chapter 6

This section of the report experimented with the optimisation of the NPV while maintaining the life of the operation. In this way, the optimal mining method that would reduce the operating costs and thus maximise the NPV was used. This approach strengthened the NPV by approximately R164 million. It is important to note that both on-reef and off-reef development was maintained and not modelled. In this chapter, the focus was on using the @Risk software, introduced in Chapter 2, and applying the different inputs into the programme to document the outputs. Based on the Monte Carlo simulation, @Risk simulated the optimal NPV output based on the constraints of the various inputs described previously.

7 CONCLUSIONS AND RECOMMENDATIONS

Modder East employs the breakeven cut-off grade approach to calculate cut-off grades for its respective reef types, using the average AISC per tonne. Their approach does not consider the effect of the cost of the various mining methods on the average AISC and the effect on the resource to reserve calculation. In testing the hypothesis, a comparison was drawn between the traditional method of calculation and the method proposed in this research. The conclusions of the @Risk simulations showed that there was an increase in the NPV of the operation when using the latter method.

7.1 Conclusions

In first establishing whether the correct principles were in place and ensuring that the approach being proposed was in line with current research, further interesting insights were gleaned. The paper by Githiria and Musingwini (2018) highlighted the comparison of the various approaches to cut-off grades and the impact that these approaches had on the NPV of the operation. Their research supported the conclusion that the optimisation of NPV is associated with a shortened life (Githiria & Musingwini, 2018). The cut-off grade dilemma discussion led to a peripheral goal of this study of maximising NPV while maintaining the life of the operation. Given the strategy of the parent company, Gold One, and the general view of stakeholders, it was important to simultaneously maximise the life of mine and produce an optimal NPV. The principal requirements included converting resource to reserve, mining this in line with current capacity and ensuring that value is maintained.

Throughout the study, the importance of costs and the measurement thereof was highlighted. Being a relatively new operation, Modder East was able to establish a basis for recording costs from the start of its operations, which would ultimately lead to the outcomes presented in this study. The costing system was built on the ABC approach, but for a long time, the recording of costs was not adequately managed to ensure that the benefits thereof were obtained. It was not until the implementation of SC that costs were appropriately analysed and recorded. The premise of the accurate recording of costs in ABC was highlighted by several authors, Poniewierski, 2016; Swamidass, 2000; Cooper & Kaplan, 1991. Accurate cost recording associated with ABC formed the theoretical framework that underpinned the analysis of costs associated with the respective mining methods and used in the cut-off grade calculation relevant to this research.

With the data in place, the analysis of the cut-off grade by mining method and reef type produced a marginally improved cut-off grade. However, this had the desired effect of adding more tonnes to the reserve. Additionally, a further year of processing was added to the life-of-mine of the operation. When the various tonnages were applied to the different mining methods, the net effect was an improved NPV. Therefore, the stated goal had been achieved and an incidental goal of improving the life of the operation, which arose from the research was also achieved.

The improved results were fed into the @Risk software and various simulations were performed. The first step was to optimise the output of the operation (NPV) while leaving the life of the operation constant. Once this was done a simulation based on the optimisation was done, which again increased the NPV of the operation, ultimately by nearly R164 million. The statistics showed that the original NPV before optimisation was below one standard deviation of the mean so obtained, giving confidence that there was room to improve the NPV while maintaining the life of the operation.

7.2 Recommendations

The tonnages and grades associated with the new approach to cut-off grade calculation would still need to be planned and scheduled in line with current mining activities. The mining constraints associated with each mining method would need to be taken into account, which may affect the outcomes stated in the report. Some individual geological blocks would have several mining methods associated with them such as B&P, pillar mining and conventional mining. This complication must be accounted for during the planning of the development process.

Proper planning of the different mining methods is essential, especially the extraction of the rigid pillars. While the grade of the older pillars (from, the earlier in the life of mine) is much higher than that of more recent pillars, the cost of extraction will be high. Balancing the mining method mix is critical; specifically, it would be important to mine as much of the B&P areas as possible in the earlier years to maximise NPV. In an ideal blend, the mining of the pillars associated with the high cost and high content would be combined with B&P mining, which is associated with lower cost and lower content.

The recommendation is to apply the new approach to cut-off grade calculation when the new block model is produced to allow for optimisation from inception. The new approach would improve the tonnages and product available for conversion to the reserve. Additionally, the life-of-mine plan should model these changes per mining method and reef type. At the same time, the costing system should be maintained and continually enhanced to improve the accuracy of the modelling.

REFERENCES

Asad, M. & Topal, E., 2011. Net present value maximization model for optimum cut-off grade policy of open pit mining operations. *The Journal of The Southern African Institute of Mining and Metallurgy,* 111(11), pp. 741-749.

Bascetin, A. & Nieto, A., 2007. Determination of optimal cut-off grade policy to optimize NPV using a new approach with optimization factor. *The Journal of the South African Institute of Mining and Metalurgy,* 107(2), pp. 87 - 94.

Birch, C., 2017. Optimization of cut-off grades considering grade uncertainty in narrow, tabular gold deposits. *Journal of the Southern African Institute of Mining and Metallurgy,* 117(2), pp. 149 - 156.

Birch, C., 2019. Optimisation of Mining Block Size for Narrow Tabular Gold Deposits. In: S. I. Publishing, ed. *Proceedings of the 27th International Symposium on Mine Planning and Equipment Selection - MPES 2018.* s.l.:Springer International Publishing, pp. 121 - 141.

Chen, J., 2019. *Heuristics.* [Online]
Available at: https://www.investopedia.com/terms/h/heuristics.asp

Cook, E., 2020. *Erstwhile Mineral Resource Manager - Gold One: Modder East Operations* [Interview] (2 March 2020).

Cooper, R. & Kaplan, R., 1991. Profit Priorities from Activity-based Costing. *Harvard Business Review,* 69(3), pp. 130 - 135.

Dehghani, H. & Ataee-pour, M., 2012. Determination of the effect of operating cost uncertainty on mining project evaluation. *Resources Policy,* 37(1), pp. 109 - 117.

Githiria, J. M. & Musingwini, C., 2018. Comparison of cut-off grade models in mine planning for improved value creation based on NPV. *Society of Mining Professors 6th Regional Conference,* 6(March), pp. 347 - 362.

Harris, J., 2013. *World Gold Council.* [Online]
Available at: https://www.gold.org/
[Accessed 05 November 2018].

Heuberger, R., 2005. Risk analysis in the mining industry. *Journal of The South African Institute of Mining and Metallurgy,* 105(2), pp. 75 -79.

Labro, E., 2019. Costing Systems. *Foundations & Trends in Accounting,* 13(April), pp. 267 - 404 .

Lane, K., 1964. Choosing the optimum cut-off grade. *Colorado School of Mines Quarterly*, pp. 811 - 829.

Lane, K. F., 1988. *The Economic Definition of Ore. Cut-off Grades in Theory and Practice.* London: Mining Journal Books Ltd.

Lind, G., 2001. Activity Based Costing: Challenging the way we cost underground coal mining systems. *The Journal of The South African Institute of Mining and Metallurgy,* 3(April), pp. 77-82.

Martin, R, 2019. *Bord and Pillar Drawing.* Springs: Modder East.

Martinelli, R., Collard, J. & Gamache, M., 2019. Strategic planning of an underground mine with variable cut-off grades.. *Optimization and Engineering,* pp. 1 - 49.

Merriam-Webster Inc., 2020. *"Heuristic".* 2020 ed. Springfield : Merriam-Webster Inc..

Minnitt, R., 2004. Cut-off grade determination for the maximum value of a small wits-type gold mining operation. *Journal of The South African Institute of Mining and Metallurgy,* 104(5), pp. 277 -283.

Modder East, 2009. *Chart of Accounts.* Springs: Modder East.

Modder East, 2019. *Modder East Budget,* Johannesburg: Gold One.

Mugwagwa, D., 2017. *Cut-off Grade Optimisation for a Bimetallic Deposit: A Case Study of the Ruashi Mine Copper-Cobalt Deposit,* Johannesburg: University of Witwatersrand.

Noreen, E., 1991. Conditions Under Which Activity-Based Cost Systems Provide Relevant Costs. *Journal of Management Accounting Research,* 3((fall)), pp. 159-169.

Ogilvy, 2019. *Geological Modelling and Resource Estimation of the Kimberley and Black Reefs at Modder East Gold Mine,* Johannesburg: Shango Solutions.

Palisade, 1984. *Maker of @Risk.* United Kingdom.

Poniewierski, J., 2016. *Negatively Geared Ore Reserves - A Major Peril of the Break-Even Cut-Off Grade.* Brisbane, Australasian Institute of Mining and Metallurgy Publication Series, pp. 236 - 247.

Pooley, 2006. *Aflease Gold Ltd Modder East Gold Project, Bankeable Feasibility Study,* Johannesburg: Turgis Consulting (Pty) Ltd.

Rendu, J., 2014. *An Introduction to Cut-Off Grade Estimation.* Engelwood: Society for Mining, Metalurgy and Exploration.

SRK Consulting (South Africa) (Pty) Ltd. , 2013. *Modder East Underground operation Independent Technical Report,* Johannesburg: SRK Consulting (South Africa) (Pty) Ltd..

Stewart, R., 2014. *Maximising value in mature mining operations through the application of strategically aligned cut-off grades,* Coventry: University of Warwick (un-published).

Swamidass, P., 2000. *Encyclopedia of Production and Manufacturing Management.* Boston: Kluwer Academic Publishers.

Taylor, H., 1972. General Background Theory of Cut-Off Grades. *Transactions of the Institutions of Mining and Metallurgy*, pp. 160-179.

Thomson, M. & Barr, D., 2014. Cut-off grade: A real options analysis. *Resources Policy,* Volume 42, pp. 83 - 92.

Venmyn Deloitte, 2016. *Competent Persons' Report on the Mineral Assets of Gold,* Johannesburg: Venmyn Deloitte.

Yapo, A. & Camm, T., 2017. All-in sustaining cost analysis: Pros and cons. *Mining Engineering,* 69(9), pp. 16 - 28.

Yi, R. & Sturgul, J., 1987. Analysis of Cutoff Grades Using Optimum Control Theory. *Journal of South African Institute of Mining and Metallurgy,* Volume 3(Geostatistics), pp. 263 - 269.

ANNEXURE A: WORLD GOLD COUNCIL CLASSIFICATION OF COSTS

Table 21 World Gold Council classification of costs (Harris, 2013)

Cost Category	Source	US $ / gold ounces sold
On-site mining and processing costs (on a sales basis)	Income Statement	(a)
On-site general and administrative costs	Income Statement	(b)
Royalties and production taxes	Income Statement	(c)
Realised gains and losses on hedges of operating costs	Income Statement	(d)
Community costs related to current operations	Income Statement	(e)
Permitting costs related to current operations	Income Statement	(f)
3rd party smelting, refining and transport costs	Income Statement	(g)
Non-cash remuneration (site-based)	Income Statement	(h)
Stockpile, leach pad and product inventory write-downs	Income Statement	(i)
Operational Stripping Costs	Income Statement	(j)
By-product and co-product credits (*Note: will be a credit*)	Income Statement	(k)
Sub-total (Adjusted operating costs)		(l) = (a)+(b)+(c)+(d)+(e)+(f)+(g)+(h)+(i)+(j)+(k)
Corporate or regional general and administrative costs, including share-based remuneration (sustaining)	Income Statement	(m)
Reclamation & remediation – accretion & amortisation (operating sites)	Income Statement	(n)
Exploration and study costs (sustaining)	Income Statement	(o)
Capital exploration (sustaining)	Cash Flow	(p)
Capitalised stripping & underground mine development (sustaining)	Cash Flow	(q)
Sustaining capital expenditure	Cash Flow	(r)
Sustaining leases	Cash Flow	(s)
All-in Sustaining Costs		(t) = (l)+(m)+(n)+(o)+(p)+(q)+(r)+(s)
Growth and development costs not related to current operations	Income Statement	(u)
Community costs not related to current operations	Income Statement	(v)
Permitting costs not related to current operations	Income Statement	(w)
Reclamation and remediation costs not related to current operations	Income Statement	(x)
Exploration and study costs (non-sustaining)	Income Statement	(y)
Capital exploration (non-sustaining)	Cash Flow	(z)
Capitalised stripping & underground mine development (non-sustaining)	Cash Flow	(aa)
Non-sustaining capital expenditure	Cash Flow	(bb)
Non-sustaining leases	Cash Flow	(cc)
All-in Costs		= (t)+(u)+(v)+(w)+(x)+(y)+(z)+(aa)+(bb)+(cc)

ANNEXURE B: COSTS FOR MODDER EAST

Table 22: Modder East budgeted costs 2019 (Modder East, 2019)

Modder East Operations: Budgeted Cost 2019

Division	Fixed	Variable	Total
Management	23 955		23 955
Mining	200 751	348 570	549 320
Egineering	186 081	389 276	575 357
Safety, Health, Environment & Qulaity	53 416		53 416
Mineral Resource Management	32 117		32 117
Metalurgical Plant	69 152	136 428	205 580
Human Capital	91 395		91 395
Finance & Administration	59 966		59 966
Capitalized Mine Development	(262 649)		(262 649)
Total	**454 184**	**874 273**	**1 328 457**

ANNEXURE C: STANDARD COSTING PER MINING METHOD/ACTIVITY

D1: Trackless Mechanised Machinery – Development

Standard Development End	Unit	Standard
WIDTH	m	6
HEIGHT	m	5.50
ADVANCE PER BLAST	m	2.80
DRILL STEEL LENGTH	m	3.20
ADVANCE PERAMETER	m	360.00
No. of Holes per Blast	m	76.00
No. of support holes per Blast	no.	5.00
SQUARE METERS	m^2	30.25
DEPTH PER HOLE	m	3.20
BURDEN	cm	75.00
METERS PER BIT	no.	160.00
HOLES PER BIT 45MM	no.	50.00
HOLES PER BIT Support 32MM	no.	34.00
Ground Density	g/dm^3	2.78
Ton per Square Meter	t	15.29

Stores & Material	R/m	R/t
Stores: Blasting Consumables	1 950.09	12.77
Stores: Drill Steel & Bits	886.50	6.41
Stores: Pumps	683.01	9.71
Stores: Scrapers & Winches	5.14	0.02
Stores: Support	207.48	5.24
Stores: Stores - Other	381.37	5.11
Total Stores & Material	**4 113.59**	**39.26**

Labour	R/m	R/t
@7m/development empl.	1 855.29	121
TOTAL	**5 968.88**	**161**

D2: Conventional Development

Standard Convl End	Unit	Standard
RAISE OR WINZE WIDTH	m	1.80
RAISE OR WINZE HEIGHT	m	2.40
ADVANCE PER BLAST	m	1.00
DRILL STEEL LENGTH	m	1.50
ADVANCE PERAMETER	m	30.00
HOLES PER GULLEY	no.	27.00
SQUARE METERS	m^2	4.32
DEPTH PER HOLE	m	1.20
Ground Density	g/dm^3	2.78
Ton per Square Meter	t	5.00

Stores & Material	R/m	R/t
Stores: Blasting Consumables	811.46	26.37
Stores: Drill Steel & Bits	187.80	19.24
Stores: Hydropower	305.93	18.27
Stores: Scrapers & Winches	553.03	40.36
Stores: Support	-	-
Stores: Stores - Other	327.57	32.92
Total Stores & Material	2 185.80	137.16
Labour	**R/m**	**R/t**
@ 4m/development employee	3 246.75	648.83
TOTAL	**5 432.55**	**786**

D3: Conventional Pillar Mining

Summary Variable cost per m² Pillar Mining		
Description	**R/m²**	**R/t**
Zero Based Budget Consumables per m²	R352.44	R105.65
Labour	R325.03	R97.43
Grout Mix	R1 995.68	R598.22
Contractor Labour Variable	R485.45	R145.52
Total Rand per m²	**R3 158.59**	**R946.82**

D4: Conventional Stoping

Standard Panel	Unit	Standard
FACE LENGTH	m	23
FACE LENGTH excl. GULLY	m	21,20
STOPE WIDTH	m	1,20
GULLEY WIDTH	m	1,60
GULLEY HEIGHT	m	2,70
ADVANCE PER BLAST	m	0,80
M² PER BLAST	m²	18,40
HOLES PER PANEL	no.	92,00
HOLES PER GULLY	no.	27,00
HOLES PER VENT HOLING	no.	8,50
TOTAL HOLES	no.	127,50
DEPTH PER HOLE	m	1,00
Bits per Blast	no.	4,11
Meters per Bit	m	31,00
Holes per Bit	no.	31,00
Ground Density	g/dm³	2,78
Ton per Square Meter	t	3,34

Stores & Material	R/m²	R/t
Stores: Blasting Consumables	**168,30**	**50,45**
Stores: Drill Steel & Bits	**45,39**	**13,61**
Stores: Hydropower	**28,26**	**8,47**
Stores: Scrapers & Winches	**55,04**	**16,50**
Stores: Support	**41,05**	**12,31**
Stores: Stores - Other	**19,32**	**5,79**
Total Stores & Material	**357,37**	**107,12**

Labour	R/m²	R/t
@ 27m2/stope emp; therefore 0.04 emp/m2	481,00	**144,18**
TOTAL	**838,37**	**251,31**

D5: Mechanical Stoping

Standard Panel	Unit	Standard
WIDTH	m	15
HEIGHT	m	5.00
ADVANCE PER BLAST	m	3.00
DRILL STEEL LENGTH	m	3.20
ADVANCE PERAMETER	m	360.00
No. of Holes	no.	161.00
No. of Holes Support	no.	9.00
SQUARE METERS	m²	75.00
DEPTH PER HOLE	m	3.20
BURDEN	cm	85.00
BLASTING HOLES PER BIT	no.	55.00
BITS PER BLAST	no.	2.93
METERS PER BIT	m	176.00
SUPPORT HOLES PER BIT	no.	50.00
SUPPORT BITS PER BLAST	no.	0.54
SUPPORT METERS PER BIT	m	110.00
ROOFBOLTS PER BLAST	no.	27.00
Ground Density	g/dm³	2.78
Ton per Square Meter	t	13.90
Emulsion kg. per Hole	kg.'s	6.8

Stores & Material	R/m²	R/t
Stores: Blasting Consumables	**161.47**	**11.62**
Stores: Drill Steel & Bits	**79.31**	**5.71**
Stores: Pumps	**69.32**	**4.99**
Stores: Scrapers & Winches	**-**	**-**
Stores: Support	**18.08**	**1.30**
Stores: Stores - Other	**2.92**	**0.21**
Total Stores & Material	**331.11**	**23.82**

Labour	R/m²	R/t
@30m²/mechanical empl.	433	31
# TOTAL	357	55