Title: Economic Viability of K04 Open Pit Mineral Resource to Ameliorate Production Hiatus in 2019 and 2022

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ABSTRACT

Venetia Mine is the largest diamond mine for the De Beers Group of Companies (DBGC) in South Africa contributing approximately 80% of revenue for the company. The Venetia Kimberlite Cluster (VKC) comprises of 13 pipes and one dyke, of these K01, K02, and K03 are mined productively while the satellite bodies that are intersected during waste stripping and push backs are stockpiled separately.

The current open pit operation will come to an end in 2022, whereupon the K01 and K02 ore bodies will be mined from underground. Shaft sinking and development of underground infrastructure is underway, and it is envisaged that the change over from open pit to underground will be effected in 2022.

There are two challenges facing the company that may result in a production hiatus in 2019 and 2022. The first challenge relates to failure to open kimberlite ore in the Cut4 south push back by 2019 and the second one is failure to seamlessly change from open pit to underground in 2022 when the open pit reaches its Life of Mine (LoM).

The backlog of waste stripping in the Cut4 south and technical challenges faced in the implementation and execution of the Canadian Shaft Sinking method at Venetia Mine have both pointed to periods where there will be less ore mined thus resulting in less carats recovered. In order to cushion for these scenarios, the economic viability of K04 satellite pipe was investigated by conducting a Whittle optimum pit selection of the pipe to mine the ore. This was previously excluded by the current pit design.

The strategy for K04 optimum pit selection was based on net present value (NPV), revenue factors, stripping ratios, and the life of the mine. This strategy was employed when analysing 13 pits from scenario 2. The selection process identified pit 9 as an optimum pit. It must be noted that this pit inventory does not add significant numbers of carats to the overall carat profile to lift the valley of despair in 2019 and 2022 but will contribute some NPV of ZAR13.2 million over a period of 6 years.

K04 project will contribute 402 thousand carats against 1.5 million carats required to fill the hiatus gap, hence does not support a sound business case for Venetia Mine. It is therefore recommended that DBCM deploy a different strategy to reduce the impact of the carat shortfall in 2019 and 2022.

Some recommendations include reducing mining activities in the north of the pit and focus more on accelerating waste stripping in the south in order to expose ore sooner. Cost saving initiatives such as reducing operating costs and postponing some capital projects that are deemed not urgent will go a long way in the reducing the financial impact in 2019 and 2022.

Dedication

To my wife Rudo, my daughter Ruvarashe and my son Tinashe with love and appreciation.

Acknowledgements

I would like to acknowledge the invaluable input of the following persons and organisations:

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LIST OF ABBREVIATIONS

ACK	Aphanitic Coherent Kimberlite
CIMVAL	The Canadian Institute of Mining, Metallurgy and Petroleum on Valuation of Mineral Properties
CRB	Country Rock Breccia
CK	Coherent Kimberlite
CKB	Coherent Kimberlite Breccia
CKBE	Eastern Coherent Kimberlite Breccia
CKBN	Northern Coherent Kimberlite Breccia
CRD	Coarse Residue Deposit
CV	Competent Mineral Asset Valuator
DBCM	De Beers Consolidated Mines
DBGS	De Beers Group Services
DCF	Discounted Cash Flow
DMS	Dense Media Separation
EBIT	Earnings Before Interest & Tax
FOS	Factor of Safety
Ha	Hectare
IRR	Internal Rate of Return
JORC	Joint Ore Reserves Committee Code
LCC	Life Cycle Costs
LoM	Life of Mine
LDD	Large Diameter Drilling
LMB	Limpopo Mobile Belt

MCAF	Mining Cost Adjustment Factor
MCF	Mine Call Factor
MVK	Massive Volcaniclastic Kimberlite
MVKE	Eastern Massive Volcaniclastic Kimberlite
MVKW	Western Massive Volcanoclastic Kimberlite
NPV	Net Present Value
ORU	Ore Resource Update
OWC	Ore Waste Contact
RF	Revenue Factor
R&M	Repair and Maintenance
RsCr	Resource Carat Ratio
RPO	Recognized Professional Organization
SAMCODE	South African Mineral Reporting Code
SAMREC	South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves
SAMVAL	South African Code for the Reporting of Mineral Asset Valuation
SARS	South African Revenue Services
SBP	Strategic Business Plan
SSP	Satellite Sampling Program
SIB	Stay in Business
TMR	Tailings Mineral Resources
TS	Technical Services
UCS	Uniaxial Compressive Strength

VREP	Venetia Resource Extension Program
VUP	Venetia Underground Project
VIP	Value Improvement Project
VKC	Venetia Kimberlite Cluster

LIST OF SYMBOLS

cpht	carats per hundred tonnes
km	kilometer
m	meter
m^3	cubic meter
Ma	million years
masl	meters above sea level
mbgl	meters below ground level
Mt	million metric tonnes
t/h	tonnes per hour
t/m^3	tonnes per cubic meter
USD	United States Dollar
ZAR	South African Rand

1. INTRODUCTION

Diamonds have played a pivotal role in the operations of De Beers Consolidated Mines Proprietary Limited (DBCM) for the last century. Its oldest mines namely, Kimberley Mines, Finsch, Koffifontein and Cullinan, have all been sold to third party operators.

Venetia Mine has an estimated life of mine (LoM) of 20 years post open closure in 2022 while Voorspoed Mine is coming to the end of its life later in 2019. This scenario puts Venetia Mine in a position where the carat production profile must be around 4.2 million carats per annum in order to keep the DBCM business viable and profitable.

Production at Venetia Mine is currently coming from the open pit whose life is coming to an end in 2022 when the underground will come into production. Underground development operations are currently in progress with the open pit to underground transition envisaged in 2022.

The split shell mining method employed at Venetia Mine has resulted in exhausting the ore in the northern half of the pit, thus delaying waste stripping in the south creating a bottle neck to expose ore in 2019. This coupled with underground development delays has created a production hiatus resulting in carats deficit in 2019 and 2022.

The study will cover the economic viability of K04 kimberlite in order to assess if there is a business case to mine the pipe. The work will centre around Whittle optimisation and the selection of the optimum pit that can be mined. Sensitivity analysis will be conducted to measure the impact of economic parameters to the project. Thus, the prime importance of this research is to identify other viable ore sources that can be treated and thereby significantly lift the valley of despair projected in 2019 and 2022 using the optimal pit selection process.

1.1 Background

The Venetia Kimberlite Cluster, commonly known as the (VKC), comprises of 13 pipes and one dyke (Tait, 2007). Of these, three kimberlite pipes namely, K01, K02 and K03, represent the 99% of 194 million tonnes of the total mineral resource. The remaining satellite kimberlite bodies are all less than 1.0 ha in size, and most are less than 0.2 ha.

Figure 1-1 shows the spatial distribution of these kimberlite pipes around Venetia Mine. The VKC pipes are characterised by irregular, elongated shapes generally orientated west-northwest. An age of 519 million years (Ma), obtained from mica separates using Argon-Argon dating, has been determined for the Venetia kimberlites (Phillips et al., 1999). This age roughly agrees with the Allsopp et al. (1995) U-Pb model for perovskite.

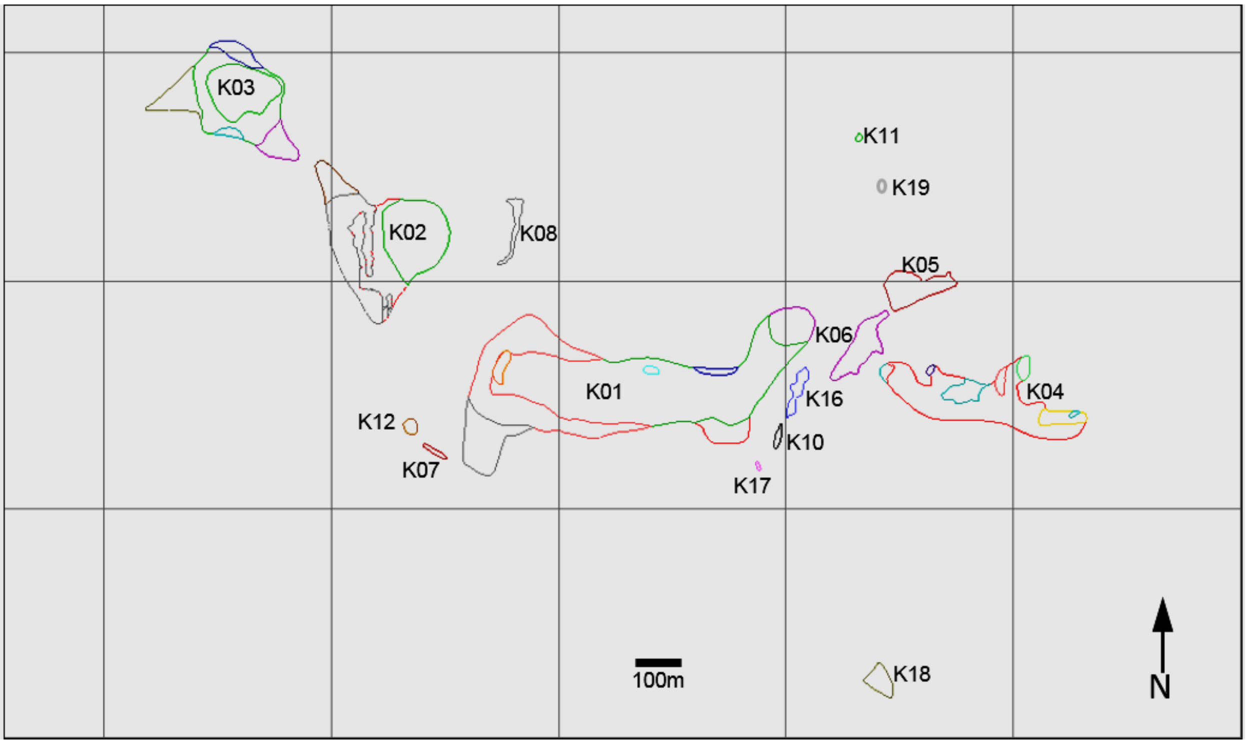


Figure 1-1: Plan-view showing the Venetia Mine kimberlite cluster.

The ore treated is being mined from K01, K02, K03 and from satellite pipes intersected as the pit opens. In particular, K04, K05 and K06 provide other resources for treatment at adhoc basis depending on where mining is taking place in a specific period.

The current LoM for the open pit will end in 2022 when mining will be at a depth of 456 mbgl, beyond which it will be unviable for surface mining. However, resource extension programs have proven that there are viable resources below the open pit totalling about 117 million tonnes of ore, which can be exploited by underground mining. The Venetia

Mine is currently developing an underground mine that will continue production beyond 2022 when open operations cease.

The delay in waste stripping in the Cut4 south to expose the open pit ore by 2019 and technical challenges in underground infrastructure development will result in a deficit in total carats recovered in 2019 and 2022. Figure 1-2 shows the two valleys of despair where there will be a drop in the carat profile. The shortfall in carats is a function of both grade and tonnages. The first valley in 2019 is when the open pit has exhausted Cut4 north ore and Cut4 south ore is not yet fully exposed. The second valley of despair in 2022 is when there is no seamless transition from open pit to underground. These two events will result in carats shortfall thereby impacting the cash flow of the mine. The spike in carat production between the first and second valleys is a function of grade.

Thus, to cushion for the production hiatus, other sources of ore and buffers such as stockpiles should be prepared and investigated so that production will not be interrupted. One such ore source to be investigated is K04.This work will provide an opportunity to understand alternative potentially viable ore sources at Venetia Mine, which can be exploited to fill in the hiatus gap during the underground ramp up process.

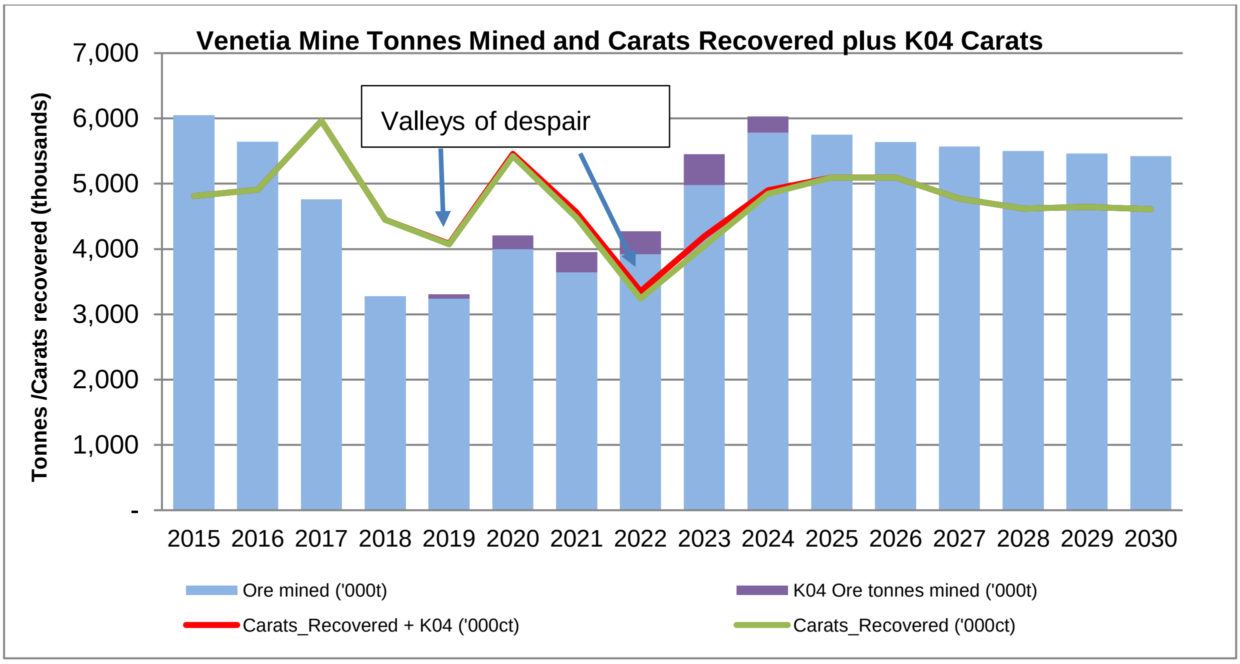


Figure 1-2: Graph showing Venetia Mine's tonnes and carats mined.

1.2 Research Motivation

In recent years, exploration companies, including experienced companies such as De Beers have not located many economic diamond deposits worldwide. This has left mining companies with no option but to exploit the already known deposits including low grade mineral resources, which has resulted in diamond demand outpacing supply. Thus, any known diamond deposit is worthwhile evaluating to determine its economic viability.

The predicted future rough diamond growth (Bain and Company, 2016) is an upside to mining K04. It presents an opportunity to analyse the economic viability of K04 with the view of it filling the anticipated production hiatus during the open pit and underground change over.

1.3 Project Location

Venetia Mine is located on the farm Venetia; 103MS owned by De Beers Consolidated Mines Limited, which was recently acquired by Anglo American (see Figure 1-3). The farm is 879 hectares in size and lies approximately 27 km south of the confluence of the Shashe and Limpopo Rivers, 80 km west of the town of Musina and 37 km north-east of the farming community of Alldays. Venetia is approximately 540 km by road from Johannesburg and is 700 masl (Venetia Mine, 2018). The mine is accessible via a tarred provincial road from Musina or Alldays.

Figure 1-3: Map showing the location of Venetia Mine (Southern Africa Places, 2015).

1.4 History of the Mine

Diamond bearing gravels were discovered close to the Limpopo River on a farm called Seta, in early 1903. De Beers began a reconnaissance program to locate the source of these river deposits in 1969. The program culminated in the discovery of kimberlite pipes Venetia farm in 1980 (Mokoatle, 2010). A thorough evaluation began the same year to assess the economic potential of the ore bodies.

Between 1980 and 1983 a total of 80 core holes (1700 m) were drilled to further explore and delineate the main pipes to a depth of 180 m. A total of 30 Mt of material was collected between the 1983 and 1985 from trenches, shafts, underground tunnels, surface alluvium and large diameter drilling (LDD) sampling. Mineral resource estimates were finalised in November 1985 resulting in the first 50 m x 50 m x 12 m local block model for K01 and K02. The satellite pipes, K03 to K12, were sampled further in 1995 with 18,280 m of 121/4 inch large diameter drill holes drilled to a depth of 155 m and the results were used to estimate the resources (Henning, 1996).

1.5 Country Rock Geology

The local country rock geology consists of two main packages of rocks associated with the Beit Bridge Complex (see Figure 1-4). The first package occupies the central portion of the syncline and is comprised of amphibolites, biotite gneiss, quartzite and quartzofeldspathic gneisses and these are the lithologies that are intruded by the kimberlites. The second package is a metasedimentary package with the outer limb enfolding the gneissic package and is comprised of marbles, limestones, metapelites and calc-silicate rocks. Crosscutting both packages are pegmatite dolerite dykes and sills. Table 1-1 provides a summary of the country rock lithological groups, their end members and description (Barnett, 2009).

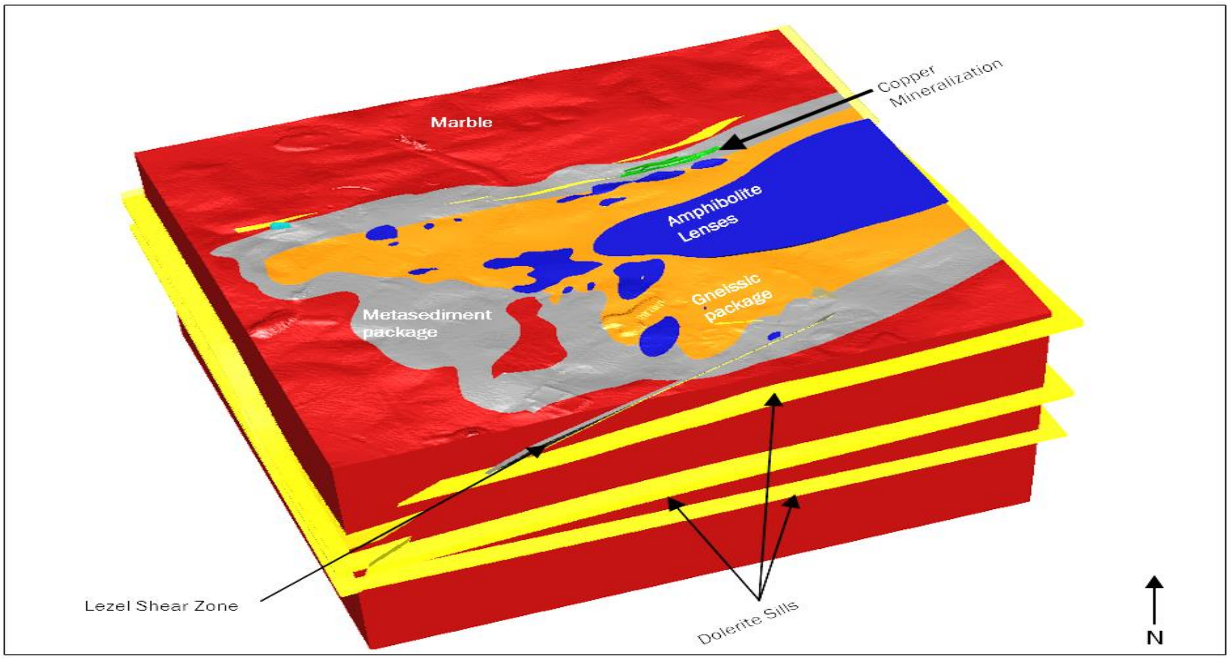


Figure 1-4: 3D view of the modelled country rock solids representing all the lithological domains (Basson, Creus and Stock, 2015).

Table 1-1: Generalised stratigraphic column used to model Venetia's country rock (Barnett, 2009).

Code	Group	Rock Type
CAL	Calcrete	Tertiary Calcrete
KM	Kimberlite	All kimberlites
DOL	Dolerite (DOL)	Karoo and Proterozoic mafic intrusives
PEG	Pegmatite	Proterozoic granitic intrusives.
SZ	Tina-Lezel Shear Zone	Cross-cutting brittle-ductile shears
BG	Malala Drift Group - Gneiss Package (BG)	Granitic gneiss, with biotite concentrated onto foliation planes
		Amphibolite and garnet bearing biotite schist
		Biotite and garnet bearing amphibolite
MP	Gumbu Group - Metasedimentary Package (MP)	Micaceous quartzite, often fuchsitic.
		Sillimanite and mica bearing, biotite-rich pelite
		Magnetite-rich quartzite
		Fine-grained carbonate, often interlayered with metapelite
MBL	Gumbu Group - Marble (MBL)	Coarse marble

1.6 Regional Geology

Venetia Mine Kimberlite Cluster is intruded in the Limpopo Mobile Belt (LMB) (see Figure 1-5). The LMB is a collision zone between the Kaapvaal craton (to the south) and the Zimbabwe Craton (to the north). Khoza et al. (2012) stated that the LMB is a highly metamorphosed Archean aged complex that comprises three zones namely, Northern Marginal Zone, Southern Marginal Zone and Central Marginal Zone. Thrust faults and shear zones separate these zones.

Venetia Mine is within the Central Zone and its kimberlite cluster (VKC) has been emplaced into the Gumbu Group of the Beit Bridge Complex. This complex comprises of a package of meta-sedimentary and upper amphibolite to granulite facies gneissic-schistose rocks (Brandl, 2002 and Barton et al., 2003).

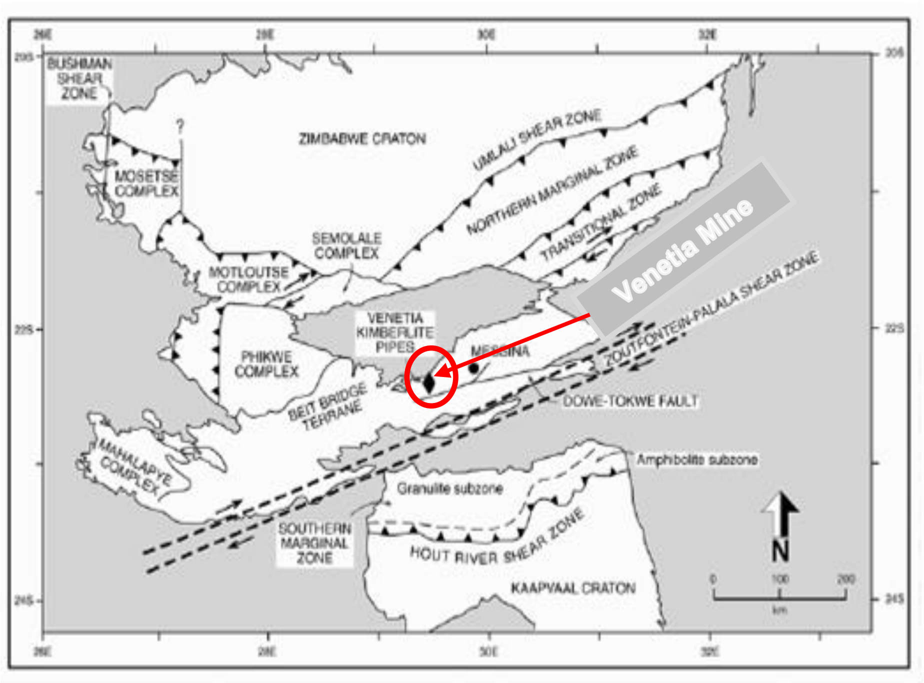


Figure 1-5: Map of Limpopo Mobile Belt (Barton et al, 2003).

1.7 Problem Statement

Venetia Mine's LoM is coming to an end in 2022 whereby it will be uneconomic to mine by open pit. In order to continue with mining operations after this date the mine is currently developing underground infrastructure with the aim of switching over to underground in 2022.

The shortfall in waste stripping of approximately 5.5 million tonnes in the Cut4 south to expose ore by 2019 have resulted in a valley of despair in 2019. Technical challenges faced by the implementation and execution of the Canadian shaft sinking method at Venetia Mine to seamlessly transition from open pit to underground have also resulted in a valley of despair in 2022. The Canadian shaft sinking method is highly mechanized and operates with few crews. This arrangement reduces costs and likelihood of accidents.

Since its inception at Venetia mine, the benefits of improved cycle times have not been fully achieved which has resulted in delayed schedules. The main reason for delays has been lack to skilled crews to fully function in this sinking arrangement that requires high level of competence to operate different sinking equipment.

It is therefore imperative to evaluate the economic viability of K04 open pit mineral resource to ameliorate production hiatus in 2019 and 2022. The spike in carats in 2020 is driven by high grade ore exposed in Cut4 south. It should be noted that K04 is a low grade resource and therefore was previously excluded as one of the main sources of carats.

On average Venetia Mine produces 4.2 million carats annually which will result in a shortfall of 0.2 million carats in 2019 and a carat deficit of 1.3 million in 2022. The aim of this research is to conduct a Whittle pit optimization on the portion of the K04 mineral resource which is outside of the current pit design (see Figure 1-6). The expected outcome of the research is to ascertain whether the carats contribution from K04 will have a significant impact to lift the valleys of despair. For K04 to have significant contribution it should produce 1.5 million carats per annum.

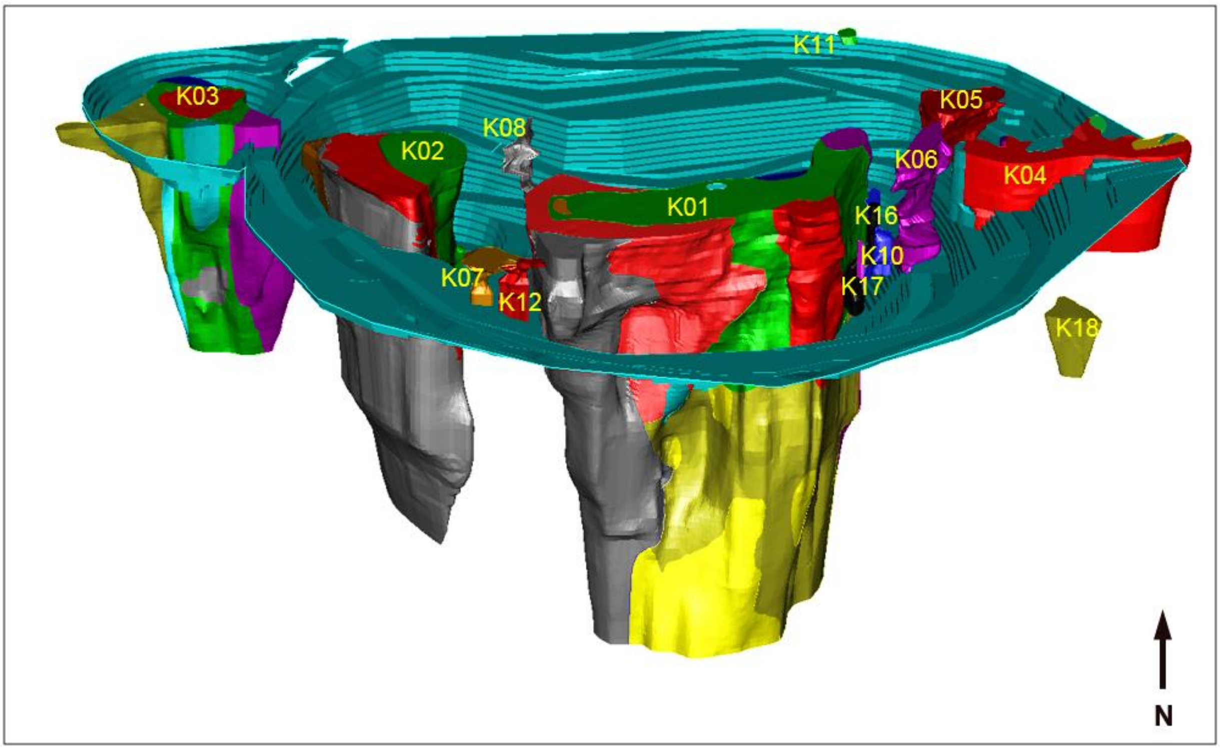


Figure 1-6: The current LoM open pit design for Venetia Mine.

1.8 Research Objectives

The aim of this study is to investigate the economic viability of mining the K04 kimberlite given the current pit design and resource estimate. Over and above this also intend to achieve the following:

- optimum pit selection of K04 resource as a standalone pit;
- Whittle pit life schedule; and
- declare a mine inventory.

1.9 Assumptions

It is assumed that there is continuity of the modelled mineral resource, ore treatability, well understood recovery factors by the plant, and that the inferred grade is applicable. The stability in the current diamond market and the bullish prices are also expected to remain on the upside. This will bolster any investment decision on mining K04.

1.10 Scope of Work

The scope of work includes: -

- Whittle optimum pit selection of K04 resource.
- To estimate how much more waste will be stripped and at what cost.
- Collation of input data on processing costs, throughput rates, recovery factors, exchange rates, royalty and mining costs with bench adjustment factors.
- A Whittle Schedule that will be used to check opportunities for ameliorating the production hiatus during the open pit and underground change over or any other earlier opportunities of treating the K04 material.

1.11 Constraints

The limitation of Whittle is the tendency to generate impractical shells, especially where the pit bottom is narrow or tight. Mitigation measures to reduce the impact of this weakness are as follows: -

- Application of a minimum practical mining width at the pit bottom

- Optimization of bench design by factoring in overall slope angles.

1.12 Research Methodology

Quantitative research techniques are used for this research. This include mine planning input parameters collection and pit optimisation using Whittle. Data analysis, derivation of different net present values (NPV) and comparison of the resultant Whittle pit life schedules are carried out.

1.13 Economic Viability Analysis

The economic viability analysis included the following phases:

- Sensitivity analysis of the resource block model
- Collation of modifying factors data of geotechnical slope angles and geotechnical domains
- Ascertaining mining costs with bench adjustment factors, processing costs, throughput rates, recovery factors, exchange rates, royalty costs and diamond prices.

  All these factors were used as inputs to run a Whittle optimum pit selection to select the best pit with optimal NPV and internal rate of return (IRR) upon which a mine inventory can be declared. Mine sections and departments that supported the undertaking of this work included the mine planning section, ore processing and geotechnical section.

1.14 Research layout

Chapter 1 introduces the research, highlights the problem statement, and states the objectives and scope of the research. Chapter 2 deals with literature review in particular the K04 geological and mineral resource models and Whittle pit optimisation. Mineral resource economic valuation methodologies which include income, market and cost approaches are also described in this section. Chapter 3 focuses on the collection of mine planning input parameters for optimum pit selection. Discussion and analysis of the results is detailed in Chapter 4. The last chapter makes comparisons between the current LoM profiles and the resultant profiles with detailed conclusions and recommendations based on the pit optimisation results. Figure 1-7 shows the layout of the research report.

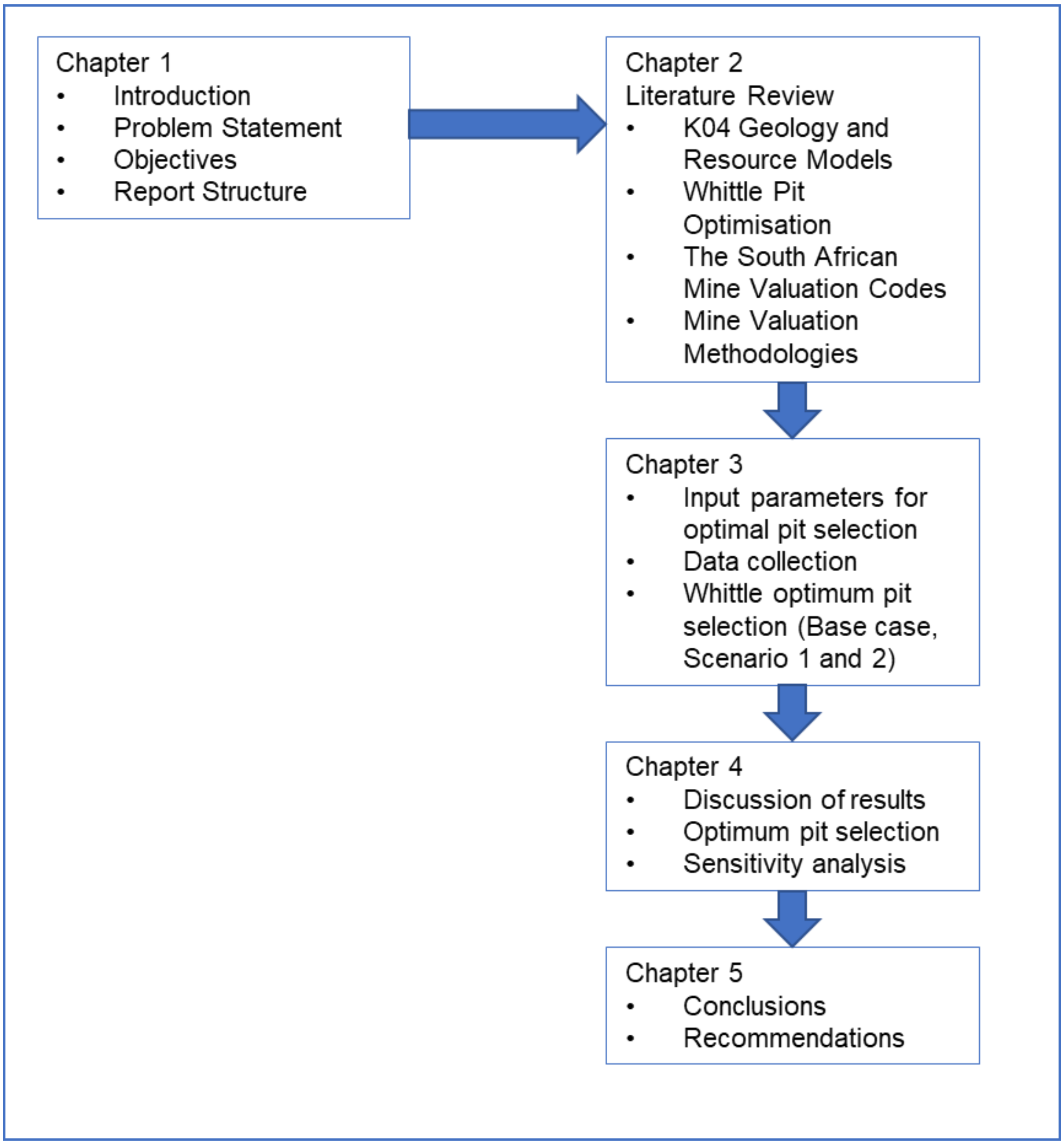


Figure 1-7: Summary of Research Report

2. LITERATURE REVIEW

2.1 Introduction

This review is divided into three sub-sections:

1. The literature on geological and resource models and historical mining of K04.
2. A brief description of Whittle pit optimisation techniques and
3. The exploration of the South African Code for the Reporting of Mineral Asset Valuation (SAMVAL) with special focus on mine valuation methodologies.

2.2 K04 Mineral Resource

2.2.1 Geological Modelling

The K04 kimberlite pipe is the fourth largest of the Venetia Mine’s kimberlite cluster and covers approximately 4.2 ha at pre-mining surface level consisting entirely of coherent kimberlite (see Figure 2-1). The eastern, central and western part of the pipe consists of a monticellite-phlogopite kimberlite (Colgan, 1982). The eastern part of the pipe is characterised by kimberlite containing abundant olivine phenocrysts, but few olivine macrocrysts (Colgan, 1983).

The northern protuberance of the pipe contains a phlogopite kimberlite breccia. The remainder of the pipe (west and central) is characterised by abundant of large and prominent olivine macrocrysts. Due to the small size and low grade of this resource compared to K01, K02 and K03, minimal additional work has been undertaken and interpretations by Colgan (1982) and Colgan (1983) have not been updated.

Nine facies were modelled and mapped for the K04 pipe**.** Old and new rock type names are captured in Table 2-1, the old geological nomenclature have been changed to align with other kimberlite pipes (K01, K02 and K03). Until additional work is done to establish facies relationships, this nomenclature should be used with caution (Mushongahande, 2012).

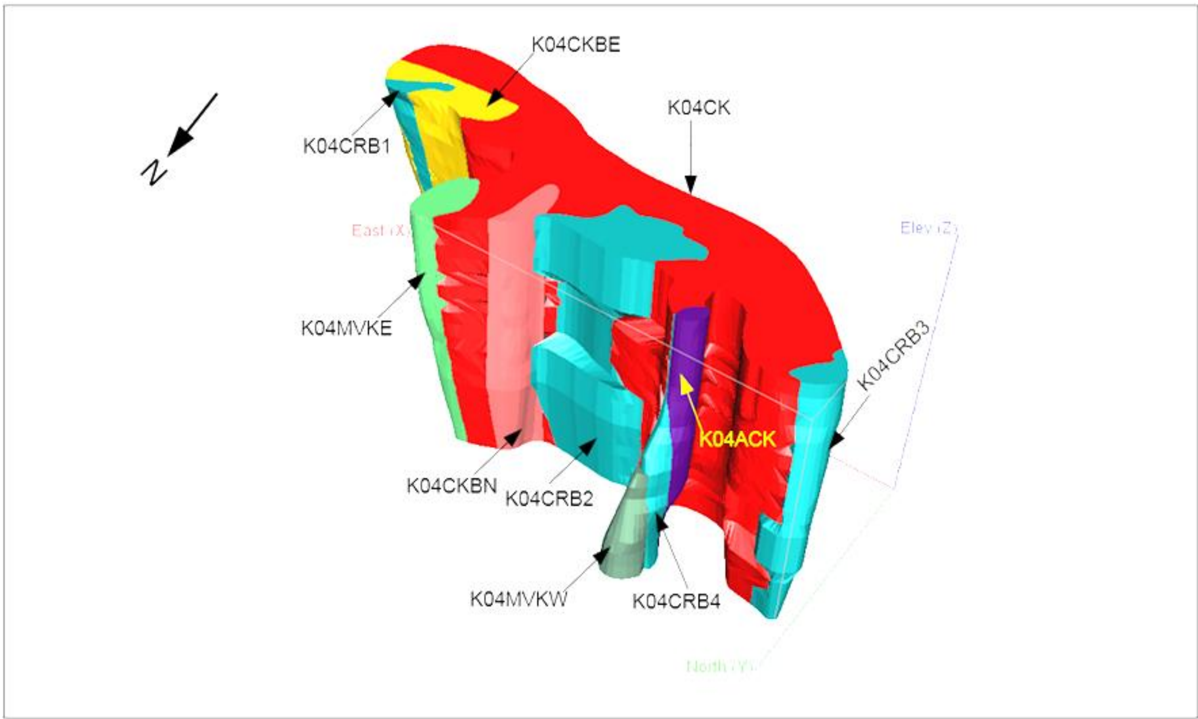


Figure 2-1: K04 Geological Model.

Table 2-1: K04 Lithofacies (Mushongahande, 2012).

Old Rock Name	New Rock Name	Description
K04CRB	K04CRB	Country rock breccia
K04HAK	K04ACK	Aphanitic coherent kimberlite
K04HK	K04CK	Coherent kimberlite
K04HKB	K04CKB	Coherent kimberlite breccia
K04HKBE	K04CKBE	Eastern coherent kimberlite breccia
K04HKBN	K04CKBN	Northern coherent kimberlite breccia
K04TKB	K04MVK	Massive volcaniclastic kimberlite
K04TKBE	K04MVKE	Eastern massive volcaniclastic kimberlite
K04TKBW	K04MVKW	Western massive volcaniclastic kimberlite

2.2.2 Mineral Resource Estimate

The K04 mineral resource was estimated in 2005 by De Beers Group Services (Pty) Ltd (Bush, 2005) from surface (~696 mamsl) to 546 mamsl (0 m to 150 m below surface) using the Satellite Sampling Program (SSP) data, drill chip log data and Venetia Resource Extension Program (VREP) drill hole data. The undepleted mineral resource of K04 comprises of 3.4 million carats contained in 21 million tonnes, however as of December 2017, the declared mineral resource comprises of 2.2 million carats contained in 15.2 million tonnes. These depletions are a result of K01 pushbacks.

A recent estimate was done by Z-Star in 2012 using the old data (Lohrentz, 2012). Although no new data was used, the reason for the new estimate, as per the recommendation during various external assurances, was to bring the resolution of the local block estimate in line with the estimates of the other resources of 25 m block size, as opposed to the current 50 m block size (Mushongahande, 2012).

2.2.3 Historical Mining

The K04 resource did not form part of the LOM resources for Venetia Mine therefore any ore currently mined from K04 is because of the pit pushbacks. This was done because K04 is a low-grade resource and the company strategy was targeted at depleting pipes with high-grade ore. As a result, the intersected K04 ore is being stockpiled on a satellite stockpile dump. Figure 2-2 shows the location of satellite stockpile. Besides K04, ore from other satellite pipes is also stockpiled on this dump. As of December 2017, the satellite stockpile had 1.5 million carats contained in 7.1 million tonnes. Although the satellite material is stockpiled separately from waste dumps is motherless contaminated with waste rock. This is because satellite pipes are encountered as puhbacks are undertaken resulting in 80% of the material being diluted. This will therefore pose challenges in treatment.

According to Bain and Company (2016), global supply of rough diamonds is expected to decline on average by 1% to 2% per annum from 2016 to 2030 because of the aging and depletion of existing mines. Therefore, this presents an opportunity to analyse K04's economic viability with the view of it filling the anticipated hiatus production gap during the open pit and underground change over.

Figure 2-2: Site map of Venetia Mine showing location of satellite stockpile (Venetia Mine, 2019).

2.3 Whittle Optimisation (Pit Optimisation Tools)

There are many software and tools in use in the mining industry to evaluate financial viability and the optimal mine strategy for mineral resources. Katakwa, Musingwini and Genc (2013) pointed out that software use has become inherent in every activity in the mining value chain. Use and choice of software is depended on various factors that include; the type of mineralisation, amount of data to be processed, organisation's preference.

Some of the tools used for financial evaluation and optimisation are; Whittle, MineMAXPlnanner, MRM, NPV Scheduler, Vulcan and Xeras. Whittle has proven to deliver excellent results at various stages of life of mine i.e. scoping; feasibility studies, life of mine scheduling and it has the embedded the Lerchs Grossmann algorithm for pit optimisation (Geovia, 2015).

Kentwell (2002) in his comparison of Whittle and MineMAXPlanner concluded that the two packages produce exactly the same optimisation results, however MineMAXPlanner is very simple and lacks the flexibility and functionality built in Whittle.

2.3.1 Pit optimisation

Madowe (2013, p.3) described pit optimisation as “a technique applied on a mineral resource to estimate the shape and extent of a potential ore reserve”. Whittle software attempts to find an optimum pit shell that results in highest NPV while meeting a selected set of production constrains. Whittle and Burks (2010) pointed out that pit optimisation without scheduling is a static analysis process because no timelines are determined for mining a block of material. The initial step is to develop nested pit shells with effective stripping ratios (Whittle and Burks, 2010).

2.3.2 Optimum pit outline

Any pit outline has a dollar value that is equal to revenue less costs. The revenues arise from tonnages and grades expected from each pit outline, price of mineral, recoveries, mining and treatment cost, taxes, royalties, etc. Thus, Snowden Mining Industry Consultants (2001, p.32) defined the optimal pit outline (see Figure 2-3) as “the one with the highest dollar value”.

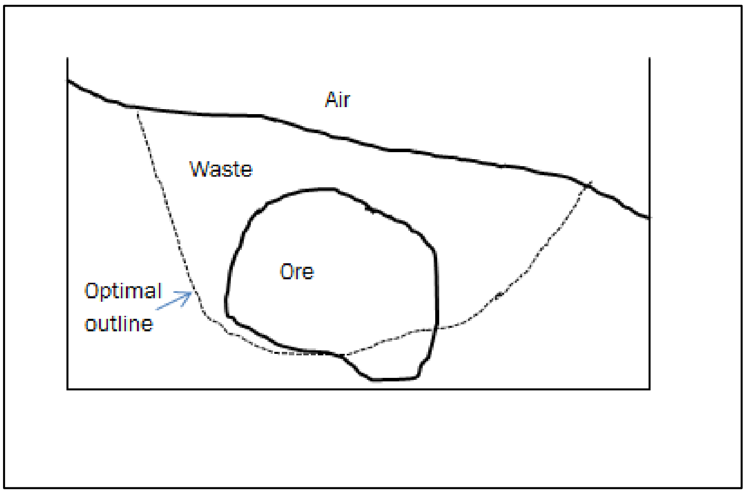


Figure 2-3: Optimal Pit Outline. (Snowden Mining Industry Consultants, 2001, p.32).

2.3.3 Factors affecting the optimal pit outline

Product prices, operating costs, slope angles, the deposit shape/geometry and stripping ratio play a crucial role in determining the ultimate optimal pit size of any mining project. Thus, if the above factors are fixed, so is the optimal pit (Snowden Mining Industry Consultants, 2001). The table describes the impact of these factors on the final pit size (Table 2-2**).**

Table 2-2: Summary of factors affecting the optimal pit outline.

Factor	Description
Product price	Generally, if the selling price of a product goes up, then the optimal pit becomes bigger. This is because the projects would now realise higher returns that can be used to fund expansion and pushbacks projects.
Operating costs	If operating costs increase, the optimal pit becomes smaller. Thus, companies would strive to minimise costs by reducing the amount of waste tonnes to be stripped thereby maximising the dollar value of a project.
Slope angles	The final slope angles have a net effect on the depth of the pit. Thus, if slope angles are steep, then the optimal pit gets deeper. However, this is heavily dependent on the surrounding rock properties. Weak rocks would need shallow angles resulting in shallow pits due to the possibility of slope failures, whereas competent rocks can tolerate steep angles since they tend to be stable.
Size of equipment	The size of equipment has significant influence on the size of the ramps that need to be developed. Larger equipment requires wider ramps to operate safely, thereby resulting in wider pushbacks which lead to larger pits.
Deposit geometry	Deposit geometry affects stripping ratio which is a function of the contact shape of the mineral deposit. The need for pushbacks depends on the cost of stripping. Generally, the cost for stripping a cut in tabular ore bodies is the same as the predecessor (provided the haul road distance to dump is constant), but for the near vertical ore bodies the cost of pushback increases from one cut to another (Songolo, 2010).
Ramps	Gallagher and Kear (2001) pointed out that choosing an optimal ramp design system that maximises NPV and practical to implement is one of the critical factors to consider for an optimal pit. Width, gradient and direction of these ramps need to be considered and evaluated to achieve an optimum pit. They further stated that the haul road width also has a significant effect on the final pit perimeter.

2.3.4 The effect of mining sequence on NPV

In general, the simplest method to mine a pit is to start with the top benches and proceed downwards. According to Snowden Mining Industry Consultants (2001), this is called worst case mining because it results in the lowest NPV. The advantage, however, is that it's always practical and produces a smaller pit. On the other hand, if each pit shell is mined one after the other with the related ore and waste mined at almost the same time, then the resultant NPV is higher and it this is called best-case mining. Its disadvantage is that it is very impractical. The third schedule is called the specified case. This case is mined by specifying the number of pushbacks required based on the mining constraints. The resultant NPV is determined by the amount of waste stripping requirements. Generally narrow pushbacks give better NPV.

2.3.5 Generation of Pit Shells using Whittle 4X

Generally, when value models of different pits are prepared, the resultant outlines after a 3D optimisation forms a set of nested pits. Thus, Whittle optimisation is done first for infinite price which results in a bigger pit. The next optimisation is for the lowest price which gives the smallest pit, and this allows for exclusion of all blocks within the two pits (smallest and biggest) for further consideration. Repeated optimisation for different prices between the lowest and highest price gives a series of nested pits as shown in Figure 2-4.

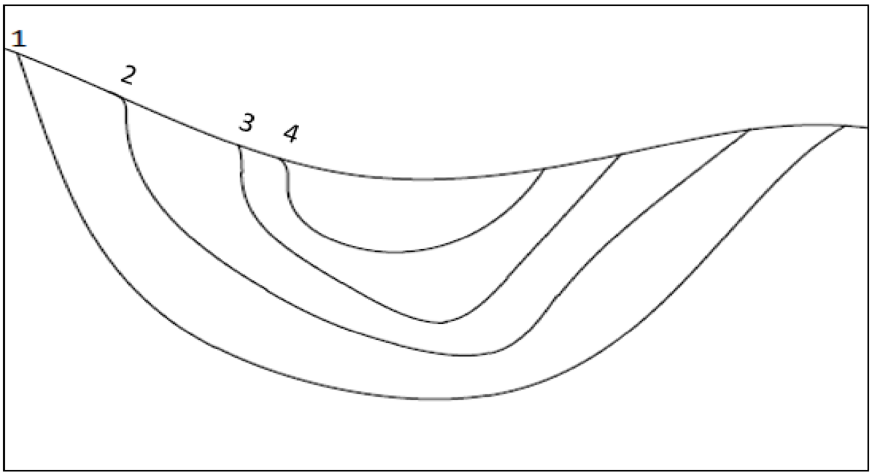


Figure 2-4: A sequence of pit shells (Snowden Mining Industry Consultants, 2001).

When using Whittle 4X for pit optimisation, revenue factors are applied since there is more than one product price being considered. These revenue factors are a ratio between incremental cost and incremental revenue. Thus, these revenue factors are used to generate different pit shells by varying the product price. Figure 2-5 shows a set of pit shells. These pit shells should be mined without breaking the slope constrains. This means ‘a’ is mined before ‘f’ is mined (Snowden Mining Industry Consultants, 2001). In South Africa, valuation inputs in determining the value of the mineral resource or reserve are constrained by guidelines set in The South African Code for The Reporting of Mineral Asset Valuation (SAMVAL Code).

Different mining jurisdictions use different codes, e.g. Australia use The Joint Ore Reserves Committee Code (JORC Code) and Canada use The Canadian Institute of Mining, Metallurgy and Petroleum on Valuation of Mineral Properties (CIMVAL). For the purpose of this report, the determination of an optimum K04 using Whittle was subjected to SAMVAL Code.

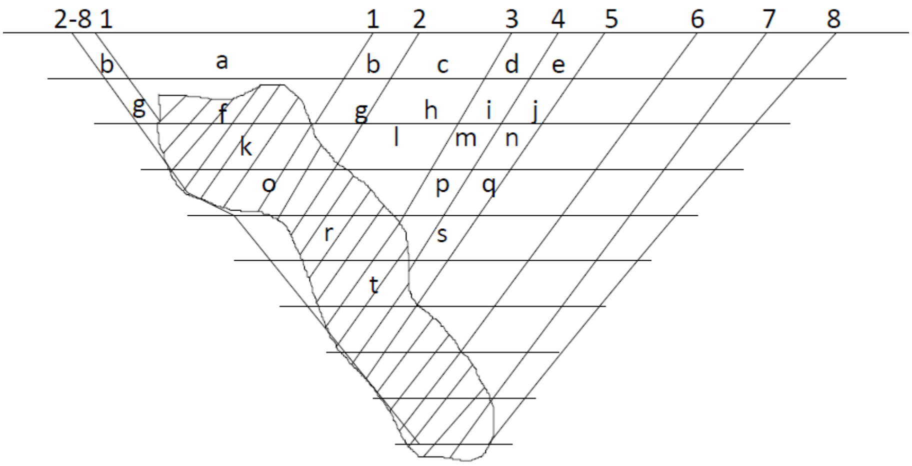


Figure 2-5: A Set of Pit Shells and Benches (Snowden Mining Industry Consultants, 2001).

2.4 Mine Valuation: SAMVAL Code

The South African Code for The Reporting of Mineral Asset Valuation (SAMVAL Code) stipulates minimum standards and guidelines for Reporting of Mineral Asset Valuation in South Africa.

The SAMVAL Code forms part of the South African Mineral Reporting Codes (SAMCODE) document and therefore relies upon the requirements of The South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC Code).

2.5 Mine Valuation Methods

The SAMVAL Code states that the CV shall apply at least two valuation approaches to assess the value of a Mineral Asset. The CV shall also justify where it is not possible to use more than one method. The three approaches are (SAMVAL, 2016: p9):

- ***"Income Approach, i.e.*** *the Income Approach relies on the 'value-in-use' principle and requires determination of the present value of future cash flows over the useful life of the Mineral Asset;*

- ***Market Approach, i.e.*** *the Market Approach relies on the 'willing buyer, willing seller' principle and requires that the monetary value obtainable from the sale of the Mineral Asset is determined as if in an arm's-length transaction.*

  The application of certain logic in Mineral Asset Valuation, such as 'gross in-situ value' simply determined from the product of the estimate of mineral content and commodity price(s), is considered unacceptable and inappropriate;

- ***Cost Approach, i.e.*** *the Cost Approach relies on historic and/or future amounts spent on the Mineral Asset. It is a valuation approach based on the economic principle that a buyer will pay no more for an asset than the cost to obtain an asset of equal utility, whether by purchase or by construction".*

The choice of a valuation method to be applied is entirely based on the decision and judgement of the CV and should be justifiable to their peers. The SAMVAL Code also states that the results from the valuation approaches and methods employed should be weighted and reconciled into a concluding opinion of value as shown in Table 2-3**.** The reasons for giving a higher weighting to one method or approach over another should also be stated and justified.

Table 2-3: Relationship between stages of development and valuation approaches for Mineral Assets (SAMVAL, 2016).

Valuation approach	Early stage exploration	Advanced stage exploration	Development properties	Production properties	Dormant properties		Defunct properties
					Economically viable	Economically not viable	
Income	Not generally used	Less widely used	Widely used	Widely used	Widely used	Not generally used	Not generally used
Market	Widely used	Widely used	Less widely used	Quite widely used	Quite widely used	Widely used	Widely used
Cost	Widely used	Widely used	Not generally used	Not generally used	Not generally used	Less widely used	Quite widely used

The value assigned to a mineral asset helps mining companies to make informed investment decisions as to whether to sell, develop the asset into production or raise money at the stock markets. This value can be determined through various methodologies and techniques and the choice of the technique to be used depends primarily at the stage at which the project has been developed (Lilford and Minnitt, 2005).

Table 2-3 shows the relationship between stages of project development and valuation techniques included in the SAMVAL code as a guide. However, SAMVAL (2016) states that the CV can choose the methodology to be used if it is justifiable to his/her peers.

Out of the three valuation approaches the income approach will be considered for the K04 project as it is already in production. The other two approaches are widely used for projects that are in the early stage or advanced exploration.

Lilford and Minnitt (2005) stated that the income approach relies on the valuer's determination of a cash flow from selling the product over the life of mine. Thus, free cash flow will be realised after deduction of taxes, royalties, working costs and capital expenditure (valuation variables). The discounted cashflow (DCF) is generally used to determine the net present value (NPV) and internal rate of return (IRR) of the project. Thereafter the sensitivity analysis is used to determine which valuation variables have a significant impact on the project.

2.5.1 Discounted Cash Flow Method

The DCF valuation technique is most commonly used valuation tool primarily because of its strengths over other techniques in the market and cost approaches. Lilford and Minnitt (2005) stated that the DCF method is used to produce numerous quantitative results and is based on the principle of future returns and cash flow for any initial investment. For this method to be applied effectively there should be some critical inputs which are incorporated into the LoM plan. These inputs include (Lilford and Minnitt, 2005):

- “compliant reserve and resource estimate;
- forecast tonnage profiles on a daily, monthly or annual basis;
- forecast grade profiles and associated recoveries. This, together with the tonnage profile, allows the valuer to calculate the volume of saleable product;
- estimated working costs, preferably unitized to either an amount per ton milled or an amount per unit of metal or product sold;
- forecast capital expenditure profiles over the life of the operation, including ongoing or sustainable capital expenditure; and
- rehabilitation liabilities or trust fund contribution, retrenchments costs, plant metal lock-up and any other specific factors that will impact on costs or revenue.”

Baurens (2010) stated that the most important factors considered in DCF analyses are the discount factor and the assumption of long-term prices. He further pointed out that other factors which need to be estimated as inputs to DCF are:

- Tonnage and grade of mineable reserves;
- Revenue (volume x price); and
- Production costs:
  - Operating costs;
  - Capital expenditure; and

– Taxes and royalties.

Table 2.4 shows a simplified DCF valuation spreadsheet. In practical terms, the value of money in the distant future is relatively small, thus it is not normally recommended to consider long life reserves in the valuation process.

Lilford and Minnitt (2005) stated that once all economic modifying factors have been applied and the DCF determined, the resultant cash flow should be used to derive the NPV at a pre-determined discount rate or a range of discount rates. This derived NPV will be used as basis for calculating the return on investment. They further alluded to the fact that inflation plays a role in any cash flow forecasting, however the effect of taxation should be considered and applied to the real taxable income.

Table 2-4: Simplified DCF Valuation Spreadsheet (Baurens, 2010).

	Year 0	**Year 1**	**Year 2**	**Year 3**	**Total**
Gold produced		200	250	280	730
Gold price		500	500	500	500
Sales Revenue		**100'000**	**125'000**	**140'000**	**365'000**
Less: Site Operating Cost		55'000	70'000	70'000	195'000
Refining		1'500	1'700	1'700	4'900
Operating Profit		**43'500**	**53'300**	**68'300**	**165'100**
Less: Income Tax		5'000	9'000	11'000	25'000
Capital Expenditure	80'000	640	1'600	9'600	91'840
Net Cash Flow	**-80'000**	**37'860**	**42'700**	**47'700**	**48'260**
Discount Factor (13%)		0.885	0.783	0.693	2
Net Present Value	**-80'000**	**33'504**	**33'440**	**33'058**	**20'003**
Total Net Present Value ($ millions)	**20'003**				

2.5.2 Net Present Value

NPV predicts the net income flows of an asset over its entire economic life. Thus, it involves forecasting future net revenues generated by a mineral resource if it were to be exploited optimally. This is done by discounting the revenue stream using an appropriate discount rate. NPV is used by investors or management as measure of a project's value and to make decisions in capital budgeting (Domingo and Lopez-Dee, 2007).

Domingo and Lopez-Dee (2007) also stated that NPV has four key advantages and these are listed below:

- **Time Value of Money:** NPV recognises the concept that a dollar earned today is worth more than a dollar earned five years from now.

- **Income Flows:** NPV calculates a resource's expected income flows and considers unique risks associated with the project. Using NPV helps to eliminate accounting inconsistencies since the income flows encompasses all the benefits not just the profits.

- **Risks:** NPV incorporates the risks associated with a resource via the expected income flows and/or discount rate.

- **Flexibility:** NPV provides flexibility and depth, since the NPV equation can adjust for inflation and be used with other analytical tools such as Scenario Analysis.

Regardless of its advantages, NPV has some limitations of its own which valuers need to be aware of, namely-(Domingo and Lopez-Dee, 2007).

- NPV has two main inputs which are income flow and cost of capital, with the former representing the forecasted net benefits during the lifetime of the resource. It is therefore difficult to determine the accurate (near-actual) income flow considering assumptions associated with the composition of future income stream.

- NPV calculation also relies on an appropriate discount rate, but there are several ways of doing this. Thus, it calls for the expertise of the valuer to choose the best discount rate.

2.5.3 Internal Rate of Return

Stojanović (2013) defined IRR as the rate that brings the present value of the project to zero. He stated that calculation of the internal rate of return is a complex procedure, performed by using an iterative procedure. i.e. the method of "trial and error". This is done by increasing or decreasing the value of the discount rate until it reaches the rate with which the net present value equals zero.

2.5.4 Sensitivity Analysis

In any project, uncertainties exist and these need to be evaluated to have a clear picture of the performance of the project if there are changes in variables. The process of addressing these “what if” concerns is known as sensitivity analysis (Tschabrun, 2005).

This involves evaluating the project’s economics by varying one parameter while others are kept constant. Typically, variable parameters are product prices, grade, operating costs, and capital cost. Results are plotted on spider diagrams where the slope of the curve is used to determine the sensitivity of the project to that parameter. Thus, the steeper the slope the more sensitive the project is to the specific parameter.

The whole purpose of the sensitivity analysis is to identify critical parameters whose changes will result in either a significant positive or negative impact on the project. This enables companies to closely monitor changes in these parameters. Madowe (2013) stated that sensitivity analysis also allows management to take opportunities that may exist to improve various parameters if significant increase in NPV results.

2.6 Chapter Summary

This chapter gave a brief overview of the K04 models namely; the geological model and the mineral resource model, a brief history of the mining was also discussed. The second section described pit optimization tools. Among the tools described, Whittle was chosen for K04 optimum pit selection because it is the tool used at De Beers.

The third section discussed the South African Valuation Code (SAMVAL) and the different mine valuation methodologies and techniques. Out the three valuation approaches outlined in the SAMVAL (income, market and cost approaches), the income approach was discussed based on the fact the project is already in the production phase.

The tools and techniques under income approach discussed are the DCF, NPV, IRR and sensitivity analysis. These tools would be applied in the valuation process.

3. INPUT PARAMETERS FOR OPTIMUM PIT SELECTION

3.1 Introduction

This chapter describes data collection for input parameters that were used for the optimum pit selection for K04 kimberlite pipe and the steps taken in conducting the K04 Whittle optimum pit selection. Modifying factors are fundamental in determining the economic viability of a mineral deposit. Table 3-1 highlights these input parameters and are explained in detail. In addition, this chapter outlines essential elements of Whittle optimum pit selection process as well as the statement of K04 mineral resource used in the Whittle model.

3.2 Mine Planning Input Parameters

The mine planning process begins with the creation of a mine planning input parameters document (Madowe, 2013). It details inputs from other mine departments, and these include; mining, engineering and ore processing. A typical input document contains parameters and descriptions such as detailed in Table 3-1 and Table 3-2 below.

Table 3-1: Summary of Mine Planning Input Parameters.

Parameter	Unit	Description
Geotechnical parameters	degrees and metres	Slope angles and bench height define geotechnical parameters. These factors are determined based on geotechnical characteristics of each rock type.
Resource modifying factors	%	Factors include resource carat ratio, dilution and mining loss
Mining costs	ZAR/tonne	Include drilling, blasting hauling & dumping costs.
Ore treatment costs	ZAR/tonne	Different rock types, hardness of the rock and dilution affect processing costs.
Royalties	%	This is a levy paid to government as a percentage of revenue
Diamond prices	USD/carat	This is the value of each carat sold.
Overhead costs	ZAR/tonne	Administration, human resources, security and technical support form part of overhead costs
Stay in business capital costs	ZAR	Replacement of mining equipment is depended on amount budgeted for capital costs.
Discount factor	%	These are factors that determine the present value of a cash flow such as expected rate of return.

Table 3-2: Summary of other Mine Planning Input Parameters.

Parameter	Unit	Description
Shovel rates	t/h	It is the amount of tonnages to be loaded onto a dump truck by a shovel per hour.
Haul distances	km	These are distances from where the ore or waste is loaded to where they are dumped. (crusher, stockpile or dump
Haul road width	m	This determines the size of equipment that can be operated and whether two-way traffic can be utilised.

3.3 Data Collection

The primary data used for this research was obtained from the Venetia Mine's Technical Services (TS) department. The data included the following; K04 geological model, K04 mineral resource estimate, K04 Whittle model and Whittle input parameters and is summarised in section 3.7

3.3.1 Geotechnical Parameters

The country rock at Venetia is part of the Limpopo Mobile Belt and consists mainly of metamorphic and intrusive igneous rocks. The rock in the immediate vicinity is part of the highly deformed Beit Bridge Complex and does not have a clearly defined stratigraphy. Structures have been mapped at Venetia Mine (see Figure 3-1) and these are mainly ductile structures (foliation) and brittle structures (shear zones and joints) (Basson, Creus and Stock, 2015).

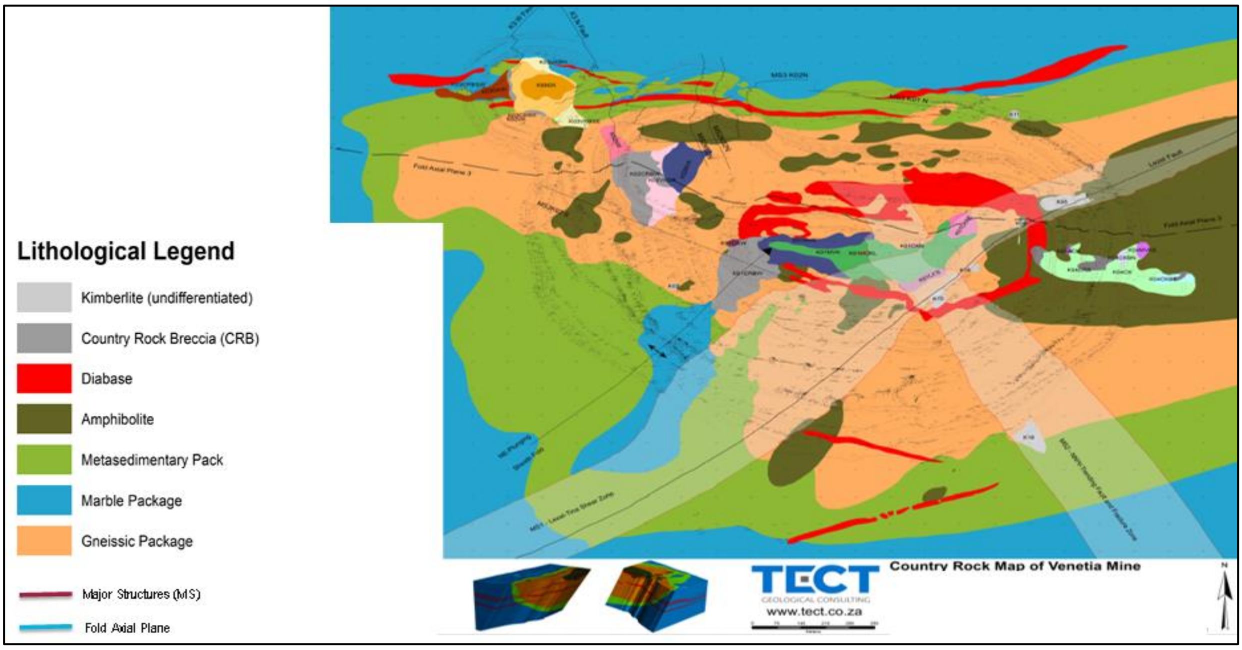


Figure 3-1: Detailed Bench Scale Geology Map of the Country Rock at Venetia Mine showing the major structures (Basson, Creus and Stock, 2015).

The pit is further divided into five geotechnical domains with similar geological, structural and geotechnical properties (see Figure 3-2). Table 3-3 details the failure mode of each domain. The stability of each domain is controlled by the structural geology and as a result, the stack angle designs are highly dependent on the orientation of the pit slopes. Table 3-3 shows brief descriptions of each domain.

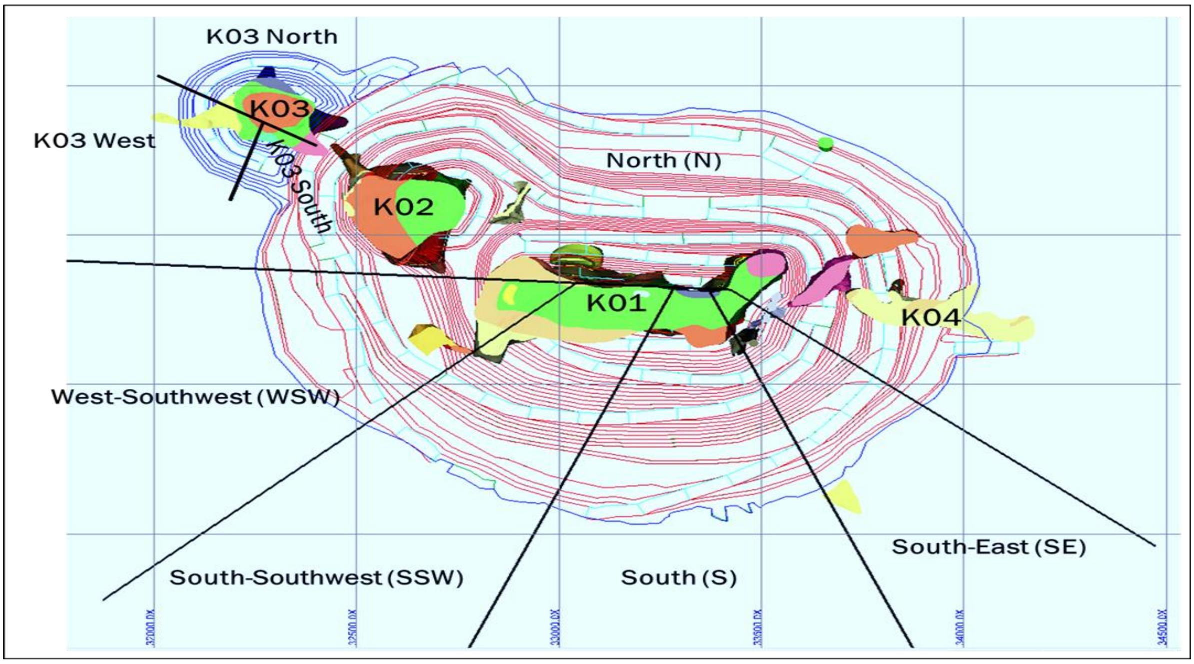


Figure 3-2: Geotechnical domains for Venetia Mine.

Table 3-3: Shows Geotechnical Domain Failure Mode (Venetia, 2018).

Domain	Description	Failure mode
N	Foliation is steeply dipping into the face. In the east of this domain, the folding and foliation angles are oblique to the pit geometry.	The main failure mechanisms in this domain are toppling in wedge type failures.
SE	This is a transitional zone between the South domain and North domain.	Wedge failures are the dominant failure mechanism.
S	In this domain, the foliation and the pit wall are parallel to each other. Foliation is dipping out of the face and shallower than the north domain foliation.	The dominant failure modes are planar, wedge and step-path type failures along the foliation.
SSW	This domain is characterised by both a change in rock type and slightly more favourable pit wall geometry. This domain is within the metasediments.	Planar failure is still the dominant failure mechanism.
WSW	This is a transitional zone between the North and South-Southwest domains.	Planar and Wedge failures are the dominant failure mechanisms.

3.3.2 Slope Angles

The stack angles for each of the above domains are based on acceptable level of Factor of Safety (FOS) and consist of bench, inter-ramp and overall slope scale analyses. Haul roads and ramps are also considered when deciding on the final slope angles. Incorporation of ramps results in shallowing of slope angles and increases the stripping ratio as more waste is mined to accommodate ramps and haul roads. Values adopted are from studies conducted by Steffans, Roberts and Kirsten (SRK) consultants (SRK Consulting, 2008). Table 3-4 shows overall slope angles used for optimum pit selection.

Table 3-4: Geotechnical Domains and Associated Slope Angles.

Parameter	Domain	Stack Angle	Bench height (m)
Geotechnical Parameters	Domain N	43^0	12
	Domain SE	37^0	12
	Domain S	36^0	12
	Domain SSW	36^0	12
	Domain WSW	44^0	12

3.4 Mineral Resource Modifying Factors

The mineral resource estimate is vital in the process of estimating an ore reserve. However, other input parameters also play a crucial role in modifying the mineral resource to reserve and these parameters are referred to as modifying factors in the SAMREC code.

The resource to reserve modifying factors considered for K04 pit optimum pit selection are the following: Mine Call Factor (MCF) or Resource Carat Ratio (RsCr), dilution, and mining losses. This section details how these factors were derived.

3.4.1 Mine Call Factor (MCF) or Resource Carat Ratio (RsCr)

Tetteh and Cawood (2014) defines MCF as the ratio, expressed as a percentage, which the specific product accounted for in recovery plus residue bears to the corresponding product called for by the mine's measuring methods. The ratio can be expressed as follows (*ibid*):

$$\mathrm{MCF} = \left(\frac{Metal\ produced\ in\ recovery\ plus\ residue}{Metal\ called\ for\ by\ mines\ evaluation\ methods}\right) \times 100\%\ (Equation\ 1)$$

A 100% MCF means that all the variables in the mine value chain which includes, sampling, assaying, and tonnage measurements in a mine and plant are perfect and that there is no mineral lost at any stage during handling and processing. Thus, a MCF is a measure of mine value chain processes efficiency (Tetteh and Cawood, 2014). In diamond mining, the MCF is referred to as RsCr and can be expressed as follows (Venetia, 2010):

$$RsCr = \left(\frac{Actual\ carats\ recovered}{Survey\ called\ for\ carats}\right) \times 100\%\ (Equation\ 2)$$

The historical RsCr performance for K04 from treating satellite stockpile material has been used to compute a resource factor that can be applied in the resource to reserve convention. A 95% RsCr was computed for use in the optimum pit selection for K04 in line with the mine's Strategic Business Plan (SBP) standard.

3.4.2 Dilution

Dilution is the mixing of waste material and mineralized ore during mining. This results in an increase in tonnage and a decrease in recovered grade when reconciled with estimated mineral resource grade.

It can also be defined as the ratio of the tonnage of waste mined and sent to the plant or stockpile to the total tonnage of ore and waste that are processed or stockpiled. The equation below expresses this ratio (Ebrahimi, 2013).

$$\text{Dilution} = \left(\frac{Waste\ tonnes}{Ore\ tonnes + Waste\ tonnes}\right) * 100\ (Equation\ 3)$$

Dilution results in increased overall mining cost, which reduces profits. Below are some of the ways in which dilution affects operating costs (Ebrahimi, 2013):

- It adds to drilling, mining, crushing and treatment costs for which there are no financial benefits or economic return,
- It adds indirect costs since the non-ore material has different properties to ore and may adversely affect recoveries and reduce the grade of the ore,
- It results in lost opportunities since resources are directed at handling waste as opposed to ore for the plant, and
- It displaces treatable ore, resulting in high yields because of heavy minerals such as garnets from waste rock that impact on the throughput rates and contributes very little to useful production of the final product.

In general, there are two categories of dilution and these are internal dilution and external dilution. The internal dilution refers to waste rock or crustal rocks that are found inside the ore. These were incorporated inside the ore body during mineralisation and emplacement. The rocks are distributed randomly across the ore body and this makes them difficult to model, predict or delineate (Ebrahimi, 2013). If the waste rocks such as dykes are modelled and of known location, these could be excluded during blasting and as such will not form part of internal dilution.

The external dilution arises when waste material from outside ore boundaries is introduced into the ore material. Several factors within the mining value chain can influence and contribute to external dilution (Ebrahimi, 2013):

- Uncontrolled ripping of stockpile floors leading to the loading of non-ore material, hence dilution,
- Poor cleaning practices where excessive spillage from top benches can contaminate lower ore material, and
- Poor loading practices of kimberlite ore along the ore-waste contact zone.

The above factors result in insignificant dilution. Controls such as grade control can minimise this type of dilution. However, greater levels of external dilution come from design and sequencing and drilling and blasting (Ebrahimi, 2013).

3.4.3 Dilution Due to Design and Sequencing

This form of dilution occurs where the required minimum extraction width or minimum operating width extends into waste rock, resulting in more waste being mined (Mubita, 2005). It can also result from instances where slivers of ore left behind in the previous depletions are required to be mined simultaneously with waste pushbacks. In areas where these slivers are not wide enough and where they cannot be drilled separately, they will be mined with pushback waste and resulting in highly diluted ore.

3.4.4 Dilution Due to Drill and Blast Activities

Dilution from drill and blasting happens when drilling through areas where ore-waste contacts are not well defined and where blasting is done close to ore-waste rock contact. This results in the over-break of highwalls and introduction of waste into ore muck piles. Mokoatle (2011) quantified and modelled the dilution from the over-breaks of waste rock highwalls using the skin analysis method in GEMCOM.

Three scenarios, 1 m, 2.8 m and 3.5 m of blast back break were considered (see Figure 3-3). Volumes of extra waste generated per back-break were calculated to compute the external dilution per back-break for K01 (see Table 3-5). A 5% external dilution was assigned to K04 due to anticipated contamination from the stripping of overburden because it is not fully exposed since it is not part of the bigger pit.

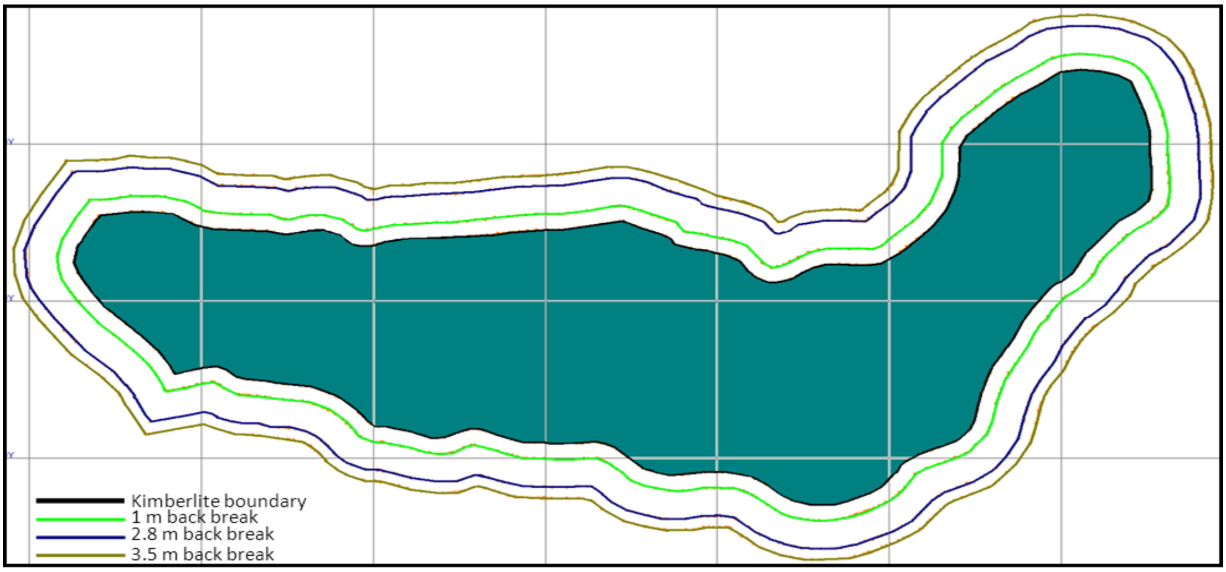


Figure 3-3: Plan-view illustrating the external waste modelling based on blast back break from the ore waste contact (OWC) (Mokoatle, 2011).

Table 3-5: K01 External Dilution per Blast Back Break (Mokoatle, 2011).

			Dilution % by volume		
Pipe	Bench	Volume	1m_Back Break	2.8m_Back Break	3.5m_Back Break
K01	27	874,900	2.25%	6.19%	7.72%
	28	865,101	2.15%	6.16%	7.74%
	29	862,174	2.06%	6.12%	7.67%
	30	861,221	2.18%	6.19%	7.70%
	31	862,315	2.10%	6.11%	7.63%
	32	868,236	2.10%	6.07%	7.54%
	33	868,467	2.07%	6.09%	7.61%
	34	873,903	2.13%	6.10%	7.63%
	35	877,293	2.26%	6.18%	7.77%
	36	872,897	2.09%	6.22%	7.67%
	37	870,673	2.14%	6.18%	7.78%
	38	865,665	2.29%	6.42%	7.97%
	39	858,962	2.17%	6.32%	7.95%
	40	845,369	2.44%	6.54%	8.23%
	Total	12,127,176	2.17%	6.21%	7.76%

3.4.5 Mining Loss

Ore loss refers to any unrecoverable economic ore left inside a stope (broken or not properly blasted at the boundaries), or to any valuable ore not recovered by the mineral processing system (Xingwana, 2016). This concept can be extended to surface mining where ore loss could be due to high dilution (both internal and external), ore wrongly consigned to waste dump or ore left behind as slivers. The overall effect of ore loss is a reduced MCF. Thus, mining loss is expressed as follows (Venetia, 2010):

$$Mining\ loss = \left(\frac{ore\ to\ waste\ dump + mining\ recovery}{Total\ ore\ in\ LOM\ plan}\right)\ (Equation\ 4)$$

Where:

- *Ore to waste dump is ore delivered to the waste dump*
- *Mining recovery is the percentage of the recovered broken material to the total broken material per mining polygon*
- *Total ore in LoM plan is the total ore tonnes planned for the LoM*

Mining loss can arise when mined kimberlite is extremely diluted and instead of being sent to the treatment plant, it is consigned to the waste dump. Ore with dilution of more than 90% will be taken to waste dump as this cannot be economically treated through the plant.

Mining loss due to improper blasting can occur when the economic ore included in the business plan is unrecoverable and left inside the pit due to improper blasting, insufficient blast breakage on the contacts, or improper cleaning (free digging) of the face (Xingwana, 2016). This form of ore loss has not been quantified but is assumed to be insignificant for business planning purposes.

Table 3-6 summarizes the resource to reserve modifying factors applied when optimising K04. A 95% resource carat ratio shows the efficiency of the diamond recovery in the plant. 5% external waste is expected during mining whilst 3% of the ore will be lost due to inefficiency in mining processes such as loading as per Venetia (2010).

Table 3-6: Resource to Reserve Modifying Factors.

Resource Parameter	Kimberlite Pipe	Input Parameter
Resource carat ratio	K04	95%
External dilution	K04	5%
Mining loss	K04	3%

Table 3-7 shows the K04 mineral resource model input parameters. The geological model was modelled by Farrow and Nordin (2005) and classified internally the same year to an inferred level of confidence. The 2012 re-estimate was conducted in order bring the resolution of the local block estimate in line with the estimates of the other resources of 25 m block size, as opposed to the previous 50 m block size (Mushongahande, 2012).

Table 3-7: Resource Models and Mineral Resource Estimates.

Resource Model Input Parameter	Description
Geology and Volume Models	K04 Geology and Volume Model; Farrow and Nordin (2005)
Grade and Density Models	K04 Resource Estimate by Z-Star; Lohrentz (2012)
Classification Score Cards	K04 Classified in 2005

3.5 Financial Parameters

Financial parameters used in the K04 Whittle optimum pit selection include; mining costs, processing costs, diamond prices, exchange rates, royalties, discount factors, overhead costs and stay in business capital costs. These parameters are described below.

3.5.1 Mining Costs

In open pit mining, mining (excavation) cost is described as the average cost of producing a tonne of ore, taking into account waste stripping (Thompson, 2005). Venetia Mine has a relatively high waste stripping ratio of 5.2:1 and this has a huge impact on the overall cost per tonne. Cost drivers for mining costs include drilling, blasting, loading and hauling. Figure 3-4 shows typical mining cost distribution for open pit mines.

The overall cost of mining is R34.68 per tonne (ore and waste combined). A mining cost adjustment factor (MCAF) is applied in Whittle to compensate for differences in elevation and dumping distances during mining. Cost drivers per section namely; drilling, blasting, loading and hauling costs, are described by Venetia (2010) as:

- **Drilling costs:**
  - Diesel: - burn rate is based on equipment type being used (drill equipment sizes differentiate between P&H 251, Pit Viper, DM30 and D60).
  - Repair and Maintenance: - drills have specifically calculated LCC rates that are usually divided into 5 000 hour brackets; and payment is done based on the drill hours burned multiplied by the specific LCC rate that the shovel is currently in.
  - Drill string/consumable costs: - costs are based on actual replacement cost of a drill string/consumable; and drill string/drill consumables are replaced either when it reaches a specific hour life or when any unwanted fractures are noticed that can negatively impact on the safety and integrity of the equipment.
  - Labour: - labour cost to operate the drill (including a relief operator, overtime, bonus and fringe benefit costs).

- **Blasting costs:**
  - Powder factor: - based on the type of hole being charged (ore/waste) and further split between trim, production or pre-split and the varying depth of hole being charged.
  - Explosive accessories: - linked to the type of hole being blasted.
  - Labour: - labour cost to charge the holes (including a relief operator, overtime, bonus and fringe benefit costs).

- **Loading costs:**
  - Diesel: - burn rate is based on equipment type being used (shovel/loader sizes differentiate between CAT 6060 and CAT 994).
  - Repair and Maintenance: - shovels have specifically calculated LCC rates that are usually divided into 5 000 hour brackets, and payment is done based on the shovel hours burned multiplied by the specific LCC rate that the shovel is currently in.
  - Bucket costs: - costs are based on actual replacement cost of a bucket; and bucket is replaced either when it reaches a specific hour life or when any unwanted fractures are noticed that can negatively impact on the safety and integrity of the bucket.

- **Hauling costs:**
  - Diesel: - depth of pit influences diesel consumption since a deeper pit will require a higher burn rate to propel the truck because it is driving at an incline
  - Repair and Maintenance: -trucks have specifically calculated Life Cycle Costs rates (LCC) that are usually divided into 5 000 hour brackets; and payment is done based on the truck hours burned multiplied by the specific LCC rate that the truck is currently in.
  - Tyre costs: - tyre cost is based on actual replacement cost of a tyre; and tyres are replaced either when they reach a specific hour life or when any

unwanted cuts are noticed on the tyre that can negatively impact on the safety and integrity of the tyre.

– Labour: - labour cost to operate the truck (including a relief operator, overtime, bonus and fringe benefit costs).

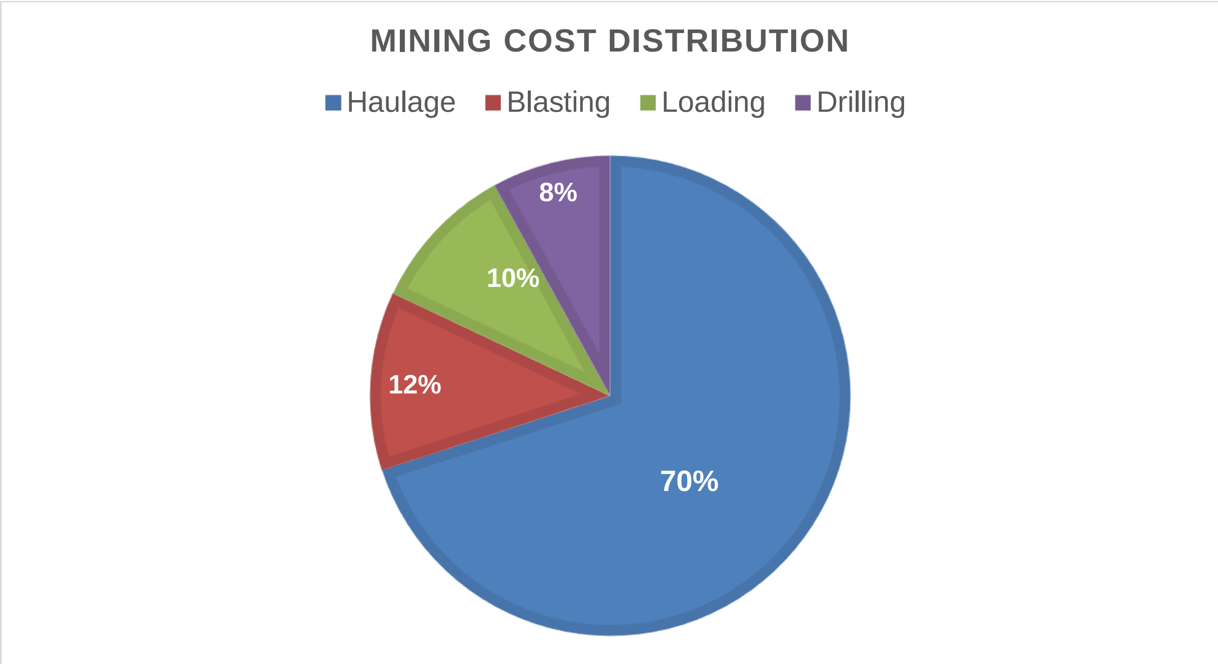


Figure 3-4: Mining cost drivers for open pit mines (Mohutsiwa and Musingwini, 2015).

3.5.2 Ore Treatment Costs

The processing of the ore at Venetia Mine is done by De Beers and costs are largely driven by power and other consumables such as ferrosilicon for processing and flocculent for water recovery. The breakdown of the processing cost per element is shown in Figure 3-5. The overall processing cost is pegged at R240.00 per tonne of ore treated; this is inclusive of overhead costs summarised in Table 3-9.

The ore treatment cost drivers include electricity, treatment chemical agents (ferrosilicon, flocculent, coagulant and caustic soda) and services. These are explained briefly below:

- **Electricity:**
  - Electricity consumption per tonne is based on the type of ore being treated in the plant multiplied by the rate/kWh that is paid to Eskom.
- **Ferrosilicon, Flocculent, Coagulant and Caustic Soda:**
  - All the consumption per ore tonne treated is based on the type of ore being treated in the plant multiplied by the cost / kg that is paid for the product.
- **Services:**
  - All costs to do required maintenance and services on the plant equipment. These costs include contractor costs for services, general planned maintenance and emergency / unplanned maintenance.
  -

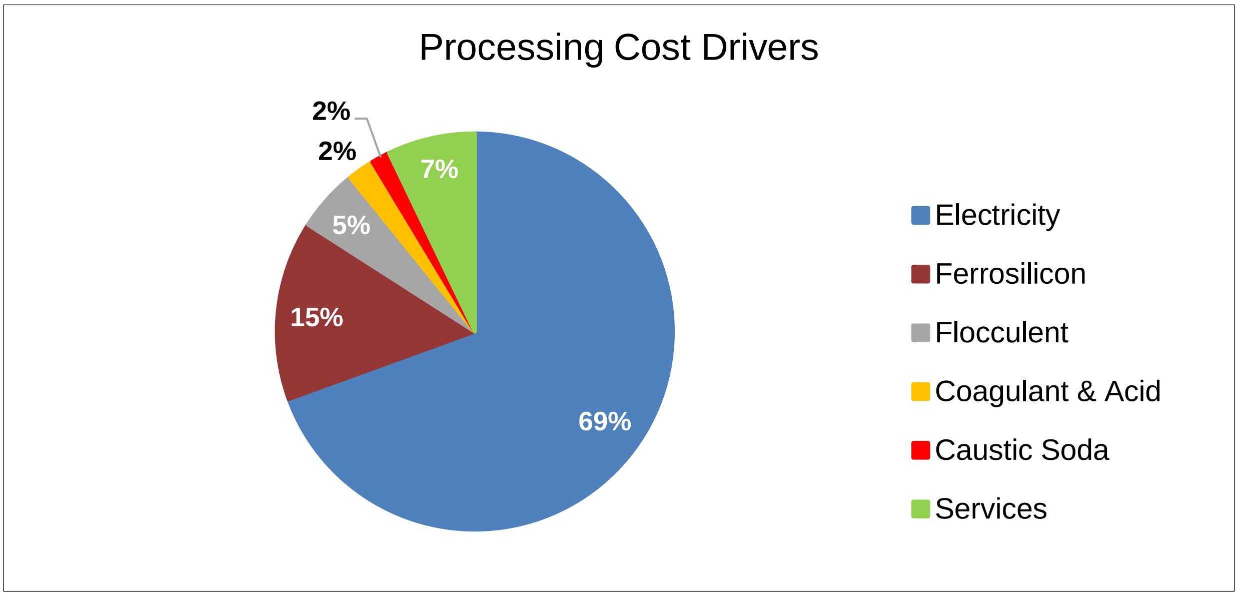


Figure 3-5: Processing cost drivers (Venetia Mine, 2010).

3.5.3 Royalty and Other Taxes

Royalties are payable on a mineral resource extracted within South Africa (National Treasury, 2008). The Mineral and Petroleum Resources Royalty Act (2008) classifies mineral resources as refined and unrefined and are listed in schedule 1 and schedule 2 respectively of the Act. Diamonds are listed in schedule 2 of Royalty Act (2008) and therefore are classified as an unrefined mineral resource.

Royalties are capped and cannot exceed 5% for refined mineral resource and 7% for unrefined mineral resource (Mineral and Petroleum Resources Royalty Act, 2008). The royalty tax formula is calculated as a percentage of EBIT (earnings before interest and taxes) over gross sales. The formula contains four variables, and these are;

- the intercept term, 0.5 (%), which acts as a minimum charge to ensure that the Government always receives some level of royalty payments for the permanent loss of non-renewable resources;
- earnings before interest and taxes ("EBIT")
- gross sales;
- constants of either 9 or 12.5, depending on whether the mineral resource is refined or unrefined

The royalty formula for unrefined mineral resources is as follows (*ibid*).
0.5 + [EBIT / (gross sales in respect of unrefined mineral resources x 9)] x 100.

3.5.4 Overhead Costs

Overhead costs refer to costs or expenses incurred in running a business that are not linked to production. The mining and processing overhead costs consist of labour (not specific to the mining or plant activity); repair and maintenance (R&M) fixed charges, planned maintenance and general overhead costs (i.e. personal protective equipment, stationery, travel, general spares, etc.).

General overhead costs consist of the support department's costs (i.e. security costs, commercial service costs, human resource costs and engineering support services costs). The activities, which drive the support service cost, are labour, transporting of employees, cleaning services, building maintenance, vehicle maintenance and DBGS support costs.

3.5.5 Stay in Business Capital Costs

Mining stay in business capital cost refers to a cost capitalised to the balance sheet to enable the business to continue to operate. In case of K04, this cost amount to R100 000 000.00 and includes the acquisition of equipment listed in Table 3-8.

Table 3-8: Breakdown of Stay in Business Costs

Equipment	Rate (ZAR)-Millions	Quantity	Total (ZAR)-Millions
Excavator	50.00	1.00	50.00
ADTs (30 t)	5.50	5.00	27.50
Dozer-D8	4.00	1.00	4.00
Grader 12H	3.00	1.00	3.00
Diesel Bowser	2.00	1.00	2.00
Water Truck	2.00	1.00	2.00
LDVs	0.50	2.00	1.00
Minor Assets	1.41	1.00	1.41
Escalation (10%)	90.91	0.10	9.09
Total			**100.00**

3.5.6 Discount Rates

Discount rates are normally applied to determine the overall NPV based on the level of risk posed by the deposit or project. These, however, vary from organisation to organisation.

A minimum of 8% is applied to on-going projects or brownfields, 10% is applied to expansion projects where the risk is well understood while 12% is applied to greenfield projects where risk is considered high. Thus, given the amount of work done to understand K04, a 10% discount factor will be applied in the optimum pit selection process (Venetia, 2010).

3.6 Other parameters

3.6.1 Haul Distances and Road Designs

Designs for in pit haul roads are driven by the size of the fleet. The largest mining vehicles at the Venetia Mine are the 793 CAT trucks, which are 8.4 m wide. To operate a double lane ramp system requires at least the size of truck between the lanes to create enough space between the trucks and at the same time maintaining safe distance from the high wall. A safety berm about half the tyre size of the truck is mandatory on the open edge to prevent trucks from driving off the road surface and falling over the edge. Figure 3-6 shows a typical haul road design at Venetia Mine and the width of the double lane ramps is 37 m (Venetia, 2010).

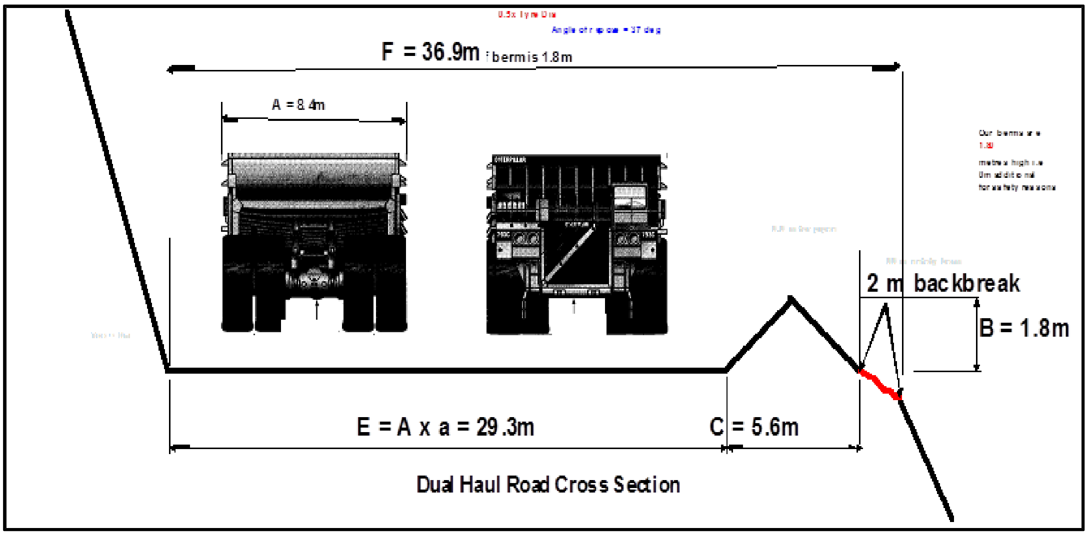


Figure 3-6: A typical haul road design for a 793 CAT truck (Venetia Mine, 2010).

3.6.2 Diamond Price

Diamond prices are largely driven by supply, demand and quality of the product. According to Bain and Company (2016), global supply of rough diamonds is expected to decline on average by 1% to 2% per year from 2016 to 2030 because of the aging and depletion of existing mines and the relatively small amount of new supply coming online. This situation will have a positive impact on the price of the diamonds in the market. In the case of K04 the base price will be ZAR1,047.98 per carat as per De Beers' internal pricing policy.

3.6.3 Exchange Rate Assumptions

The ZAR: USD exchange rate plays a crucial role in overall revenues from diamond sales. This is because the diamonds are sold in US dollars, thus negative or positive changes in exchange rate will impact revenues. According to Investing.com (2018) the Rand gained about 19% of its value to the US dollar in the first half of 2018, thus it moved to about ZAR12.1 from ZAR14.4. Based on this, an exchange rate of ZAR 12/1 USD of was used for the optimum pit selection.

3.7 Summary of inputs

Table 3-9 shows the summary of modifying factors. These factors used in the base case K04 Whittle pit optimum pit selection.

Table 3-9: Summary of Input Parameters.

Parameter	Description	Input	
		Stack Angle	Bench Height (m)
Geotechnical Parameters	Domain N	43º	12
	Domain SE	37º	12
	Domain S	36º	12
	Domain SSW	36º	12
	Domain WSW	44º	12
Resource Modifying Parameters			
	Resource carat ratio	95%	
	External dilution	5%	
	Mining loss	3%	
Financial Parameters			
	Mining costs (ZAR/t)	34.68	
	Overall processing costs (ZAR/t)	240	
	Overheads costs (ZAR/t)	116.7	
	Stay in business capital (ZAR)	100 000 000	
	Exchange Rate (ZAR: USD)	12	
	Selling cost (R/Ct)	1047.98	
	Discount factor	10%	

3.8 Mining Risks

The success of any mining project depends on a combination of many factors. Park and Matunhire (2011) identified three risk categories that influence the amount of revenues mining projects generate. These are categorised as technical risks, economic risks and political risks. Understanding of these risks is critical when making an informed investment decision.

3.8.1 Technical Risks

Park and Matunhire (2011) stated that technical risks are divided into three forms namely reserve risk, completion risk and production risk. Reserve risk is influenced by type of mineral deposit and level of confidence in mineral reserve estimates. Thus, a robust understanding of the deposit is imperative to estimate accurately the distribution, grade, and tonnage contained in reserve estimates. This is to enable the reserve performance to be close to the reserve estimate.

Completion risk reflects the possibility that a mineral development project will not make it into production as anticipated because of overspending, construction delays, or engineering or design flaws. Lastly production risk reflects the possibility that production will not proceed as expected because of production fluctuations caused either by extraction technical problems, pit wall stability issues or product extraction issues.

In conducting this research, it was assumed that there is continuity in the modelled mineral resource from 0 down to 240 mbgl and that the inferred grade is applicable.
On the production side, it was assumed that the treatability of K04 ore is well understood by the mine and that the proposed recovery factors of the ore are not over or under estimated.

3.8.2 Economic Risks

There are three risks that are collectively referred to as economic risks, namely price, supply and demand, and foreign exchange. Price risk involves the volatility of future product prices. Normally mineral prices are determined by supply and demand, thus forecasted future cash flows will change depending the supply and demand of the product at a particular time in the life of the project (Park and Matunhire, 2011).

Demand and supply risks entails the demand of the product at a particular time of a project. This is largely driven by prevailing economic conditions with the recession in 2008 resulting in low demand and low prices. This caused many mining companies to scale down and some closing operations completely (Park and Matunhire, 2011).

The foreign exchange risk relates to the operation of international markets where the exchange rate of a country's currency to the US dollar plays a crucial role in determining the revenues of mined products (Park and Matunhire, 2011). Product prices and volumes determine revenue earned. In case of diamonds, prices are determined by long term supply/demand fundamentals. According to Kimberley Process Certification Scheme (2018), estimated worldwide rough diamond production in 2017 was approximately 150.9 million carats with a total estimated value of approximately USD14.1 billion.

Although price volatility is likely in the short to medium term, the diamond market supply and demand dynamics are supportive of the long-term rough diamond value growth given the combination of positive consumer demand and growth above production growth levels as shown in Figure 3-7 (Bain & Company, 2016). The combined upside of economic risk factors may bolster a decision to mine K04.

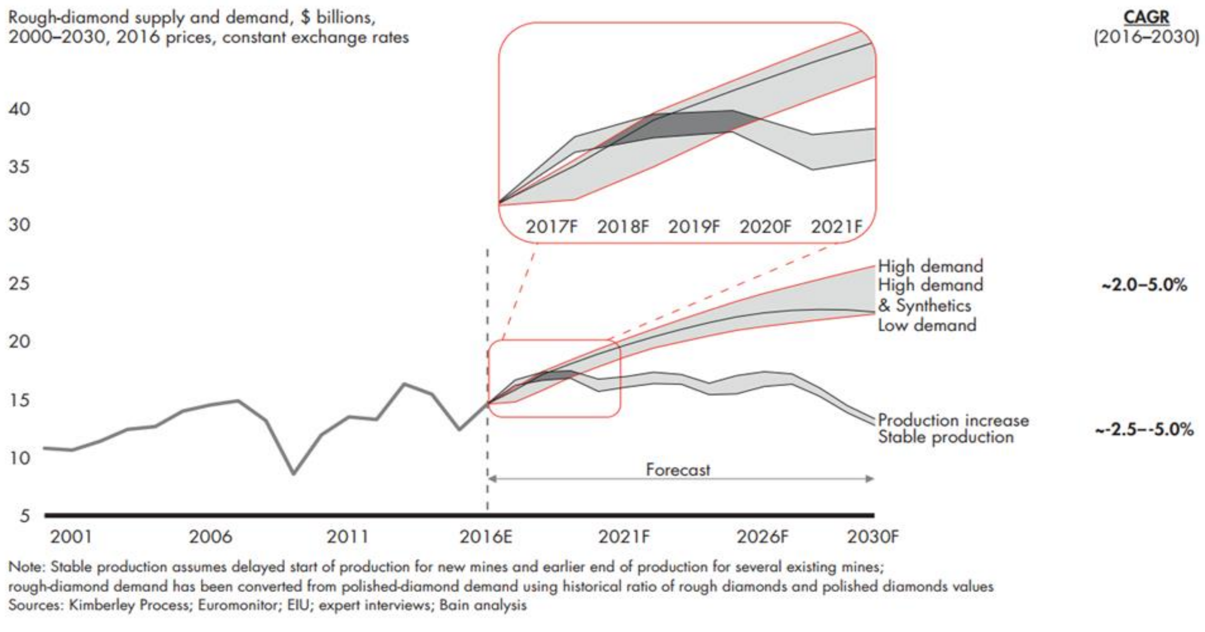


Figure 3-7: Rough diamond value growth (Bain & Company, 2016).

3.8.3 Political risks

The level of political risk is largely determined by the action of the sitting government and politicians. These actions can translate into changes in government policies and regulations relating to taxation and monetary policies, among others. The political risk for mining projects can be divided into four forms, namely, currency convertibility, environmental, tax regime and resource nationalism risks (Park and Matunhire, 2011). Currency convertibility affects freedom of capital movement and transfers while environmental regulations often increases costs by requiring technologies that are environmentally friendly. Taxes are a constant threat because even though companies have tax knowledge of countries they operate in, there is still a risk as these taxes can be reviewed anytime by the host governments.

In the case of South Africa, the tax regime looks stable and will not likely change in the near future. The tax regime stability will have a positive impact in investments. The resource nationalism is unlikely because all mineral resources in South Africa belong to the state.

The biggest risks are currency fluctuations and environmental. The ZAR has not been stable against the USD and this has positive or negative impacts to mining companies when doing business outside the country. For example, if the ZAR weakens against the USD, then it will be costly to import spares and other consumables used in the mines. The recently introduced carbon tax is split into two phases, with the first phase running from June 2019 to December 2022. The second phase will be implemented from 2023 to 2030. According to Minerals Council South Africa (2019), due to uncertainty associated with phase two, the carbon tax will cost mining companies an approximate ZAR5.5 billion per year. These estimated costs will materially affect mining business in South Africa.

3.9 Whittle Optimum Pit Selection Flowchart

Figure 3-8 shows a flowchart of the work done in K04 optimum pit selection exercise. Preparation of inputs parameter file was described in detail earlier on in section 3.1 and a summary of these is shown in Table 3-9**.** Whittle model creation, reconciliation and pit shell generation are discussed in subsequent sections. Analysis of results and selection of optimum pit will be discussed in the next chapter.

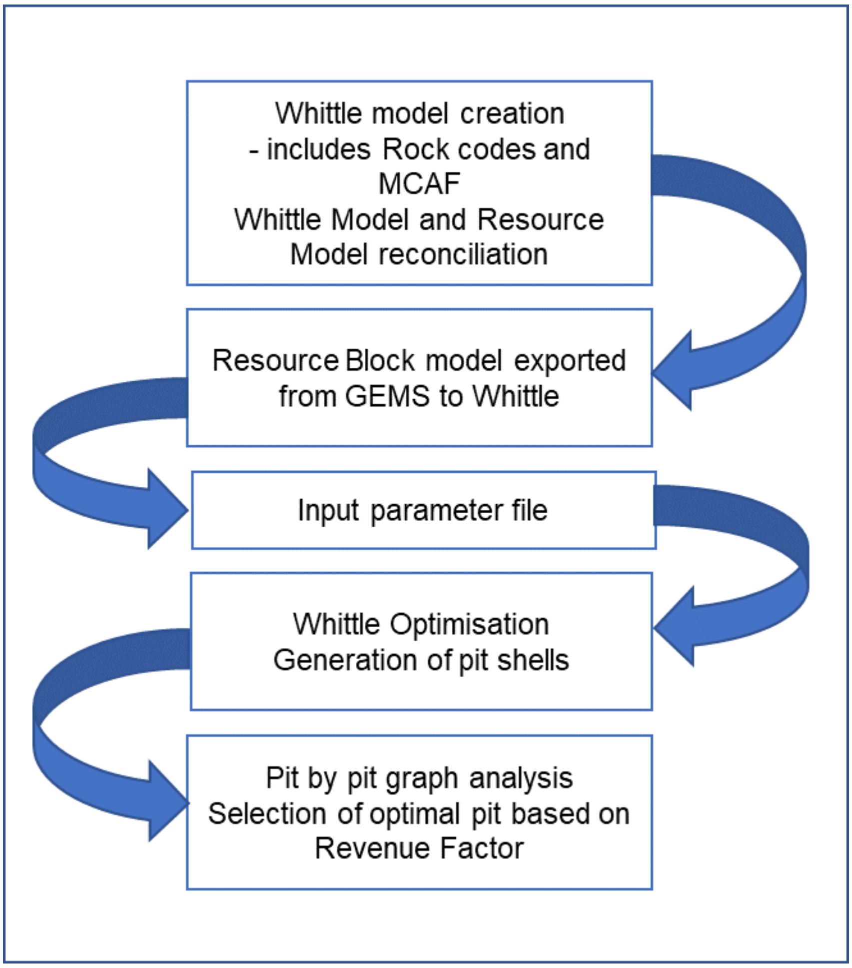


Figure 3-8: Whittle optimum pit selection flowchart.

3.10 Whittle Model Generation

The Whittle K04 block model used in the K04 optimum pit selection was created by Molatana (2015). The model was linked to the MinRASA model. MinRAS is a De Beers in-house data base system used for ore resource updates (ORU) and volume depletions. This data base stores ore reserve and mineral resource information and has an official model and a dynamic model that changes as mining progresses.

Furthermore, MinRAS has been integrated with Gemcom system where the latter has a Read Only access to MinRAS. The purpose of Gemcom is to calculate the volume change per block then compares it to the in-situ volume in MinRAS. The basic process flow is shown in Figure 3-9**.** Figure 3-10 shows the grade elements.

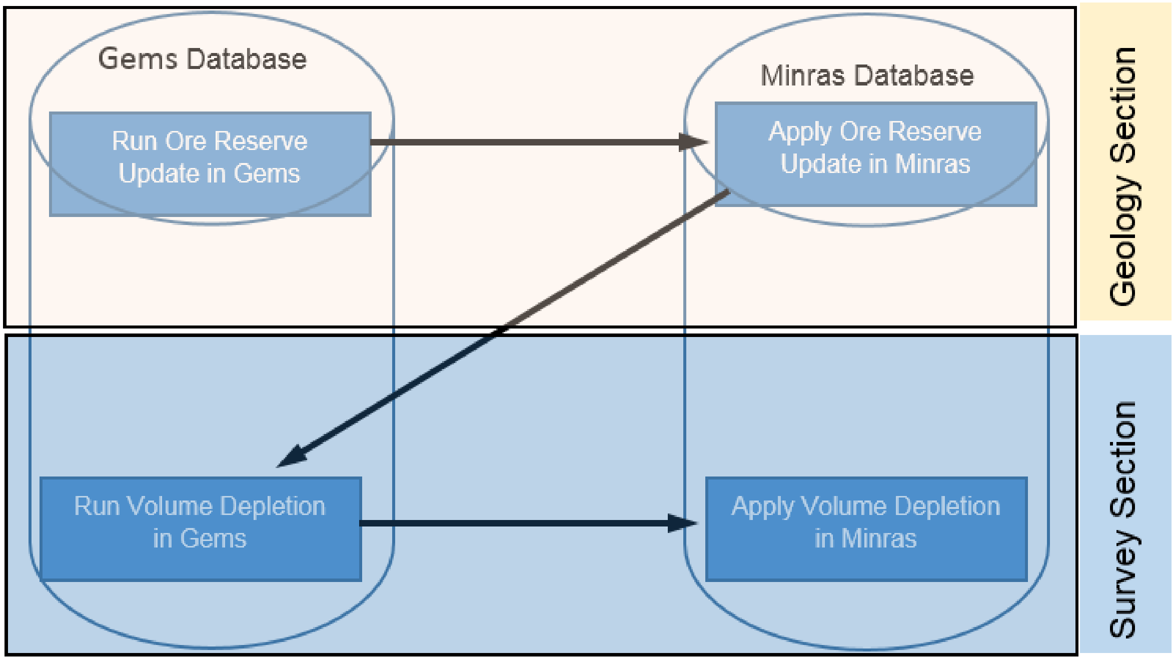


Figure 3-9: Ore Resource Process Flow.

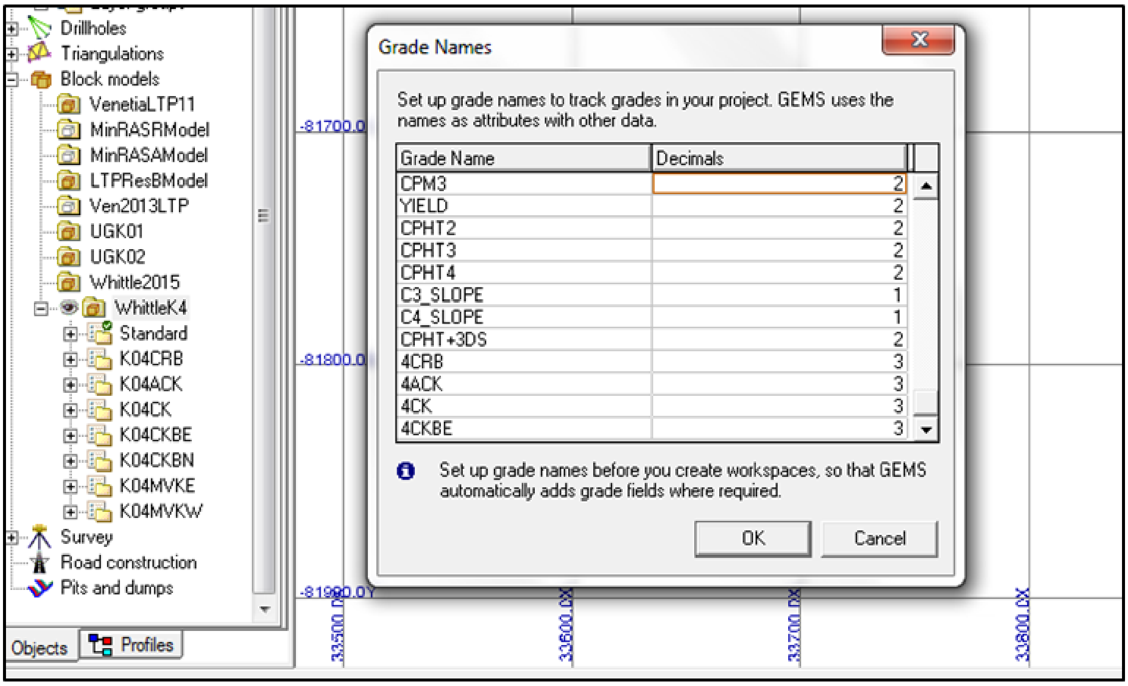


Figure 3-10: Grade Elements in GEMS.

3.11 Model Reconciliation

A reconciliation between the Whittle model and MinRAS model was done to check if the Whittle model was correctly populated when created by Molatana (2015). This was against the backdrop of a similar reconciliation by Molatana (2015) where 0% variances between MinRAS and Whittle models were recorded for volume, tonnage and carats.

Volumetric runs of the MinRASA model and the Whittle model were conducted, constrained between the original topographic surface and the bottom surface. There were no variances in volumes between the two models whilst insignificant variances of 0.34% in tonnage and 0.25% in carats in the K04CRB rock type were recorded.

The variances in tonnage and carats could be attributed to the needling density used, the Molatana (2015) used a 5 × 5 needling density accuracy while the reconciliation done for this project research used a 25 × 25 needling density accuracy. These insignificant variances show that the Whittle model was accurately constructed and populated. Table 3-10 shows the detailed reconciliation.

Table 3-10: Reconciliation of MinRASA Model and Whittle Model.

	MinRasAModel			Whittle Model			Variance		
Pipe	Volume (m^3)	Tonnage	Total Carats	Volume (m^3)	Tonnage	Total Carats	Volume (m^3)	Tonnage	Total Carats
K04	7,775,824	21,036,836	3,433,891	7,775,824	21,023,803	3,432,599	0.00%	0.06%	0.04%

3.12 Whittle Codes

Whittle cannot take the block model rock types with more than four characters. Thus, the block model rock types were manipulated into a Whittle rock type with four characters. Each Whittle grade rock type was manipulated into a Whittle grade element, however during optimum pit selection all Whittle grade elements were assigned a varying revenue factor from 0.1 to 1.8, where the optimal pit was defined by a revenue factor of 1. Table 3-11 summarises the Whittle codes.

Table 3-11: A Summary of Whittle Codes.

VenetiaRes11 Block Model					VenRes114x Block Model			Whittle Block Model Rock Codes		
No. of facies per pipe	Pipe	Block Model Code	Block Model Rock Type	Block Model Grade	Whittle Rock Type	Whittle Element Grade	Whittle Revenue Group	Whittle Rock Type	Whittle Element Grade	Whittle Revenue Group
1	K04	410	K04CRB	CPHT	K4CR	4CRB	4CRB	K4CR	4CRB	4CRB
2	K04	408	K04ACK	CPHT	K4AC	4ACK	4ACK	K4AC	4ACK	4ACK
3	K04	409	K04CK	CPHT	K4CK	4CK	4CK	K4CK	4CK	4CK
4	K04	411	K04CKBE	CPHT	K4BE	4CKBE	4CKBE	K4BE	4CKBE	4CKBE
5	K04	412	K04CKBN	CPHT	K4BN	4CKBN	4CKBN	K4BN	4CKBN	4CKBN
6	K04	414	K04MVKE	CPHT	K4KE	4MVKE	4MVKE	K4KE	4MVKE	4MVKE
7	K04	415	K04MVKW	CPHT	K4KW	4MVKW	4MVKW	K4KW	4MVKW	4MVKW

3.13 Mine Cost Adjustment Factor (MCAF)

A mining cost adjustment factor (MCAF) was applied in Whittle to compensate for differences in elevation and dumping distances during mining. The concept of determining the MCAF is shown in Figure 3-11. The factor is determined by varying the cost per operating hours of a haul truck up and down a specific bench. Thus, the top bench is assigned a zero factor and a 0.56 factor per bench as the mining depth increases as per Venetia (2010).

The steps to update the GEMS block model, include exporting the MCAF into Whittle K04 block model followed by re-coding the Z-Values to benches. Thereafter the populated block model file was imported into the MCAF model standard folder. This was to allow Whittle to take into account the MCAF factor during optimum pit selection.

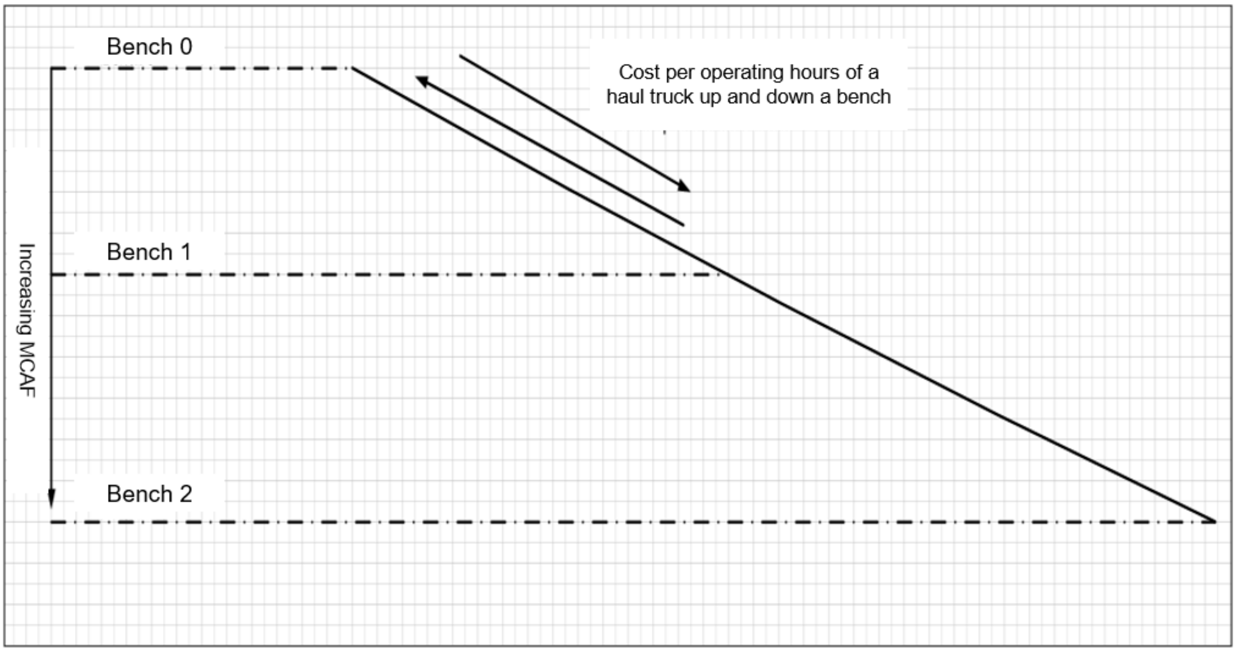


Figure 3-11: MCAF versus Bench Elevation.

3.14 K04 Mineral Statement

The mineral resource statement used for the optimum pit selection is based on MinRAS model, which is informed by the K04 2012 (25m × 25m × 12m) mineral resource estimate. Table 3-12 summarises the volumes, tonnes, carats, density, and grades for the K04 kimberlite while Table 3-13 shows the of the estimated blocks. K04CK has the highest number of estimated blocks with highest grade of 47.5 cpht and is the largest rock type by volume accounting for 74% of the available resource.

Table 3-12: K04 Under Depleted Mineral Resource Statement.

Pipe	Volume (m^3)	Density (t/m^3)	Tonnage	Carats	Grade (cpht)
K04	7,775,823.58	2.70	21,036,835.99	3,433,990.93	16.32

Table 3-13: Classical statistics for K04 estimated grades.

	Grades-cpht									
Column1	*K04ACK*	*K04CK*	*K04CKBE*	*K04CKBN*	*K04CRB1*	*K04CRB2*	*K04CRB3*	*K04CRB4*	*K04MVKE*	*K04MVKW*
Mean Grade	**20.10**	**18.13**	**6.65**	**6.70**	**2.70**	**12.68**	**2.70**	**2.70**	**6.60**	**6.60**
Standard Error	0.00	0.23	0.12	0.00	0.01	0.36	0.00	0.00	0.00	0.00
Median	20.10	17.00	6.60	6.70	2.70	15.80	2.70	2.70	6.60	6.60
Mode	20.10	17.00	6.60	6.70	2.70	15.80	2.70	2.70	6.60	6.60
Standard Deviation	0.00	8.78	1.54	0.00	0.03	6.32	0.01	0.02	0.01	0.03
Sample Variance	0.00	77.16	2.38	0.00	0.00	39.89	0.00	0.00	0.00	0.00
Kurtosis	(2.08)	0.28	0.98	(2.03)	28.79	0.07	90.00	24.34	104.00	15.00
Skewness	(1.03)	0.71	0.33	(1.01)	4.07	(0.63)	(9.49)	(5.04)	10.20	0.00
Range	-	46.70	8.50	-	0.30	29.80	0.10	0.10	0.10	0.20
Minimum Grade	20.10	0.80	2.50	6.70	2.60	0.10	2.60	2.60	6.60	6.50
Maximum Grade	**20.10**	**47.50**	**11.00**	**6.70**	**2.90**	**29.90**	**2.70**	**2.70**	**6.70**	**6.70**
Number of blocks	56.00	1,485.00	171.00	158.00	43.00	307.00	90.00	54.00	104.00	31.00

3.15 Exporting Resource Block Model from GEMS to Whittle

The resource block model in Gemcom was prepared for Whittle. This involved ensuring that the revenue grades per rock code were created as there are different revenue for different facies. In exporting to Whittle, the mapping of all the grades per rock code were done ensuring that in Whittle all of these will be reflected in order to assign the relevant revenues to those grades. The export was confined to September 2017 mining surface.

In Whittle the resource block model was imported using the “Block Model Import Wizard” functionality to start the optimum pit selection process. In Whittle the processing method and units of measurements were defined in the early stages of importing and setting up of the project before running the optimum pit selection process.

3.16 Pit Shell Generation in Whittle

Three scenarios were investigated, these are base case, scenario 1 and scenario 2. Input parameters utilised for the optimum pit selection exercise are detailed in Table 3-14. The base case used the inputs provided by Venetia Mine’s TS department.

For scenario 1, it was posited that K04 alone could not attract extra security or HR costs, which are independent from the rest of Venetia Mine. Thus, DBGS support costs, security and HR costs were removed and the overall processing costs were reduced to R123.30.

Scenario 2 was based on the variation of the initial ZAR and USD exchange rate. The softening of world stock markets coupled with low oil prices in recent months have had an impact on the exchange rate of the Rand to other currencies. The Rand has weakened significantly against the US dollar. However, based on the current performance of the Rand against the US dollar, the reasonable forecast of the exchange rate until the end of 2018 would be 15. Therefore, the exchange rate to be used for scenario 2 is ZAR15.00 to USD1.00 (Investing.com, 2018).

Table 3-14: K04 Optimum Pit Selection Input Parameters.

		Base Case		Scenario 1		Scenario 2	
Parameter	**Description**	**Input**		**Input**		**Input**	
Resource Modifying Parameters	Resource carat ratio	95%		95%		95%	
	External dilution	5%		5%		5%	
	Mining loss	3%		3%		3%	
		Stack Angle	**Bench Height (m)**	**Stack Angle**	**Bench Height (m)**	**Stack Angle**	**Bench Height (m)**
Geotechnical Parameters	Domain N	43º	12	43º	12	43º	12
	Domain SE	37º	12	37º	12	37º	12
	Domain S	36º	12	36º	12	36º	12
	Domain SSW	36º	12	36º	12	36º	12
	Domain WSW	44º	12	44º	12	44º	12
Operating Costs	Mining Costs	**ZAR**		**ZAR**		**ZAR**	
		34.68		34.68		34.68	
	MCAF	0.56		0.56		0.56	
	Overall Processing Costs	240		123.3		123.3	
	Capital Costs	100 000 000		100 000 000		100 000 000	
Exchange Rate (ZAR: USD)		12		12		15	
Selling cost R/Ct		1 047.98		1 047.98		1 309.97	
Discount factor		10%		10%		10%	

3.17 Chapter Summary

This chapter discussed the data collected for the modifying factors used in the K04 optimum pit selection. These include resource modifying factors, geotechnical parameters, operating costs, exchange rates and discount factors.

Table 3-14 shows the summary of these inputs which were used for the base case optimum pit selection scenario. Steps taken in the creation of the Whittle model and the reconciliation conducted between MinRAS model and the Whittle model were also discussed. Insignificant variances between the two models shows that the Whittle model was correctly constructed and populated.

Optimum pit selection was carried out for three scenarios (base, scenario 1 and scenario 2) based on variation of some of the input parameters. Interpretation, discussion of results and sensitivity analysis of the variables will be detailed in the next chapter.

4. RESULTS AND DISCUSSION

4.1 Introduction

This chapter details and discusses the results of the Whittle optimisation of K04. Optimum pit selection outcomes of the three scenarios were analysed in excel to come up with pit by pit graphs. The optimal pit of the project was selected based on the pit by pit graphs, revenue factors, strip ratio but more essentially looking at the pit which mines more ore and returns the highest NPV. A life of pit schedule for the optimal pit was then conducted to get a production profile. Furthermore, the sensitivity analysis showing the impact of change in diamond price, mining costs and processing costs is also discussed in this chapter.

4.2 Whittle Optimum Pit Selection Results

4.2.1 Base case

The detailed optimum pit selection for base case produced four pits with revenue factors (RF) ranging from 1.05 to 1.2. Whittle was unable to produce a pit with a revenue factor of 1 (RF of 1). Table 4-1 details the optimum pit selection summary. The output of the Whittle optimum pit selection is shown in Figure 4-1. The graph in Figure 4-1 shows revenue factors associated with generated pit shells, waste and ore tonnes and the discounted cash flows. All the four pits generated a negative discounted cash flow.

Table 4-1: K04 optimum pit selection Base case pit summary.

Pit Number	Revenue Factor	Total Tonnes	Waste Tonnes	Ore Tonnes	Strip Ratio	Grade-cpht	Recovered Carats	Discounted Pit Value Rand (millions)
1	1.05	198,530	73,036	125,494	0.58	33.08	41,513	(101.05)
2	1.10	743,622	423,188	320,434	1.32	34.74	111,319	(106.41)
3	1.15	986,045	590,918	395,127	1.50	34.95	138,097	(109.62)
4	1.20	1,097,218	668,468	428,750	1.56	34.93	149,762	(111.34)

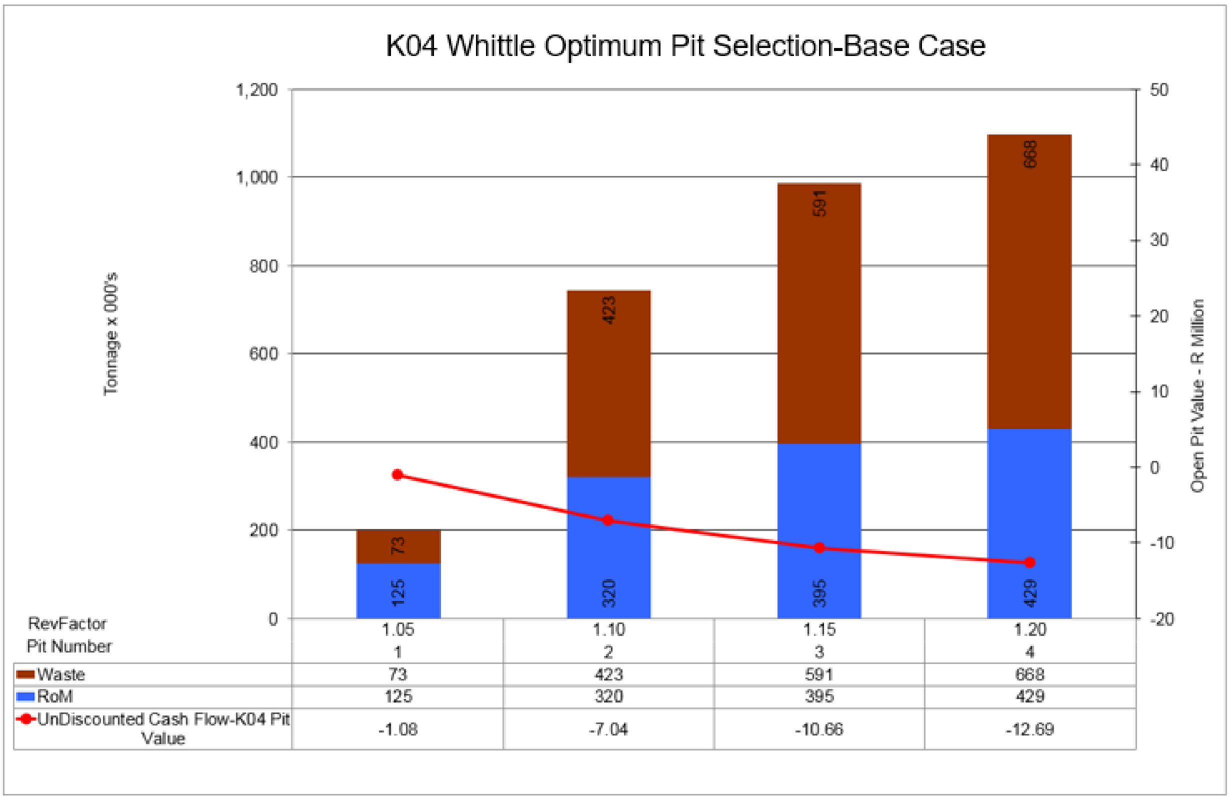


Figure 4-1: Base case pit by pit K04 optimum pit selection and discounted cash flow value.

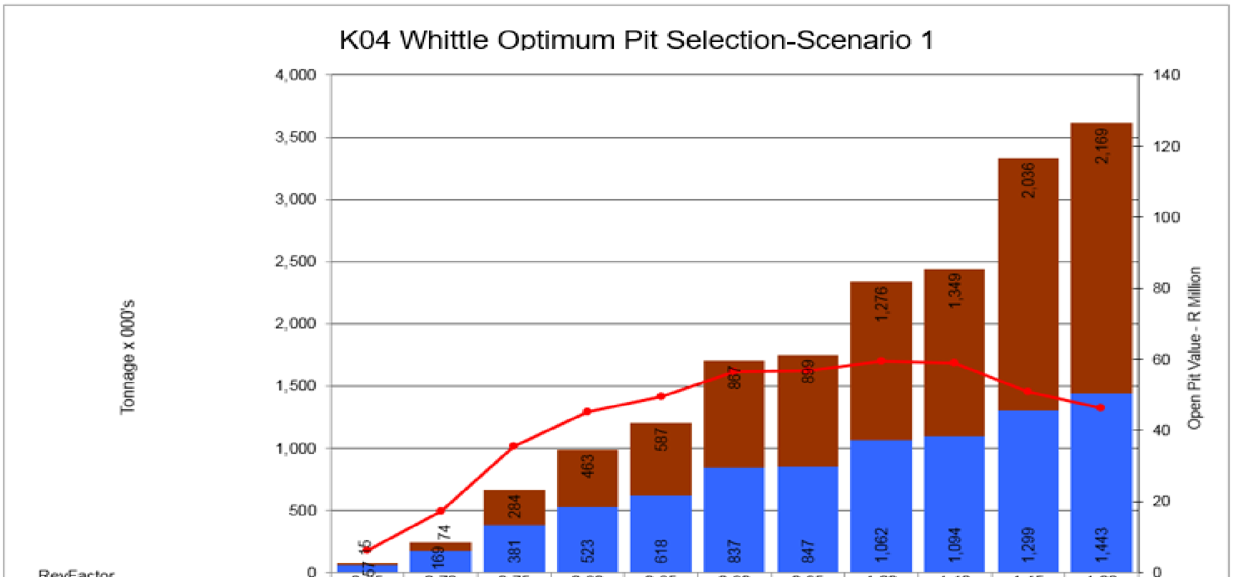
4.2.2 Scenario 1

The detailed optimum pit selection for scenario 1 produced eleven pits with RF ranging from 0.65 to 1.2. Table 4-2 details the optimum pit selection summary. Pit 8 with RF of 1 mines a total of one million ore tonnes, producing about 0.32 million carats and generating a revenue of -ZAR49.74 million. The output of the Whittle optimum pit selection is shown in Figure 4-2. The graph in Figure 4-2 shows revenue factors associated with generated pit shells, waste and ore tonnes, and the discounted cash flows.

There is a steady increase in both waste and ore tonnes mined from pit 1 to pit 11, however the discounted cash flow drops from pit 9 indicating that larger pit push backs results in higher waste stripping in order to produce ore, where pit 11 mines 70% more waste than pit 8.

Table 4-2: K04 optimum pit election Scenario 1 pit Summary.

Pit Number	Revenue Factor	Total Tonnes	Waste Tonnes	Ore Tonnes	Strip Ratio	Grade-cpht	Recovered Carats	Discounted Pit Value Rands (millions)
1	0.65	72,687	15,269	57,418	0.27	31.60	18,144	(93.86)
2	0.70	243,604	74,217	169,387	0.44	31.63	53,577	(83.37)
3	0.75	665,132	284,262	380,870	0.75	32.04	122,031	(67.50)
4	0.80	986,045	463,273	522,772	0.89	31.84	166,451	(59.38)
5	0.85	1,204,606	586,817	617,789	0.95	31.47	194,418	(55.86)
6	0.90	1,704,230	866,981	837,249	1.04	30.40	254,524	(50.97)
7	0.95	1,746,134	898,652	847,482	1.06	30.47	258,228	(50.75)
8	1.00	2,337,985	1,275,641	1,062,344	1.20	29.81	316,685	(49.74)
9	1.10	2,443,119	1,349,424	1,093,695	1.23	29.75	325,374	(50.11)
10	1.15	3,334,843	2,035,891	1,298,952	1.57	29.55	383,840	(56.40)
11	1.20	3,612,623	2,169,180	1,443,443	1.50	28.50	411,381	(59.38)



RevFactor	0.65	0.70	0.75	0.80	0.85	0.90	0.95	1.00	1.10	1.15	1.20
Pit Number	1	2	3	4	5	6	7	8	9	10	11
Waste	15	74	284	463	587	867	899	1,276	1,349	2,036	2,169
RoM	57	169	381	523	618	837	847	1,062	1,094	1,299	1,443
UnDiscounted Cash Flow-K04 Pit Value	6.20	17.19	35.58	45.10	49.57	56.45	56.80	59.31	59.01	50.89	46.21

Figure 4-2: Scenario 1 pit by pit K04 optimum pit selection and discounted cash flow value.

4.2.3 Scenario 2

The detailed optimum pit selection for scenario 2 produced thirteen pits with RF ranging from 0.55 to 1.2. Table 4-3 details the optimum pit selection pit summary. Pit 9 with a RF of 1 mines a total of 1.6 million ore tonnes producing about 0.45 million carats and generating an NPV of ZAR13.2 million. The output of the Whittle optimum pit selection is shown in Figure 4-3. The graph in Figure 4-3 shows revenue factors associated with generated pit shells, waste and ore tonnes and the discounted cash flows. There is a steady increase in both waste and ore tonnes mined from pit 1 to pit 13, however the discounted cash flow becomes flatter and begins to drop from pit 10.

Table 4-3: K04 optimum pit selection Scenario 2 pit summary.

Pit Number	Revenue Factor	Total Tonnes	Waste Tonnes	Ore Tonnes	Strip Ratio	Grade-cpht	Recovered Carats	Discounted Pit Value Rands (millions)
1	0.55	243,604	74,217	169,387	0.44	31.63	53,577	(71.9)
2	0.60	665,132	284,262	380,870	0.75	32.04	122,031	(42.8)
3	0.65	986,045	428,757	557,288	0.77	30.68	170,976	(25.5)
4	0.70	1,204,606	552,302	652,304	0.85	30.50	198,953	(17.0)
5	0.75	1,746,134	864,137	881,997	0.98	29.79	262,747	(1.1)
6	0.80	2,337,985	1,241,126	1,096,859	1.13	29.28	321,160	8.9
7	0.90	2,443,119	1,314,909	1,128,210	1.17	29.24	329,889	9.8
8	0.95	3,418,678	1,959,728	1,458,950	1.34	27.92	407,339	12.7
9	1.00	3,823,697	2,155,206	1,668,491	1.29	26.77	446,655	13.2
10	1.05	3,852,268	2,165,065	1,687,203	1.28	26.65	449,640	13.1
11	1.10	4,795,097	2,849,548	1,945,549	1.46	26.07	507,205	9.7
12	1.15	5,230,647	3,168,131	2,062,516	1.54	25.84	532,954	7.8
13	1.20	5,579,214	3,386,117	2,193,097	1.54	25.35	555,950	5.5

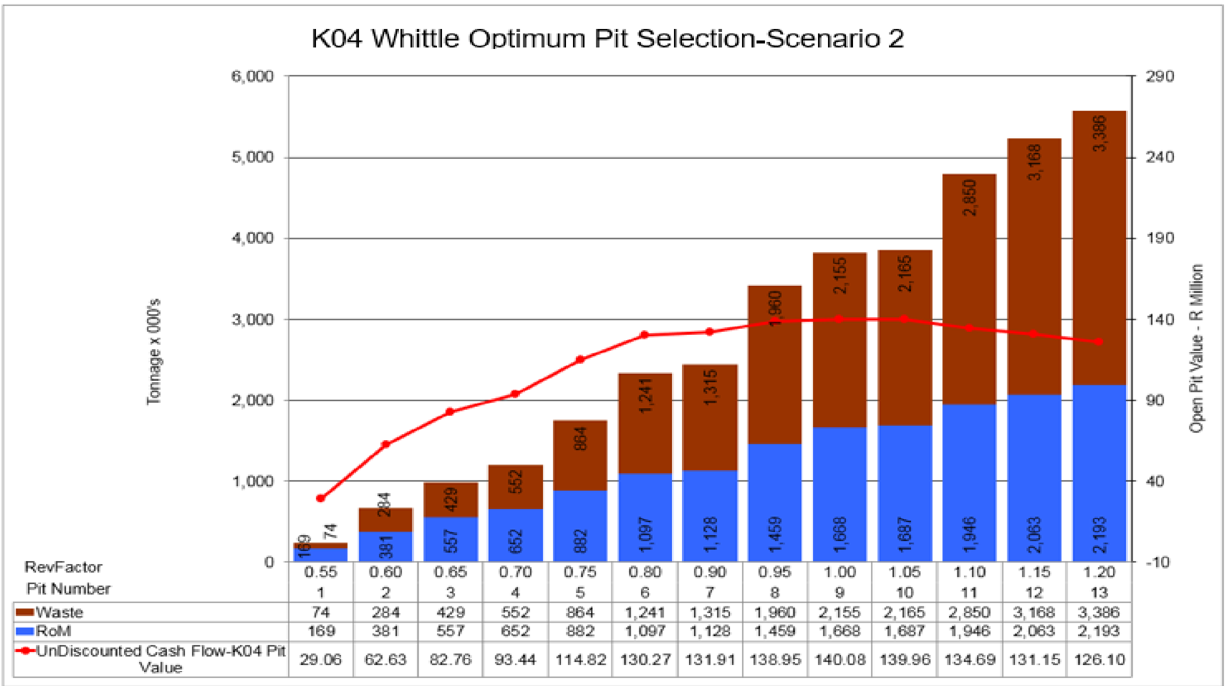


Figure 4-3: Scenario 2 pit by pit K04 optimum pit selection and discounted sash flow value.

4.3. Selection of Overall Optimal Pit

Whittle optimisation process produces a series of economic pit shells. These pit shells define what can be mined from an ore body. Once these shells are produced, the next step is to select an optimal pit (Breed and van Heerden, 2016).

Breed and van Heerden (2016) stated that the selection of the final pit is based on many measures and these varies depending on the company's strategy. Typically, selection is done by maximising key performance indicators which are;

- Annual cash flow;
- Life of mine;
- Operating risk;
- Technical, economic and political risks; and
- Product requirements (blending strategies).

4.3.1 K04 Optimum Pit Selection Strategy

Breed and van Heerden (2016) stated that the most important strategy of a life of mine is to maximise value. Thus, besides maximising value, the strategy for K04 optimum pit selection is based on NPV, revenue factor, stripping ratio, and life of mine.

4.3.2. K04 Optimum Pit Selection

After generating pit shells for the base case, scenario 1 and scenario 2, the results for the base case and scenario 1 showed negative pit values whilst some pits for scenario 2 had positive pit values. Whittle generates three types of schedules namely; best, specified and worst case showing the discounted pit value for each pit. Figure 4-4 shows the Whittle output pit by pit graph for scenario 2.

Based on the strategy of maximising value/NPV, pit 9 was selected as an optimum pit. The pit has the discounted pit value of ZAR13.2 million for the best case. This value is slightly better than pit 10, even though pit 10 mines slightly more ore and has a better carat recovery. The driving factor for the differences in value is that pit 10 mines more waste than pit 9.

Revenue factor is defined as the ratio of incremental cost of various pits to incremental revenue. Whittle (2009) stated that depending on the level of confidence in the resource model and the various estimates, the contents of the resulting revenue factor of 1 pit shell using the marginal cut-off grade, may define how much of the deposit is economically sound.

He further discussed that when the time value of money is considered, the outer shells of the revenue factor of 1 pit can display a reduced value because the cost of waste stripping precedes the profits derived from ore mined. Thus, pit 9 with a revenue factor of 1 was selected as an optimal pit.

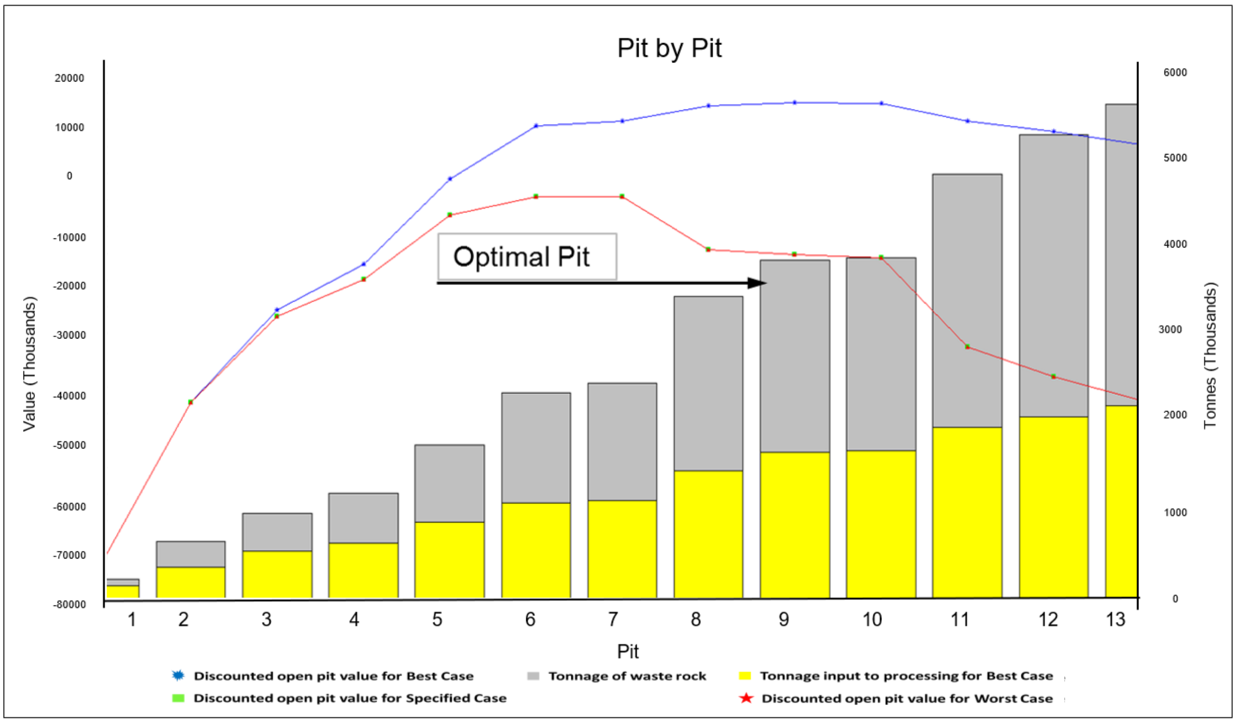


Figure 4-4: Pit-by-Pit Graph for Scenario 2.

The overall strip ratio was also used as a qualifying measure for selecting the optimum pit. Pit 9 has a favourable overall stripping ratio of 1.29 compared to pit 8, 11, 12 and 13. Pit 10 has a slightly better strip ratio; however, it has a slightly lower NPV compared to pit 9. Thus, based on strip ratio, pit 9 was selected as an optimum pit.

The life of mine also plays a role in selecting the optimal pit. The proposed project's objective is to address carats shortfall in a specific period, thus in this scenario the period of concern spans at least four years. The schedule for different pits resulted in pit 9 having a LoM of at least 5 years which can be implemented to address the carat shortfall.

4.4. Sensitivity Analysis of Economic Parameters

Changes in economic parameters play a crucial role in the overall profitability of mining projects. Positive or negative changes in these parameters will result in loss or gain of revenues and thereby determine whether a project is viable or not. These parameters include operating costs, product prices and exchange rates. Table 4-4 shows the overall picture where the optimum pit was optimised by varying prices, exchange rates, mining and processing costs using one input per optimisation whilst the other inputs were kept constant.

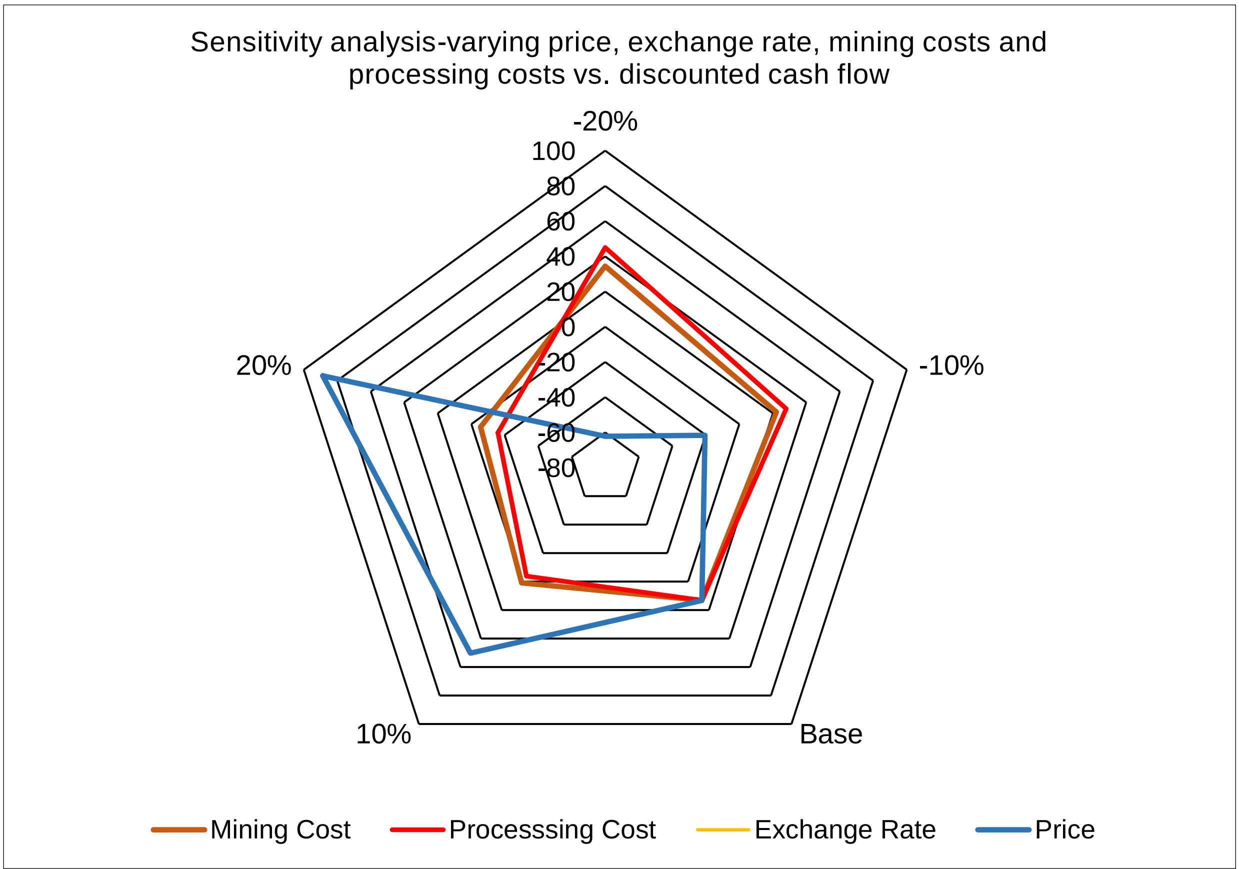


Figure 4-5: Effect of change in parameter on cash flow and ore mined.

Figure 4-5 above shows the effect of changing one input parameter whilst others are kept constant on cash flow. Increases in diamond price will have a positive impact on the cashflow, whilst increases in mining and processing costs negatively affects the cashflow. Positive or negative price changes have the largest impact on the overall cashflow.

On the other hand, reducing mining and processing costs creates opportunities for favourable cashflow and any increase in either of these inputs results in reduced cashflows. Changes in mining costs have the least effect on cashflow as shown by the shallow slope.

Table 4-4: Sensitivity Results.

Variable	Change	Pit No.	Mining Cost R/ton	Price (ZAR)	Total Proc. Cost R/ton	Royalt 10%	Mining Recovery (%)	Mining Dilution (%)	Exchange Rate (ZAR:US$)	Optimisation Results				
										Ore (Mt)	Grade (cpht)	Waste (Mt)	Strip Ratio	Disc Pit Value Rand (millions)
	-20%	9	34.68	1047.98	123.23	62.88	95	1.05	15	1.55	27.81	2.27	1.46	-62.14
	-10%	9	34.68	1178.98	123.23	70.74	95	1.05	15	1.15	29.06	1.34	1.16	-20.47
Price	Base	9	34.68	1309.97	123.23	78.6	95	1.05	15	1.67	26.77	2.16	1.29	13.20
	10%	9	34.68	1440.97	123.23	86.46	95	1.05	15	1.60	27.00	2.04	1.28	50.25
	20%	9	34.68	1571.96	123.23	94.32	95	1.05	15	1.75	26.03	2.08	1.19	88.66
	-20%	9	27.74	1309.97	123.23	78.60	95	1.05	15	1.55	27.5	2.09	1.35	34.41
	-10%	9	31.21	1309.97	123.23	78.60	95	1.05	15	1.59	27.28	2.13	1.34	22.16
Mining Cost	Base	9	34.68	1309.97	123.23	78.60	95	1.05	15	1.67	26.77	2.16	1.29	13.20
	10%	9	38.15	1309.97	123.23	78.60	95	1.05	15	1.15	29.06	1.34	1.17	0.92
	20%	9	41.62	1309.97	123.23	78.60	95	1.05	15	1.21	28.47	1.34	1.11	-5.56
	-20%	9	34.68	1309.97	98.58	78.6	95	1.05	15	1.86	25.40	2.14	1.15	44.98
	-10%	9	34.68	1309.97	110.91	78.6	95	1.05	15	1.71	26.42	2.12	1.24	28.01
Processsing Cost	Base	9	34.68	1309.97	123.23	78.6	95	1.05	15	1.67	26.77	2.16	1.29	13.20
	10%	9	34.68	1309.97	135.55	78.6	95	1.05	15	1.58	27.35	2.13	1.35	-3.83
	20%	9	34.68	1309.97	147.88	78.6	95	1.05	15	1.34	29.13	2.03	1.51	-15.89
	-20%	9	34.68	1309.97	123.23	62.88	95	1.05	12	1.55	27.81	2.27	1.46	-62.14
	-10%	9	34.68	1309.97	123.23	70.74	95	1.05	13.5	1.15	29.06	1.34	1.16	-20.47
Exchange Rate	Base	9	34.68	1309.97	123.23	78.6	95	1.05	15	1.67	26.77	2.16	1.29	13.20
	10%	9	34.68	1309.97	123.23	86.46	95	1.05	16.5	1.60	27.00	2.04	1.28	50.25
	20%	9	34.68	1309.97	123.23	94.32	95	1.05	18	1.75	26.03	2.08	1.19	88.66

4.4.1 Sensitivity analysis of cash flow to changes of diamond price and exchange rate based on the optimal pit

Product price is a very critical economic parameter with regard to generation of revenues. Based on the optimal pit the diamond price and exchange rate were varied between -20% and 20% at 10% intervals whilst other parameters were kept constant and the resultant cash flows were noted. A 20% increase in price or exchange rate will result in 572% increase in discounted cash flow whilst a 20% decrease in price will result in 571% decrease in cashflow.

This shows that any fall in diamond price or exchange rate will a have a huge impact on the K04 project. Thus, the overall project is extremely sensitive to price change as shown by a steep slope of the graph in Figure 4-6.

4.4.2 Sensitivity analysis of cash flow to changes of mining cost based on the optimal pit

The optimal pit was tested to indicate changes in mining costs if the price, exchange rate and processing costs were kept constant. Mining costs were varied between -20% and 20% at intervals of 10% and resultant cash flows were recorded. A 20% increase in mining cost will result in 142% decrease in discounted cash flow whilst a 20% decrease in mining cost will result in 160% increase in cash flow.

This shows that the project is less sensitive to a decrease in mining costs as shown by a flatter slope and more sensitive to increases in mining costs as shown by a fairly steep slope, thus any increase to mining costs will have an impact on the project (see Figure 4-6).

4.4.3 Sensitivity analysis of cash flow to changes of overall processing costs based on the optimal pit

Processing costs were varied between -20% and 20% at intervals of 10% and resultant NPVs were recorded. A 20% increase in processing cost will result in 220% decrease in undiscounted cash flow whilst a 20% decrease in processing costs will result in 240% increase in cashflow. This shows that the project is reasonably sensitive to changes in processing costs, thus any changes to processing costs will have an impact on the project (see Figure 4-6).

Overall, the profitability of the K04 project is dependent on diamond price. Thus, the project is highly sensitive to changes in diamond price. Figure 4-6 shows the sensitivity of the four parameters, where the diamond price shows a steeper slope compared to changes in mining or processing costs.

Thus, if the price of the diamonds decreases, then the business needs to reduce production to cushion the impact caused by the drop in price. However, reducing production introduces a risk that K04 will not be mined as the organisation will target high grade resource. The other factor that will affect the project is the exchange rate where the effect is the same as the changes in price and the organisation has no influence over exchange rate.

Quality of diamonds determines the price the diamonds will be sold for and this is inherent in the ore body. The organisation has no control over the quality of diamonds recovered.

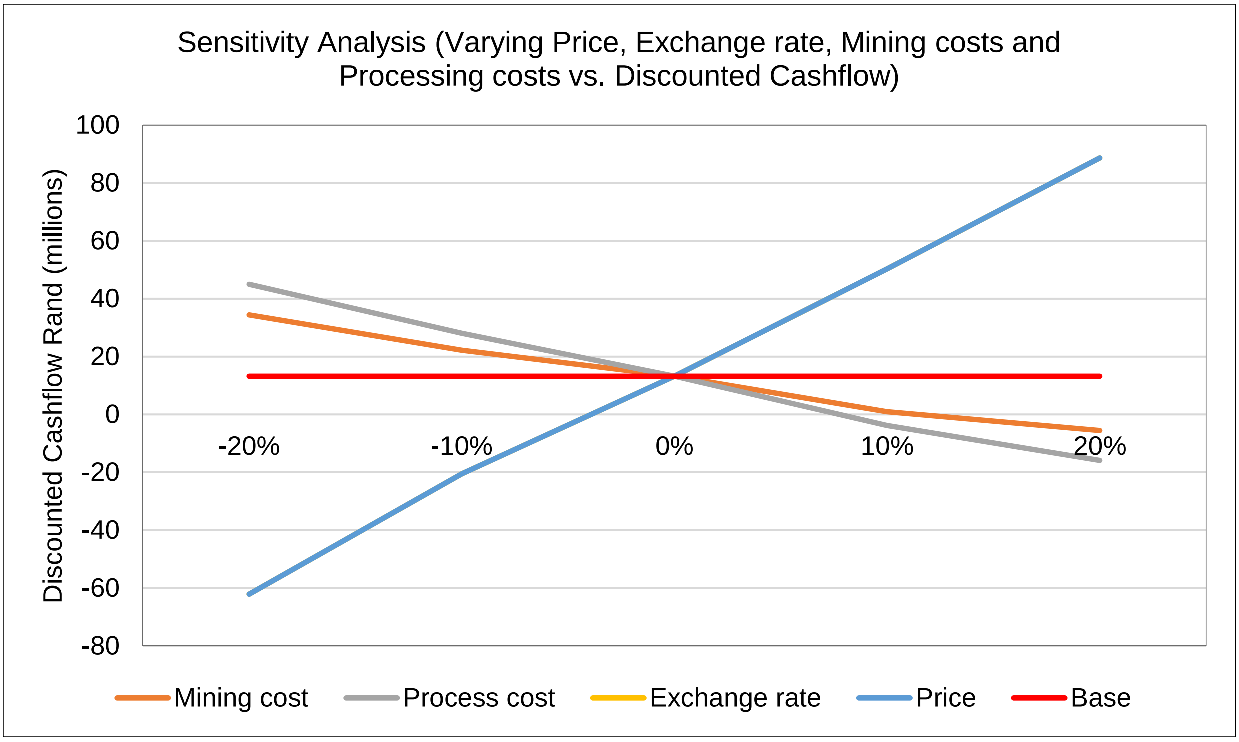


Figure 4-6: Summary of Sensitivity Analysis.

4.5. Mineable Inventory

Table 4-5 summarizes the minable inventory for K04's pit number 9. This inventory was realized after applying modifying factors which include dilution, mining losses and mining recovery. The K04CK constitute 74% both by volume and tonnes of the overall K04 mineral resource and occupies the central position of the geological model. The other facies (K04ACK, K04CKBE, K04CKBN, K04CRB, K04MVKE and K04MVKW) occurs in small quantities on the margins of the model and are associated with low grades.

Thus, during the Whittle optimization, these minor facies are stripped as part of the waste, hence all the ore is coming from a single facie of K04CK and this mineral resource is classified in the inferred category.

Table 4-5: K04 Minable Inventory.

Lithofacie	Reserve Classification	Mineral Resource Classification	Waste Tonnes	Ore Tonnes	Grade Input-cpht
K04CK			2,155,206		
	Proven	Measured		-	-
	Probable	Indicated		-	-
		Inferred		1,668,491	26.77
	Minable Inventory			1,668,491	26.77
Total			**2,155,206**	**1,668,491**	**26.77**

4.6. K04 Pit Life Schedule Based on Scenario 2 of Pit 9

NPV schedule based on Whittle's Milawa algorithm for the optimal pit 9 was conducted with one push back, based on Whittle ore output. The exercise to conduct a second pushback created higher amount of waste that could not covered by corresponding exposed ore and therefore the idea of more than one pushback was dismissed.

The stripping ratio for pit 9 with one pushback was at its peak during the first year due to substantial high waste stripping to expose ore. The pit life is 5 years 5 months with a highest yearly carats' recovery of 149, 607 contained in 477 520 tonnes of ore. Figure 4-7shows the pit life schedule for K04 and detailed production figures are shown in Table 4-6**.**

The substantial amount of waste stripping to expose ore in the first year resulted in a negative cash flow, however cash flows improved in the coming years with an NPV of ZAR13.2 million generated in the pit life.

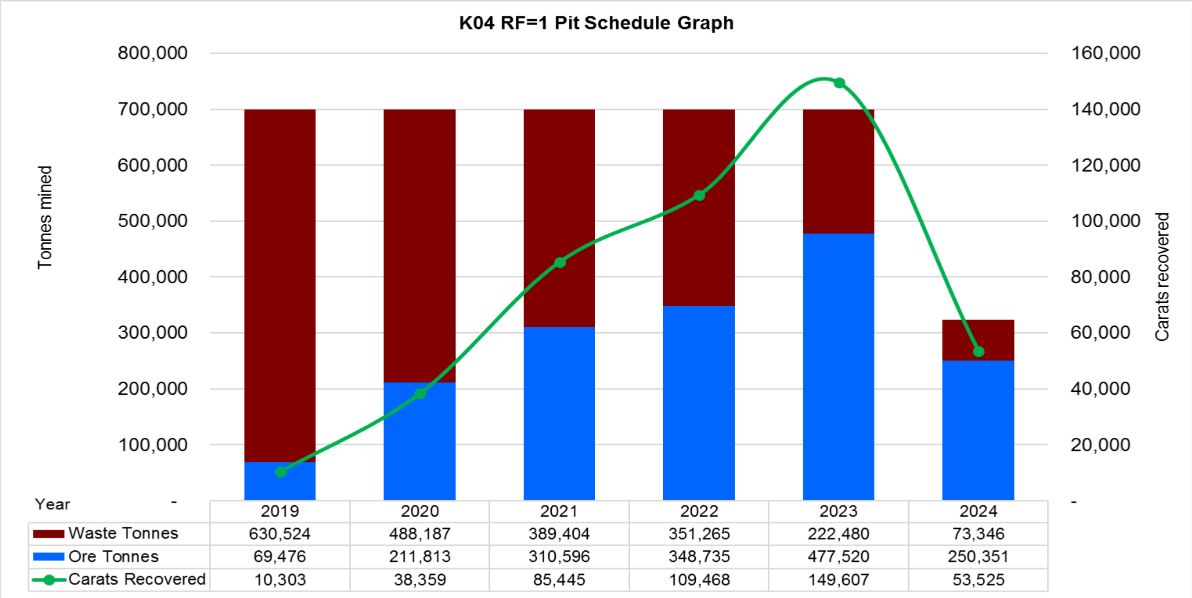


Figure 4-7: K04 Pit Life Production Schedule.

Table 4-6: K04 pit life production profile.

Year	Ore Tonnes	Waste Tonnes	Strip ratio	Grade input-cpht	Undiscounted cashflow	Discounted cashflow	Carats recovered
1	69,476	630,524	9.08	14.83	(23,006,851)	(20,915,319)	10,303
2	211,813	488,187	2.30	18.11	(10,136,583)	(8,377,341)	38,359
3	310,596	389,404	1.25	27.51	29,248,203	21,974,608	85,445
4	348,735	351,265	1.01	31.39	50,687,025	34,619,920	109,468
5	477,520	222,480	0.47	31.33	78,542,741	48,768,863	149,607
6	250,351	73,346	0.29	21.38	14,744,986	8,728,820	53,525

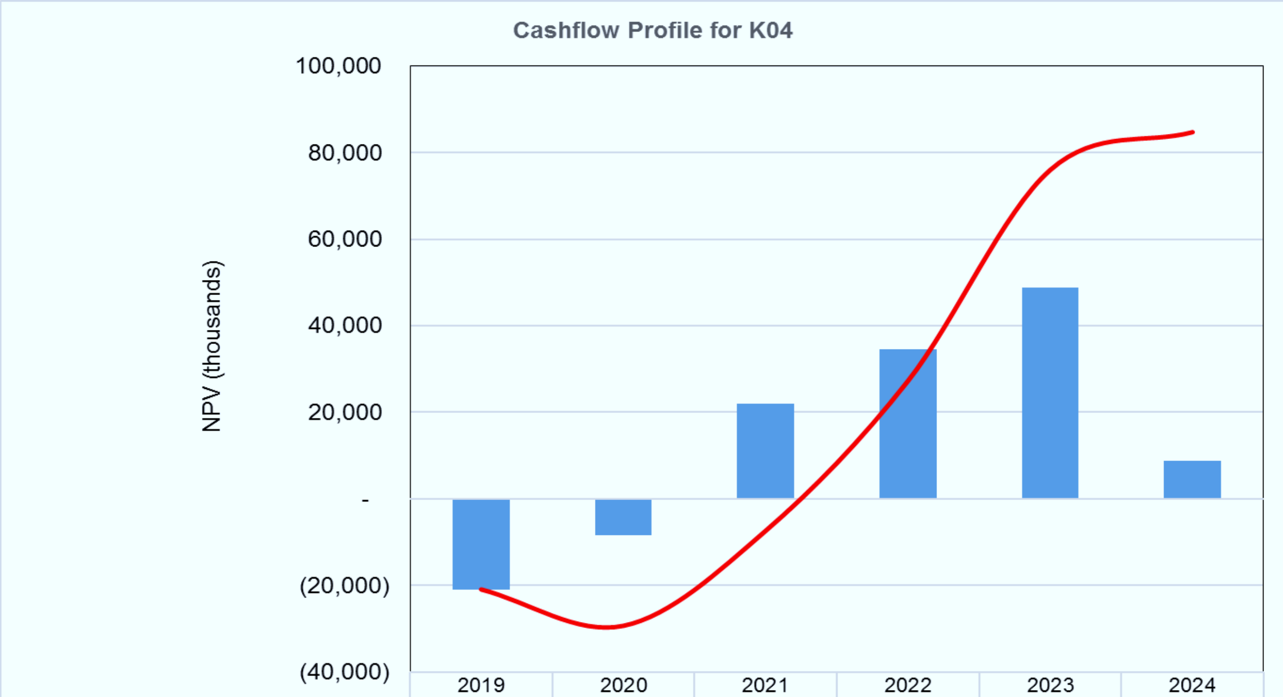
Table 4-7 shows the schedule summary for the optimal pit. An NPV of ZAR13.2 million will be generated from K04 at the end of pit life. The payback period is 2.7 years and 16.67% IRR.

Table 4-7: K04 Schedule Pit Summary.

K04 RF=1 Pit Summary				
Movement	Tonnes			
Ore	1,668,491			
Waste (reject)	698,881			
Waste (other)	1,456,325			
Total	3,823,697			
Strip Ratio	1.29			
Product	Input	Recovered	Grade Input	Pt Util. %
4CK (carat)	446,692	402,023	26.8	80.7
4CR (carat)	0	0	0	0
4AC (carat)	0	0	0	0
4BE (carat)	0	0	0	0
4BN (carat)	0	0	0	0
4KE (carat)	0	0	0	0
4KW (carat)	0	0	0	0
Measures				
NPV (Rand)	13,164,395			
Life (year)	5.46			
Payback (year)	2.73			
Payback ratio	0.5			
IRR%	16.67			

4.7. Optimal pit Cash Flow Analysis

The optimal pit cash flow profile is shown in Figure 4-8 in discounted cash flow as well as in cumulative cash flow terms. In the first year of operation, the project will experience a negative cash flow due to high amounts of waste stripping requirements to expose ore. Once the ore is exposed, with a favourable stripping ratio of less than 1.5, the project will transition to a positive cash flow in year three.



	2019	2020	2021	2022	2023	2024
Discounted cashflow	(20,915)	(8,377)	21,975	34,620	48,769	8,729
Cummulative discounted cashflow	(20,915)	(29,293)	(7,318)	27,302	76,071	84,800

Figure 4-8: Discounted Cash Flow Profile.

4.8. Contribution of K04 Production to the Hiatus

The production of K04 (see Figure 4-7) shows a total of 1.6 million tonnes of ore to be mined over the period of six years producing 402 thousand carats. Ore mined peaks in the fifth year with carats production due to higher grade mined and volume of ore mined. For K04 to have a meaningful and viable contribution to solving the problem stated in the problem statement it should be able to produce at least 0.2 million carats in 2019 and 1.3 million carats in 2022. However, pit 9's individual years' and combined LoM carat production falls short of the 1.5 million carats required to lift the valley of despair where the highest yearly carats production of 149 thousand is recovered is in the fifth year. This shows that the project will not have impact in lifting carat production out of the valley of despair (see Figure 4-9).

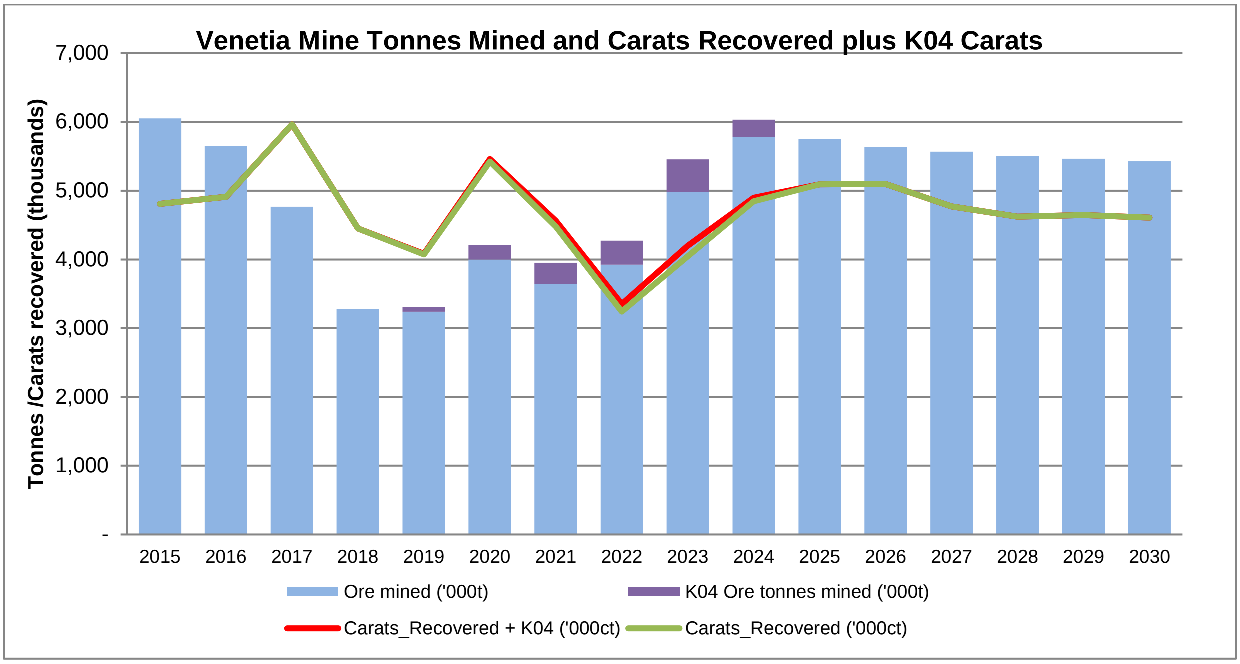


Figure 4-9: Comparison of Venetia Mine's carat profile with K04 carat profile.

4.9. Additional factors to consider

Two treatment options were considered, the first; treating K04 ore through the existing treatment plant facilities and the second, treating K04 through a dedicated plant. These options are discussed in detail below.

4.9.1 Treatment through the existing plant

With this option, the K04 ore is treated through the existing plant. The K04CK material is harder than most of the kimberlite rock types mined at Venetia Mine. Table 4-8 shows average uniaxial compressive strength (UCS) for some of the ore mined at Venetia. The K04CK has an average UCS of 127.54 which is the third hardest ore at Venetia. Thus, treating such material will pose challenges in crushing and might cause delays due to tripping of crushers. Blending K04 ore with other softer ores will improve crushability which will result in improved overall throughput rates.

Table 4-8: Average UCS Values for Venetia Ore.

Rock Type	Average of Uniaxial Compressive Strength (UCS)
CK	127.54
DVK	140.59
FRAGMENTAL	67.21
MK	128.69
MVK	66.84
MVKBR	81.38
RVK	32.53
TKB	75.20
VK	61.49
VKBR	24.70

4.9.2 Treatment through a dedicated plant

A Dense Media Separation (DMS) standalone modular plant can be used to process the K04 ore. Considering 365 days per year and assuming that there are 12 holidays per year, 10 days for maintenance shutdown, which leaves 343 days available for operation. If the plant operates 24 hours at 90% plant availability, a 70 t/h plant would treat at most 514,000 tonnes per year. This will be an ideal plant since the maximum ore tonnes from K04 is 477,520 tonnes per year which is within the yearly capacity of the proposed plant.

The Modular plant setup is estimated to be require ZAR20 million a month prior to processing in order to set up such plant. The plant will have three major sections which include; feed preparation, a dense media separation section (DMS) and a recovery section. In the feed preparation section, the ore will be crushed to under 150 mm. The crushed ore will be fed through a sizing screen to remove fines and over size particles which will be re-cycled through secondary crushers. The correct size particles will then move to the scrubbing section for washing.

Once washed the ore moves to the DMS section for concentration. The tailings report to the tailings dump whilst the concentrate goes to the recovery section. In this section the concentrate is dried before it is taken through X-ray machines for diamond recovery.

Besides capital costs of acquiring the plant, other costs that will be accrued is administration and human resources estimated at ZAR19.4 million per year. These costs will put pressure on the overall NPV. Table 4-9 shows the treatment summary for K04 and Table 4-10 shows estimated yearly human resources and administration costs for running the modular plant.

Table 4-9: K04 Treatment Summary.

Description	
Working days per year	343
Available working hours	8,232
90% Plant availability (hours)	7,409
Tonnes treated per yaer (70 t/hr)	518,616
Carats recovered at an average grade of 26.8 cpht	138,989

Table 4-10: Human Resources and Administration Costs for Modular Plant.

Description	Monthly Salary (ZAR)	Number	Yearly Salary (ZAR)
Assitant	25,000.00	16	4,800,000.00
Operator	28,500.00	16	5,472,000.00
Supervisor	36,800.00	8	3,532,800.00
Foreman	52,000.00	4	2,496,000.00
Plant Metallurgist	66,700.00	2	1,600,800.00
Admin	25,000.00	2	600,000.00
Plant Manager	78,600.00	1	943,200.00
Total			19,444,800.00

In summary, treating the K04 ore through the existing plant is preferred over a dedicated plant as it will not attract extra costs in terms of acquisition of the plant, administration and human resources requirements.

4.10. Chapter Summary

In this chapter, the optimal pit selection results of the three scenarios were discussed. An optimal pit with a revenue factor of 1 was selected from scenario 2. Sensitivity analysis carried out on the optimal pit by varying the following variables; selling cost, exchange rate, mining costs and processing cost shows that the project is extremely sensitive to change in diamond price and exchange rate and reasonably sensitive to processing cost and mining cost change.

The subsequent pit life schedule resulted in 6 years of production, generating an NPV of ZAR13.2 million. A cash flow analysis shows a negative cash flow in the first year due to high waste stripping requirements with positive cash flows thereafter. The best treatment of the ore is through the existing plant which does not require capital expenditure. Even though the project is viable, it’s carat production profile will not have a huge impact to lift the overall carat production profile during the valley of despair period.

5. CONCLUSIONS AND RECOMMENDATIONS

5.1. Overview

Venetia Mine is currently developing an underground mine, to extend the current open pit LoM which is expected to end in 2022, with an estimated lifespan of 20 years to 2042. The open pit-underground transition is planned in 2022, however the undeground project has been facing developmental challenges and indications are that the transition will run late. This will result in a period of carat deficit. On the other hand, waste stripping in Cut4 south is behind schedule and it will be difficult to expose ore from the south before 2019. These two conditions will result in period of low carat production.

Other sources of ore must be investigated in order to find viable ore to cushion the carat deficit anticipated in 2019 and 2022. This has necessitated a decision to evaluate satellite pipes and in particular K04, with the aim of finding alternative viable ore. This chapter summarizes the research findings and ends with conclusions and recommendations.

5.2. Conclusion

Out of the three scenarios investigated, the optimal pit was realised from scenario 2, where pit 9 was selected. This was on the backdrop of having removed human resources, security, overhead costs and varied the ZAR: USD exchange rate from 12 to 15 compared to the base case.

The strategy for K04 optimum pit selection was based on NPV, revenue factor, stripping ratio, and life of mine. After generating thirteen pits for scenario 2 a comparison of each pit's discounted value was conducted and pit 9 was selected as the optimum since it displayed the highest NPV of ZAR13.2 million. At cut-off grades, the revenue factor of 1 defines how much of the deposit is economic. Out of the thirteen pits, pit 9 has a revenue factor of 1 and a corresponding high pit value. On this basis pit 9 was selected as an optimum pit.

The overall strip ratio in combination with the pit value was also considered in selection of the optimum pit. The selected optimum pit (pit 9) has a slightly higher strip ratio compared to pit 10, however pit 10 was not selected based on its slightly lower NPV. Each pit's life of mine was assessed against the project's objective of addressing carats shortfall in a specific period. The results showed that pit 9's 6-year life of mine is ideal because it is adequate to cover the period where there is a carat shortfall.

The minable inventory of the final optimal pit generated contained 1,668,491 tonnes of ore and 402,023 carats at an average grade of 26.8 cpht. It is worth noting that only K04CK will be mined. This is because K04CK is the most abundant rock type compared to the other facies and it is centrally located.

The highest yearly carats production of 149 thousand recovered in the fifth year, is significantly lower than what is needed to lift the valley of despair in 2019 and 2022 where a shortfall of 0.2 million carats and 1.3 million carats is expected respectively. The K04 carat contribution per year from 2019 to 2024 if production from K04 commences in 2019 has an insignificant effect on the carat profile where the valley of despair still remains.

The sensitivity conducted on the optimal pit 9 shows that the K04 project is extremely sensitive to change in diamond price, processing costs and relatively sensitive to mining costs. Therefore, any change to the diamond price has a huge impact on the viability of the project. Negative change will make the project unviable whilst a positive change will increase the profitability of the project due to mining of lager pits and producing more carats. However, this input cannot be managed or influenced therefore there is no opportunity to maximise cash flows through managing the price.

The treatment of K04 through the main plant is considered the best option against the dedicated plant. The reason is to cut back on capital expenditure associated with new plants and maximising NPV by utilising the already operating plant.

Overall the K04 project is a viable with a net value of ZAR13.2million, thus its carat production would steadily contribute to the Venetia Mine's carat profile without necessarily offsetting the carat deficit in 2019 and 2022. Below are some recommendations that would immensely benefit DBCM to reduce the impact of production hiatus in 2019 and 2022.

5.3. Recommendations

The process followed in selecting the optimum pit is robust and it identified pit 9 as an optimum pit. However, the financials and the carat production do not support a business case on which DBCM can rely on. Therefore, it is recommended that DBCM find other ways of reducing the impact of the carat shortfall in 2019 and 2022.

Some of the ways recommended include reducing mining activities in the north of the pit and putting more focus in accelerating waste stripping in the south in order to expose ore sooner. Employing cost saving initiatives by reducing operating costs and postponing some capital projects that are deemed not urgent will go a long way in reducing the financial impact in 2019 and 2022.

While the optimal pit was selected based on the current input parameters and the results thereof, my recommendation would be not to consider implementing the K04 project. This is because the contribution of K04 falls far short on filling the valleys of despair in 2019 and 2022. Enhancements to the existing K01 pit cut4 south design could add some carats that might alleviate the carats shortfall.

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