

SELECTING AN UNDERGROUND MINING METHOD IN A KIMBERLITE DIAMOND DEPOSIT (CASE STUDY: VENETIA DIAMOND MINE)

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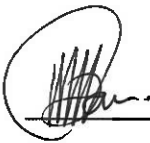
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A research report submitted to the faculty of Engineering and the Built Environment, University of the Witwatersrand in partial fulfilment of the requirements for the degree of Masters in Engineering

DECLARATION

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Takalani Mudau

This 21 day of July 2021

ABSTRACT

Due to the technological advancements, there have been a tremendous development in the mining sector in the cavings and mine design models that allow mining method applications to reach greater depths. The global mining challenges have been ushered into an era where many choices are available to exploit minerals in open pit or underground systems. The most favourable mining method amongst the available options is the one that is safe and has highest economic returns.

The decision of using open pit, underground mining or both is determined by the geometric characteristics of the minerals being mined. Open pit is often considered easier to exploit than underground mining in terms of safety, flexibility, recovery rate, working environment and economics. However, as the open pit continue to deepen, a transition decision will prevail either to mine push back or sink a shaft for going underground.

Venetia mine is a diamond producing operation, currently mining a kimberlite orebody and in the process of transition from Open Pit (OP) to Underground (UG). Open pit activities are planned to cease in 2021. This study aims to investigate the mining method selection for Kimberlite Orebody 01 (K01). Previous studies conducted for the underground mining method selection proposed the block caving mining method to exploit K01. The rigid assumption on the mining method presents a risk to the project. The challenge with block caving in K01 is the narrow orebody width with highly discontinuous nature of contacts. The narrow width limits the caving radius to maximum of 30m with the narrowest area in the eastern part to be 20m. However, this put Venetia underground project in the transitional zone which it is not stable but not caving.

Data analysis were done using Rock Mass Rating (RMR) and Modified Rock Mass Rating (MRMR) classification systems. The results of the geotechnical analysis were compiled into seven kimberlite facies. The calculated MRMR for Dark Volcaniclastic Kimberlite (DVK) facie is 45 and hydraulic radius is 35m. While the MRMR for Country Rock Brescia (CR) is 44 with hydraulic radius of 29m. The DVK facie constitute 92% of the kimberlite facies in K01 deposit which classified as strong and stable for caving. The lesser material from Coherent Kimberlite (CK) and CR are weak and constitute 8% of kimberlite facie in K01.

Expected geotechnical conditions for DVK_HARD Intact Rock Strength (IRS) which is over 100MPa and DVK_MID IRS which is between 50MPa and 100MPa have significant impact on the mining method selection. Results of the analysis performed in the study indicate a competent ore body which require pre-drilling for caving to occur. Stress induced scaling and weathering are predicted in weaker lithological domains. The results for the study shows that sub level caving mining method should be implemented for the project and care must be taken to control dilution and mud rush.

DEDICATION

This work is dedicated to my late grandmother Denga Mphephu Ramabulana and my uncle Thompson Muvhango Mphephu Ramabulana. You are dearly missed. I also dedicate this work to my mother Masindi Mudau. To my wife Dakalo, my two daughters Vhugala-Hawe and Vhuthu-Hawe with lots of love and much appreciation.

ACKNOWLEDGEMENT

I would like to acknowledge my supervisor Mr Erhan Uludag who sacrificed his time to motivate, assist and coach me with completing this research report. To my colleagues at work, Richard Greig, Johnny Gillespie and Tshepo Mokele who believed in me and DeBeers consolidated Mines for allowing me time and sponsor my studies to pursue my research report.

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LIST OF ABBREVIATIONS

ADVK	Altered Dark Volcaniclastic Kimberlite
BC	Block Caving
BG	Gneiss Package
CAL	Calcrete
CAPEX	Capital Expenditure
CIM	Canadian Institute of Mining
CK	Coherent Kimberlite
CR	Country Rock
C&F	Cut and Fill
DBCM	De Beers Consolidated Mines
DOL	Dolorite
DVK	Dark Volcaniclastic Kimberlite
IC	Inclined Caving
FF	Fracture Frequency
JF	Joint Frequency
K01	Kimberlite 01
K02	Kimberlite 02
LOM	Life of Mine
L-T	Lezel Tina Fault
M	Metre
MBGL	Metres Below Ground Level
MBL	Marble
MMS	Mining Method Selection

MK	Macrocystic Kimberlite
MRMR	Modified Rock Mass Rating
MP	Metasedimentary Package
MVK	Massive Volcaniclastic Kimberlite
NPV	Net Present Value
OP	Open Pit
PMC	Palabora Mining Company
Q	Rock Quality Index
RMR	Rock Mass Rating
SLC	Sub Level Caving
SLR	Sub Level Retreat
UBC	University of British Columbia
UG	Underground
VK	Venetia Kimberlite
VKC	Venetia Kimberlite Cluster
VREP	Venetia Resource Extension Project
VUP	Venetia Underground Project
VUP_CSR	Venetia Underground Project Conceptual Study Report

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1 INTRODUCTION

1.1 Chapter Overview

Block caving is defined as a 'distinct caving method applied mostly to large, massive ore bodies because of its inherent low cost and high production capabilities' (Ataee-pour et al., 2008). It is carried out by undercutting large areas of a low-grade orebody so that it can naturally cave into the underground mine. The caved material is then removed by collecting it at draw points so that the ore can be taken for processing. As the ore is continually removed, the overlying material will continue to cave into the cut until all the overlying material has caved in (Carter, 2011). Because, many of these orebodies are located at greater depths, this continual caving is likely to cause overlying material to subside as the entire rock mass falls into the mine opening (Peskens, 2013).

Block caving creates drawpoints under the orebody rock mass and loading ore creating a void into which the orebody caves. The method results in a reasonable recovery of the orebody at a high production rate and low cost (Azadeh et al., 2010). According to Laubscher & Page (1990), block caving is recognised being the safest, lowest cost and highest production rate mining method available. It is regarded as the first option when considering the mining of massive, weak, vertical orebodies such as kimberlites (Siversten et al., 2010). Kimberlite pipes are suited for caving methods as they tend to be massive, vertical and circular in cross section (Ataee-pour et al., 2008). The key lies in having enough footprint to ensure that the hydraulic radius can be achieved thus delivering continuous caving. Historically, fragmentation has not been a problem with kimberlite.

According to Peskens (2013), sub level caving is applied to competent orebodies which are narrow and inclined with a waste rock hanging wall that will cave. Typically, parallel drifts close to each other are developed along the short axis of the orebody, then long hole drilled and blasted on retreat to the rim tunnel developed in the waste rock. The production rate is high due to the relatively simple mining cycle and large tonnage released per blast. However, according to Carter (2011), the challenge with the method lies in dilution management and resource recovery. Tunnel spacing, extraction sequencing and draw control is vital to ensure an acceptable recovery. Sub

level caving method is also associated with mud rush risks when the muck pile contains clay rich material and water ingress occurs.

Through a case study of Venetia mine, this study explores the decision process to select an underground mining method. Venetia mine is currently in the process of transitioning from Open Pit (OP) to Underground (UG) after the OP activities are planned to cease in 2021.

This chapter introduces the research subject with a background of the study as well as the problem statement. The chapter also comprises of the purpose of the study, goals and objectives, research design, significance as well as the limitations and delimitations of the study and an outline of the dissertation.

1.2 Background of The Study

Bitarafan & Ataei (2004) stated that Open-pit mining is the commonly used surface mining method and is regarded to be economically preferable compared to most underground mining methods. This is due to high recovery and production capacity. Alpay & Yavuz (2009) stated that the underground mining method benefits are more realised in the environmental impact since it is considered more acceptable than open-pit mining from the environmental degradation, social perspectives and often leaves a smaller footprint.

The open pit is mostly considered on the outcropping orebody or near surface deposit. The exploitation of such deposit will start as open pit and continue to such depth where the breakeven point is minimal and returns are not realised. Fuentes & Caceres (2004) stated in their studies that determining the accurate depth for open pit to stop is the out most significance. The depth of transition is of critical importance since it inform the management on the path to be taken to extend the life of the mine.

The last decade saw studies conducted on the transition from open pit to underground mining. Bakhtavar et al., (2009) gave account on the mines with the potential to use both open-pit and underground systems. I.e., Chuquicamata copper mine in northern Chile, Grasberg copper-gold mine in Indonesia, Diavik diamond mine in Russia, Kanowna Belle gold mine in Australia, Argyle diamond mine in Australia, Ekati

diamond mines in Canada, Meng-Yin diamond mine in China, Bingham Canyon in the USA, Mansa Mina in Chile, Venetia in South Africa, Mount Keith and Telfer in Australia, and Kiruna in Sweden.

Venetia Mine is thus the research ground for this study in assessing the mining method as it transits from open pit to underground in kimberlite diamond deposit mining. Venetia Mine is South Africa's largest producer of diamonds, thus maintaining the country's position as a prominent world producer (Read & Stacey, 2009).

1.3 Location and History of Venetia Mine

Siversten et al., (2010) stated that Venetia Mine is situated in the Limpopo province inherited its name from the farm on which it is situated, while its emblem, the guardian lion is the biblical symbol of Saint Mark, the patron saint of Venice. Venetia Mine is situated on the farm Venetia 103MS. The farm size is 880 hectares and situated approximately 27km south of the confluence of the Limpopo and Shashe. The mine lies 80km west of the town of Musina and 40km north-east of the farming community of Alldays. Venetia Mine is 700 metres above mean sea level and approximately 540km by road from Johannesburg and as shown in Figure 1.1.

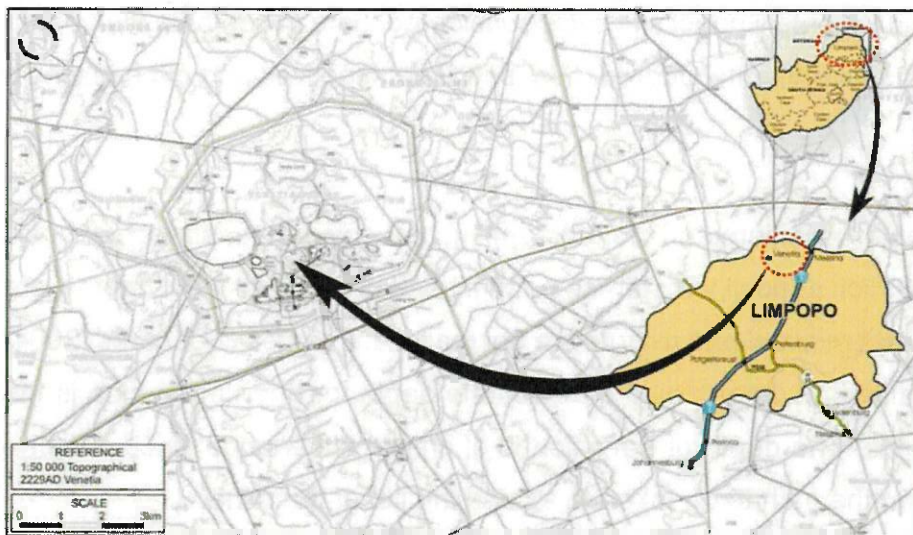


Figure 1-1: Location of Venetia Mine (Siversten et al., 2010)

Exploration for Venetia Mine began in the farm Seta as early as 1903 on the diamond bearing gravels discovered near the Limpopo river. De Beers began a reconnaissance programme in 1969 to locate the source of the deposits. A thorough evaluation began

the same year and full feasibility study was commissioned in 1988. Construction of the mine began in January 1990, with full output being achieved in December 1992 (Siversten, et al., 2010). Venetia has been the flagship of De Beers Consolidated Mines (DBCM) portfolio since achieving full production in 1992.

Two economical kimberlite pipes (K01 & K02) are currently mined in a single open-pit operation. K03 has been stopped from mining. The K03 material will form part of future opportunity studies during underground mining (Siversten et al., 2010). Venetia Mine employs a split shell mining method. This method allows for the merging of successive shell designs resulting in waste deferment and advancing revenue. The mining sequence for the different split pits shown in Figure 1.2.

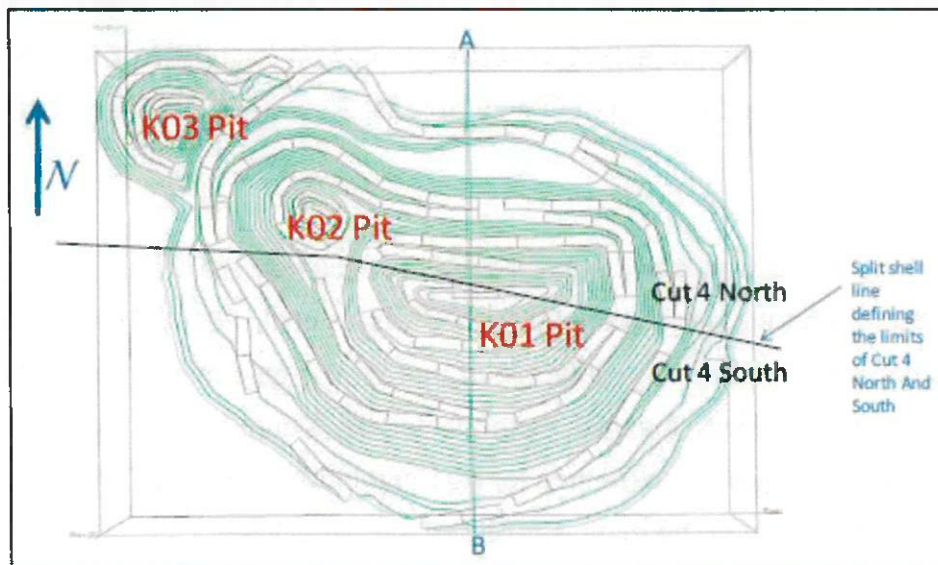


Figure 1-2: Venetia Mine pits and pit shell line (De Beers Consolidated Mines, 2015).

The open pit operation is currently mining ore from Cut 3 and stripping waste in Cut 4 south. The current production rate is 6 million tons per annum treated, producing 5 million carats. The past feasibility studies indicated that the mine would produce at 3.3 million tonnes per year and would transit into an underground operation in 2012 (Atkinson & Shchipansky, 2008). However, challenges with increased production rates, de-bottlenecking, and quick wins resulted in the originally defined 66 million tons of ore being depleted in 2007. Large mining equipment and improved mining costs led an increase in Cut size and a decision to mine Cut 4 from the OP. Currently, 80 million tons of ore have been mined from Venetia orebodies (Siversten et al., 2010).

1.4 Resources at Venetia Mine

There are two techno-economically viable ore bodies, namely K01 and K02. The other smaller satellite kimberlite pipes resources do not have enough value in the ground to support an underground operation. The Venetia Resource Extension Project (VREP) has confirmed resource below Cut 4 significant and could potentially allow underground to exist with approximately 100 million tons ore in the ground. The current plan for Cut 4 has a Life of Mine (LOM) until 2021. Whilst the underground operation is required to start ramp-up in 2021 for K02 and 2022 for K01 respectively.

Past financial modelling during the design phase has shown that extending the OP to Cut 5 is marginal at best and extends the LOM by seven years at most, while developing an underground mine shows a US \$1,120,174 million (United States of America currency, exchange rate applied as per indices published on 8 September 2010) positive NPV (stand-alone project) at a hurdle rate of 12% with IRR of 15.43% (Internal Rate of Return) and extends the LOM by at least 23 years (Siversten et al., 2010).

The conceptual study which was completed in 2009 investigated the viability of all possible mining methods for the Venetia ore bodies (Siversten et al., 2010). That work, coupled with this conceptual study, has determined that caving-type mining methods are the only techno-economically feasible methods for the Venetia resource, with Block Caving (BC) being the preferred method for K01 and Incline Caving (IC) for K02 (Mthombeni, 2006). The layout of the mine is reflected in Figure 1.3.

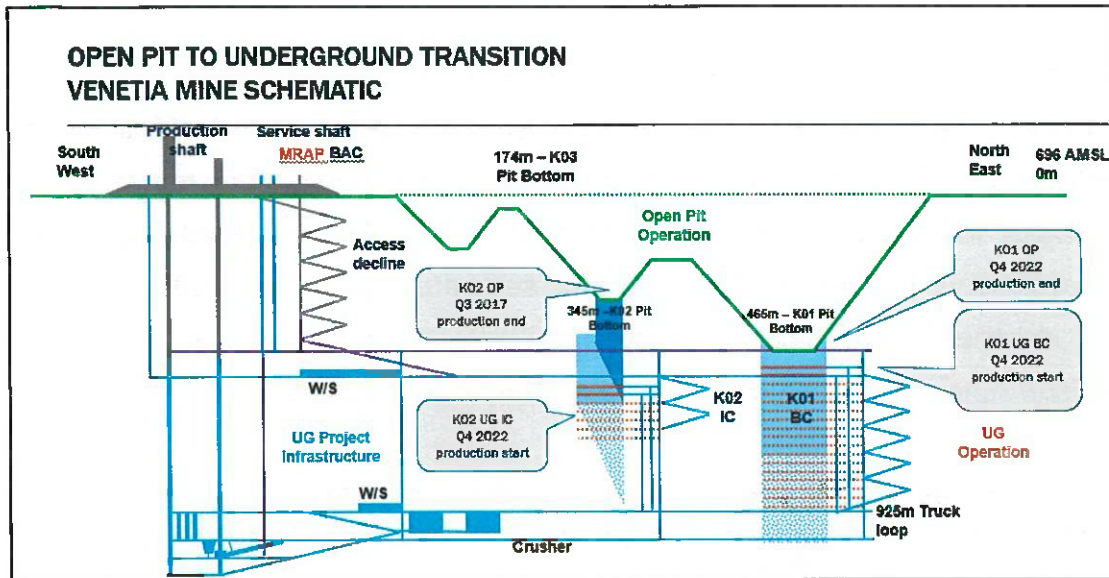


Figure 1-3: Mine layout (Mthombeni, 2006).

1.5 Geology

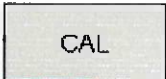
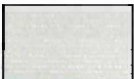
Venetia Kimberlite Cluster (VKC) were discovered in 1980, comprising of 15 known bodies (13 pipes and two dykes) (Tait, 2007). The cluster hosts different types of country rock comprising of biotite gneiss, quartzite, amphibolites and metasedimentary packages.

The two pipes referred in Section 1.4 are currently being exploited and covers an area of approximately 4km² in the Neo-archaeon Proterozoic Limpopo mobile belt (Tait, 2007). The VKC kimberlites and the nearby contemporaneous River Ranch and The Oaks kimberlites are all classified as Group I kimberlites (Seggie et al., 1999) or archetypal kimberlite (Mitchell, 1995). The VKC pipes range from 0.1ha to 12.7ha in size and are characterised by irregular elongated shapes, generally orientated west-northwest. An age of 519 million years, obtained from mica separates, has been determined for the Venetia kimberlite (Philips et al., 1999). There are other secondary resources which the mine has, and these includes, Blue Area Tailings, Red Area Tailings, and the Mix 03 stockpile.

Drilling operations have been conceded out to address the geotechnical and geological uncertainties. This programme started in the 1980 and used a range of drilling techniques. The purpose of VREP was to collect geological and geotechnical information to make an informed decision on the options after Cut 4. VREP drilling was completed in 2009 (Anglo American Technical Division , 2008)

Geological models have been created from the data collected and mapping exercise carried out by the mine personnel. Gemcom software was used to build kimberlite model. Barnett (2007) completed the model for the country rock and the model divides the geology into seven groups based on rock type (Table 1.1). Modelling of the different kimberlite facies is addressed in a separate report (Tait, 2007). Barnett (2007) identified seven main rock groups: calcrete, kimberlite, dolerite, pegmatite, Tina-Lezel zone, gneiss package, and metasedimentary package.

Table 1.1: The generalised stratigraphy column (Barnett, 2007).

Group		Rock Type	
	Calcrete		Tertiary Calcrete
	Kimberlite		All kimberlites
	Dolerite (DOL)		Karoo and Proterozoic mafic intrusives
	Pegmatite		Proterozoic granitic intrusives.
	Tina-Lezel Shear Zone		Cross-cutting brittle-ductile shears
	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
			Micaceous quartzite, often fuchsitic.
	Gumbu Group - Metasedimentary Package (MP)		Sillimanite and mica bearing, biotite-rich pelite
			Magnetite-rich quartzite
			Fine-grained carbonate, often interlayered with metapelite
	Gumbu Group - Marble (MBL)		Coarse marble

1.5.1 Kimberlite Pipe K01

The kimberlite pipe hosts the diamond ore which is found in more than 15 distinct bodies (typical pipes) across the cluster, each with variable facies, tonnages and

grade. The only pipe for direct concern is the largest pipe, K01 that forms the ore for the project. The K01 kimberlite pipe is a major deposit covers approximately 12.7ha and constitutes the largest single ore body in the VKC. The characteristics of the K01 kimberlite pipe is an irregular, elongated along its east-west axis, asymmetrical geometry with a distinct NNE–SSW-trending elongation on the eastern pipe edge. It is also an irregular bulge identified protruding to the north, at depth, in the North West corner of the pipe and steep sided with walls dipping at more than (Brown et al., 2007). Its contacts can vary from sharp to intensely brecciate and the kimberlite has been modelled to 1000m below surface.

The most volumetrically important kimberlite facies in the K01 pipe with respect to underground mine rock mass characterisation is the Dark Volcaniclastic Kimberlite (DVK). The DVK has sub-facies which include Altered Volcaniclastic Kimberlite (ADVK) (Figure 1-4). This alteration assumed to be a post depositional alteration that increases the clay content, making more susceptible to weathering and degradation of the rock strength. However, the clay enriched Kimberlite ADVK is not certainly weathered when drilled, they tend to weather when exposed to moisture. Mapping core samples and petrographic studies indicated overwhelmingly crystal population dominated by serpentine and carbonate pseudomorphs. Brown et al., (2007) had identified nine major lithofacies with further variations recognised within each of these. Brown et al., (2007) identified the Massive Volcaniclastic Kimberlite (MVK) as the most important lithofacie. The lithology is matrix-supported and crudely homogeneous though some local variations in the distribution pyroclast crystal and xenolith content are found. Crudely layered breccias have also been identified. These includes lithic lapilli, blocks and boulders in a fine to coarse grained ash matrix (Brown et al., 2007). The contacts between main pipe-fill lithofacies and breccias are generally sharp. A matrix-rich kimberlite, red to brown, outcrops along the western pipe margins, and is interpreted as re-sedimented extra-vent material. Siversten et al., (2010) stated that majority of the mining for the underground operation will be in DVK, with some mining of Coherent Kimberlite (CK), Volcaniclastic Kimberlite (VK) and Macrocytic Kimberlites (MK). Figure 1.4 indicates different facies within the orebody K01. No major challenges associated with the mining of DVK are anticipated.

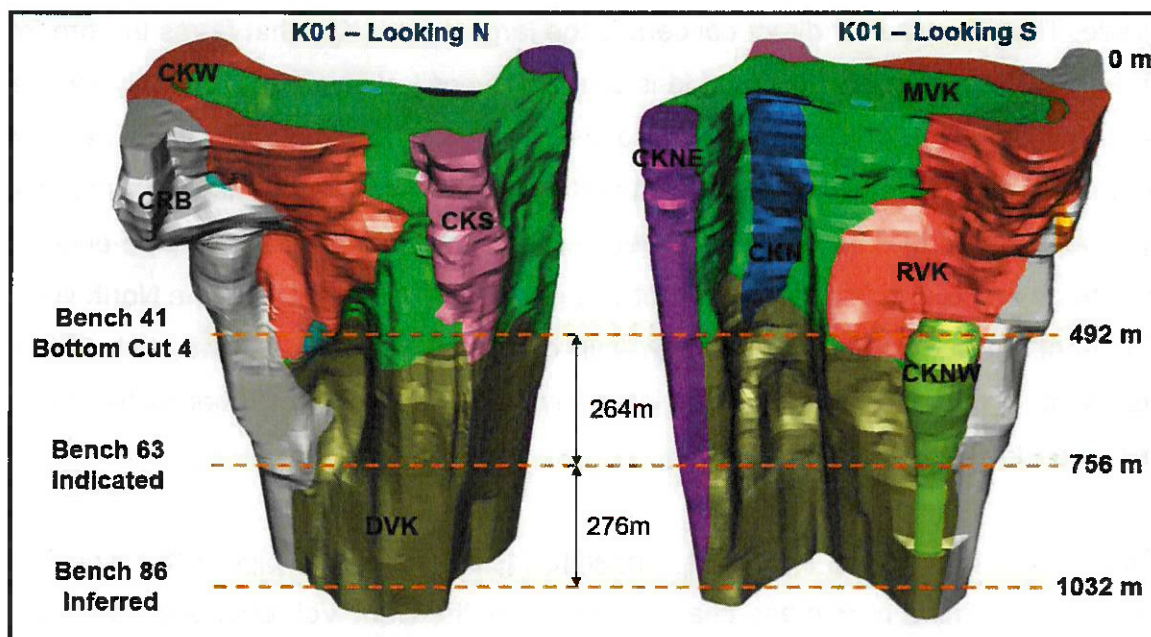


Figure 1-4: Northern and Southern view of the K01 orebody (Venetia Underground Project Conceptual Study, 2007)

Tait (2007) stated that there is presence of weathered horizons within K01 which are likely to present a challenge when adopting any mining method due to support requirements and fall of ground. This study analyses the challenges in selecting a mining method in K01. Siversten et al., (2010) allude that most of the underground mining will be done in the DVK facies and in which it is not exposed in the open pit.

1.6 Mining Method Selection

This research is based on the selection of the mining method which will best suitable to mine K01 kimberlite deposit. Siversten et al., (2010) stated that K01 is a massive and irregular in shape orebody which is likely to be a challenge with the mining method adopted. The conceptual study assumes (Venetia Underground Project Conceptual Study (2007)) block caving mining method as the most favourable mining method from the financial point of view. However, the application of block caving is not entirely supported with the existing information on the pipe footprint, ore and the country rock. There is significant risk that the mining method might not be suitable for the application. The assumed production, financial rate risks are mining method dependent and there is high probability that things may change as more study emerges.

Several mining method selection tools have been developed over time. All these methods reviewed by the researchers are associated with traditional qualitative or quantitative ranking systems. Carter (2011) stated that, mining method is a combination of variables relating to the exploitation process. Peskens (2013) describe parameters which are necessary when selecting mining method and includes the following:

- Engineering properties of the host rock and ore body;
- Production rate;
- Characteristics processing behaviour of ore;
- Geotechnical conditions including stress regime and rock mass rating strength;
- Economic factors including operating and capital costs combined with mineral price forecast.

1.6.1 K01 Kimberlite Characteristics

K01 kimberlite form part of two techno-economical orebodies which are planned to be mined underground at Venetia Mine. Its characteristics and behaviour have been researched by Siversten et al., (2010) and Barnett (2007). The K01 orebody profile covers an area of 12.7ha on land surface with vertical distance measured to be 1000mbgl (metres below ground level) (Siversten et al., 2010). It is irregular in shape, dipping at 80° and protruding to the north at depth.

Geological mapping and core sampling studies indicated nine lithofacies dominated by serpentine and carbon pseudomorphs after olivine. Brown et al., 2007 identified MVK which is a major lithofacies. Most of the underground mining excavation will be in VK, CK and MK (Siversten et al., 2010). More than 50% of kimberlite that will be mined in K01 orebody will have Uniaxial Compressive Strength (UCS) of more than 100MPa.

Mineralization throughout the orebody varies in grade content, with zones of high and low values and these zones are also intermixed which makes them difficult to separate. The indicated resources measured from 460mbgl to 756mbgl is 82 million carats with inferred resources sitting at 43 million carats. These resources are much significant for the success of Venetia underground project and bring the value of USD 110/t in the current present market.

1.6.2 Block Caving Mining Method for K01

Hamrin (1998) stated that Block Caving is a technique which utilises the conjunction of internal rock stresses to fracture rock mass into pieces that can flow by gravitational forces and handled by the miners. Caving of the material is induced by undercutting a rock and gravity forces millions of tons of rock to fractures and spread until the whole block is affected (Hamrin, 1998). Figure 1.5 shows the layout of block caving as gravity forces tons of orebody to flow down the draw points (Hamrin, 1998).

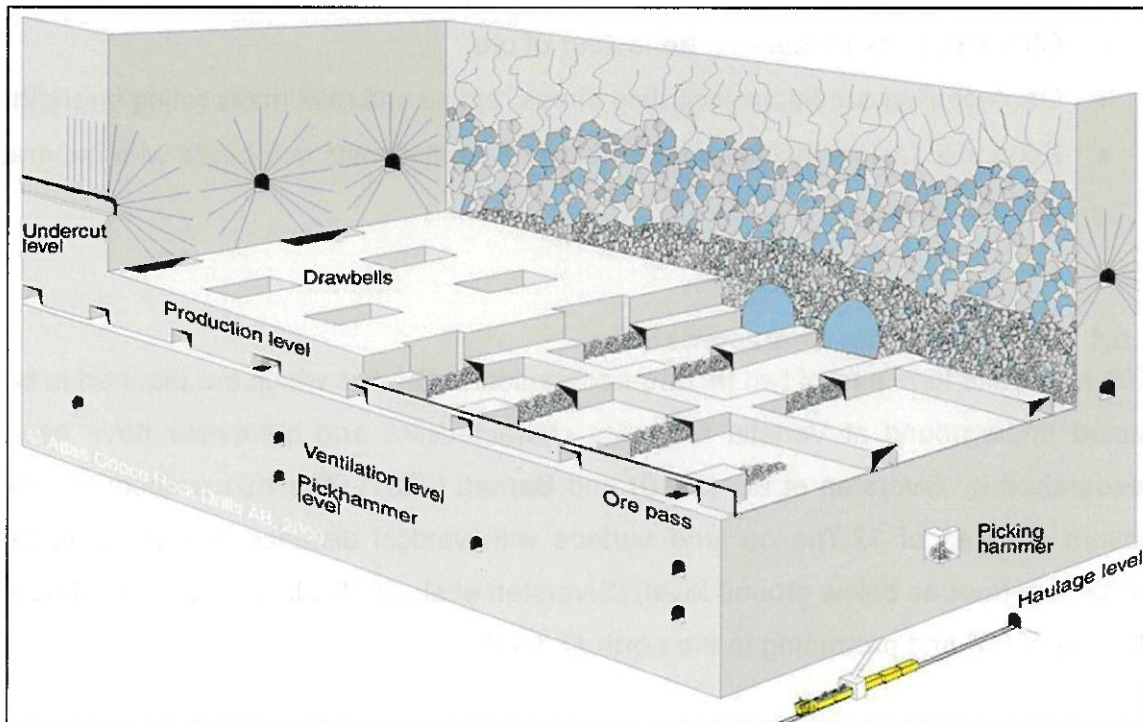


Figure 1-5: Typical layout of Block Caving (Hamrin, 1998).

Block caving is well understood and practiced in DeBeers operations, i.e.; Finsch and Cullinan mines (Siversten et al., 2010). The method seen as first option when considering mining massive, weak and vertical orebodies. The challenge with the method lies with the narrow orebody and discontinuous contacts. The other challenge is the widths of the orebody which limits achievable caving radius.

1.7 Problem Statement

Venetia Mine, in their transition to underground mining feasibility study has selected to use block caving in mining K01 kimberlite deposits. Block caving is only economic with substantial block heights due to the high capital cost of constructing the

production section. Due to the low horizontal footprint continuity, dilution issues from overhang and kimberlite losses in wedges; it is highly unlikely that the orebody will cave and deliver a reasonable resource recovery. Furthermore, when using multiple lift block caves in small orebodies, significant challenges are encountered when the transition from one lift to the other occurs as block caves have long ramp up and ramp down profiles. Since the lower cave will destabilise the upper cave, there will always be a production gap between caves.

The challenge in applying block caving to K01 is due to the narrow orebody shape and the highly discontinuous contacts. The width of the orebody limits the achievable caving radius to a maximum of 30m with the narrowest area in the east at 20m. On Laubscher's caving prediction chart, K01 requires a hydraulic radius of at least 35m, thus it requires at least a 140m by 140m square to be undercut to achieve continuous caving (Mthombeni, 2006). However, as the orebody average 120m wide, this is a significant challenge. K01 thus lies in the transitional zone and is neither stable nor caving (Mthombeni, 2006). A further challenge is that there are significant overhanging waste areas which will cave since the waste rock has a similar Rock Mass Rating (RMR) to the kimberlite. This will bring in early dilution. There are also significant amounts of kimberlite in the wedge areas which will be lost as the waste from the slope failure rushes in and cuts off the wedges. The problem is that when applying block caving to K01 orebody mining, it is difficult to structure drawpoints and bring in machines prompting this study to explore how to select the best caving method.

1.8 Purpose of The Research

The purpose of the research is to model the transition problem based on selecting the mining method to maximize the profit. This is the point at which NPV is maximized for an entire deposit by determining the spatial expanse and production quantities of both the open pit and underground mines while adhering to the operational production and processing constraints. Mine operators tend to delay the transition, leading to NPV losses of up to hundreds of millions of dollars.

This is an important decision during the life of a mine, because extraction during the transition period from open pit to underground affects the mine for the remainder of its operational life and creates further delay with regards to labour, machines, and equipment selection. Yet, some mines tend to apply methodological approach based on the previous work or similar projects done internationally or locally. It is important that this work should be carried out to alleviate the risks faced by Venetia Mine in selecting the mining method as they prepare to transit to the underground. The risk of selecting a mining method that is incapable of reaching mine target may lead to a financial loss. Eventually, this report could be used by mine planners and production personnel to consider incorporating into their conceptual studies, business plans and future massive mine projects.

1.9 Research Objectives

Block caving is the common underground mining method utilized in many massive mines deposit (Carter, 2011). However, when faced with a vertical orebody sub-level caving mining method is preferred instead of block caving (Carter, 2011). Henceforth, this study seeks to determine the most suitable and viable method when faced with a vertical orebody such as K01 through the following objectives:

- To determine a viable underground caving mining method for K01 deposits.
- To establish the challenges and benefits in caving method.
- To recommend best ways of exploiting K01 deposits.

1.10 Research Design

According to Magilvy & Thomas (2011, 151-155 p.), a research design “shows the research process in full of conceptualisation of the problem of the research, data generation, data analysis and dissemination of results and interpretation of findings”. The research is quantitative in nature and the detailed methodology outline is as follows:

- Review literature relevant to kimberlite orebody;
- Geotechnical consideration and support review for each mining method;
- Quantify the results obtained by mining method selection tool used to select the mining method for Venetia Mine;
- The data for the research report is available on site. The project is a green field done in the brown field operation;

1.11 Outline of the Dissertation

The report is structured into five chapters. Chapter 1 (Introduction) provides the study background, problem statement, objectives, and the research question. Chapter 2 (Literature review) presents arguments that relates to research question. Chapter 3 (Research methodology) highlights the method and procedure followed in collecting primary data. Chapter 4 (Results) present the results and discussion. Chapter 5 (Conclusion and Recommendation) present the conclusion emanating from the study, recommendations and future areas for further studies.

2 LITERATURE REVIEW

2.1 Chapter Overview

The purpose of this chapter is to elaborate on the current learning which influence the existing challenges with the selection of the mining method at Venetia Mine (VM). The existing data from geotechnical and geology department is used such that the K01 orebody can be extracted economical and safely. Then, literature review followed with the assumption of having in depth knowledge of the current challenges and obtain prerequisite knowledge to make an informed decision on the selection of the mining method.

Jakubec et al. (2017) stated that the primary diamond deposits have been mined on an industrial scale only within the past 150 years, mainly as open pit mines. Although underground mining followed soon after open pits reached technical and/or economic limits, it only gained significant recognition in the past 20 years except for Kimberley Mines. A relatively large number of underground mining methods were tested, i.e. block cave, sub level cave, open stoping etc. implemented, and evolved mainly in South African mines. In the mid-1990s, Alrosa started the development of the first underground diamond mine in Russia (Jakubec et al., 2017). Since then, underground mining has been implemented in several of their mines including Aikhal, Mir, and Udachny. China had also experimented with underground mining at Nhangma 701 Diamond Mine at the end of the nineties, but the largest development of underground diamond mining was experienced in Canada (Jakubec et al., 2017). Today, out of some 50 active diamond mines mining kimberlite, approximately 15 are underground and another 15 have underground plans or hold the potential to go underground (Jakubec et al., 2017).

The Koala north underground was formally opened and became North America's first underground diamond operation in 2002 (Jakubec et al., 2017). The Koala North pipe has been successfully mined as mechanised open benching down to 2115 (-350m) Level and proved that Sublevel Retreat (SLR) is technically feasible, and an economically viable mining method for exploiting kimberlite pipes in the Arctic (Jakubec et al., 2017). Since then, the Panda and Koala pipes were also mined as underground mines after open pits were completed, and several other pipes are being

investigated for underground mining. At the Ekati mine, three principal underground mining methods were utilised including SLR, SLC and IC mining.

It has been noted that consistent amount of research done on selection of mining method using common set of technical parameters for different orebodies. Nejad et al. (2014) describe the technical risk which influences project evaluation, geological resource model, geotechnical /hydrological model, mining method selection, and environmental and geopolitical risk. The application of these technical risks differs by site. Nejad et al. (2014) stated that region like Bingham Canyon has little seismic activity risk while others are characterised by high seismicity activity risk, i.e., El Teniente in Chile. Therefore, the selection of the mining method is limited to few candidates, because it is based on the known physical parameters and or the rock host. Some mines use more than one method, i.e., (Preece, 2001) stated that Finsch Mine employed different mining method for each block, i.e., block 1 and 2 were mined by combination of open pit and open stoping method respectively. Block 3 was exclusively mined by open stoping. Hence block 4 was started as open stoping and changed to block caving. Nejad et al. (2014) stated the reasons for change was inherent risk for dolomite sidewall and internal kimberlite failures.

Venetia mine considered block cave mining method based on the assumptions and history with De Beers' operations, see Table 2.1 (Siversten et al., 2010). Due to the lack of geotechnical information, an opinion was formed regarding the expected MRMR of the Venetia kimberlite K01 deposit at the depths of 640m (Keyser, 2005). The MRMR value of 40 was selected and was still needed to be quantified before a final figure would have made available. Laubscher stability graph shows a hydraulic radius of 20m after the MRMR of 40 was plotted, see Figure 2.1 below. Figure 2.2 and 2.3 show an idealised K01 structure at 640m and 790m below surface, respectively. There is potential of material not able to cave towards the edge of the orebody or radius not achieved for the material to cave. Figure 2.4 highlights the sections to which mining method will be applied, with first open stope to be utilised at the top section followed by block caving.

Table 2.1: Lesson learned from other massive mines (Herselman et al., 2009) and (Siversten et al., 2010).

Mining Principle	Method	De Beers Mine	External (lesson learned)
Natural Support	Open Stopping	Cullinan L1 & L2- Unsuccessful	No
	Open Benching	Finsch, Cullinan and Koffiefontein	Diavik & Ekati- Canada
Caving	Block Caving	Kimberly, Cullinan, Finsch, Koffiefontein-front cave	No
	Sub Level Caving	Kimberly,	Ekati- Canada
Artificial Support	Open Stopping or Post Cut & Fill. Cut & Fill Drill & Fill	No	Road header- Russia

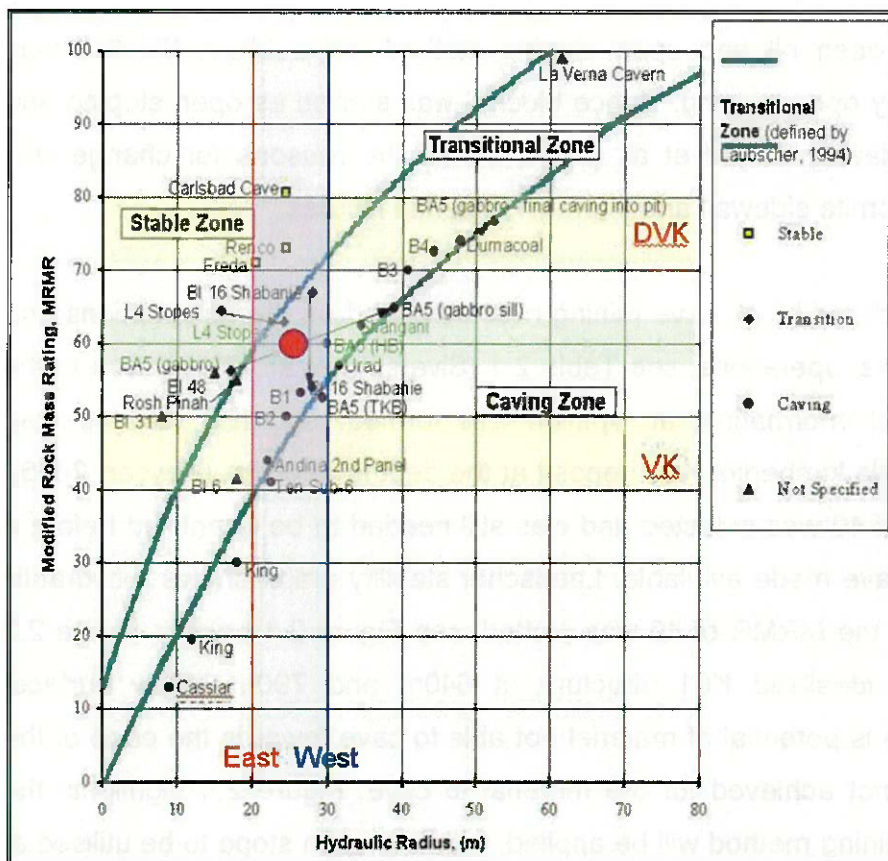


Figure 2-1: Caveability assessment using Laubscher's Curve (Herselman et al., 2009).

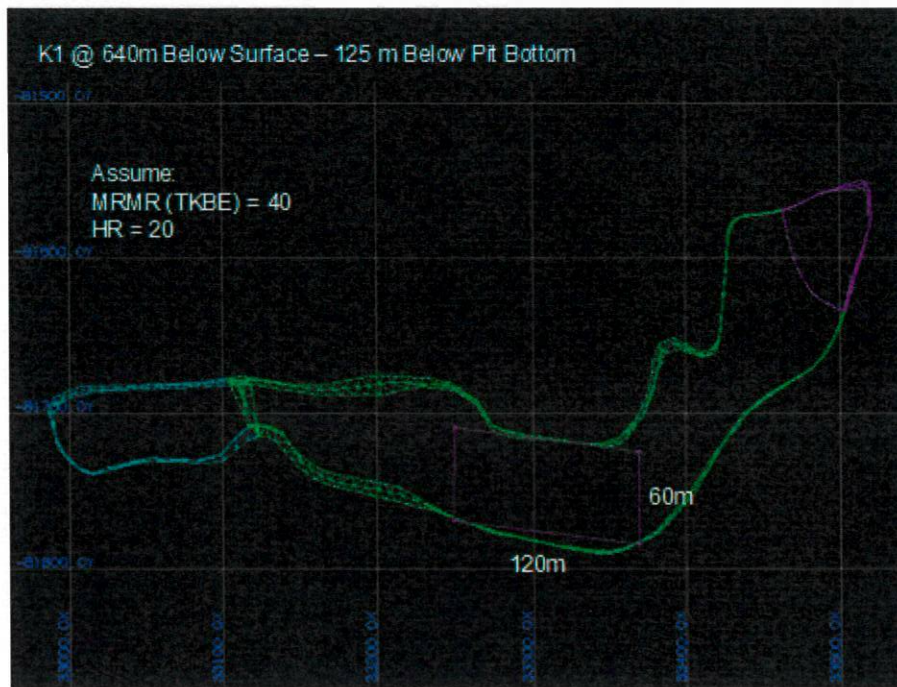


Figure 2-2: Level 640 below surface (Keyser, 2005).

At the depth of 640m, with the hydraulic radius of 20m block cave is most suitable mining method and more work need to be done to establish if it will be possible to cave through narrow section.

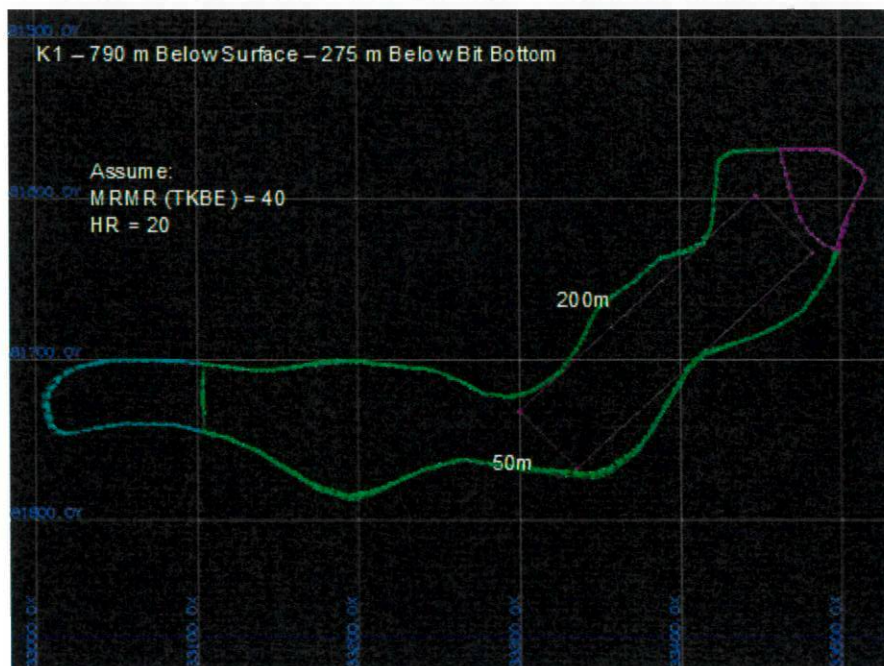


Figure 2-3: Level 790 below surface (Keyser, 2005).

At the depth of 790m, to achieve the hydraulic radius of 20m, the dimensions need to be 50m by 200m. There is a possibility that the material may not cave and even if it caves, it will be in the border line and may not cave properly. Then the decision was made by the project team to mine K01 in open stope at the crown pillar and block cave on the remainder of the ore body. Figure 2.4 illustrates sections which each mining method will be applied in K01 kimberlite.

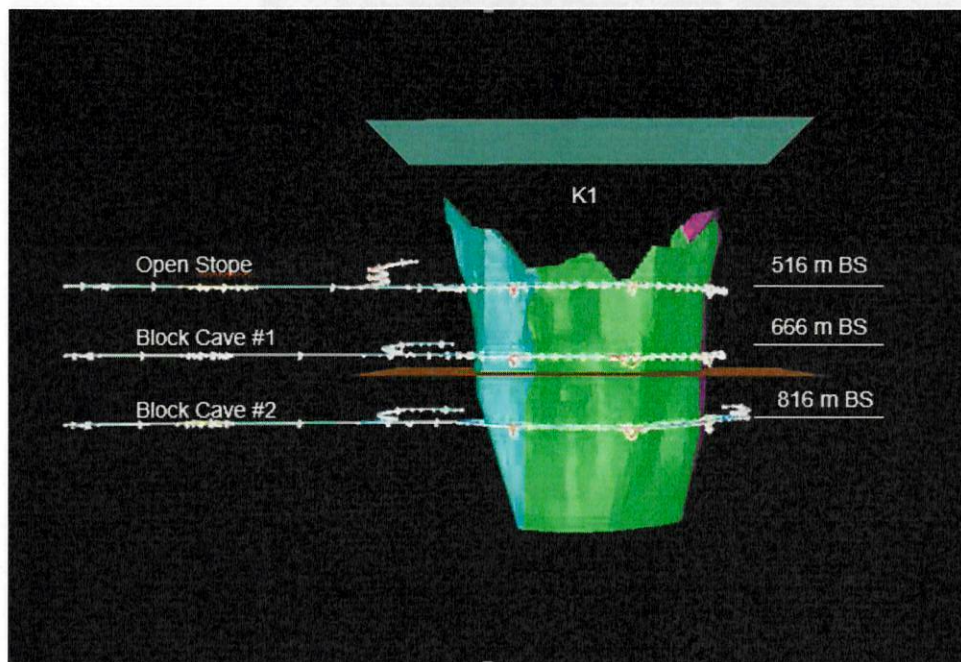


Figure 2-4: Cross section view, block cave mining method (Keyser, 2005)

The above scenarios prompt to further investigate which other mining method best suited for K01 orebody extraction. Often it is prudent to select the mining method that will work based on the data available, and if successful, the method can be modified as part of on-mine continuous improvement. The selection of incorrect mining method can be disastrous, as it cannot be easily changed once operational. The correct mining method can be chosen with wrong parameters applied, i.e. equipment's sizes, height of the panel size and draw point's interval. Nejad et al., (2014) stated that block caving is an inflexible mining method, once it is started, it cannot be changed.

2.2 Kimberlite (K01) Resource Description

There has been a numerous drilling campaigns initiated at Venetia Mine to address geological and geotechnical uncertainties. The campaigns started earlier during the start of open pit and used number of drilling techniques such as large diameter cable tool, coring and reverse circulation, air-flush and shallow percussion drilling. The different facies that have been modelled in K01 are depicted on Figure 1.4. The

classification of K01 as indicated to 750mbgl which is bench 63 and has been inferred to 1032mbgl. The inferred resources lack the bulk sampling information for revenue estimating. Table 2.2 illustrates the K01 orebody characteristics.

Table 2.2: K01 Orebody Characteristics

Kimberlite Density	Average Footprint Area	Average Orebody Length	Average Orebody Width
2.7t/m ³	64,034m ²	506m	127m

2.3 Structural Model for K01 Kimberlite

Aerial photography has been carried out around Venetia Mine to investigate major structural features. Figure 2.5 illustrate the two-dimensional model created from aerial photograph. The results from the left identifies metamorphic fabric and major geological features, and on the right shows all lineaments. A few holes intersected major geological structures. Goniometer was used to perform the structural measures from the oriented core. Photolog software was used to orient the data to true orientation and was provided by Dr Orpen (GroundTruth Ltd), (Siversten et al.,2010).

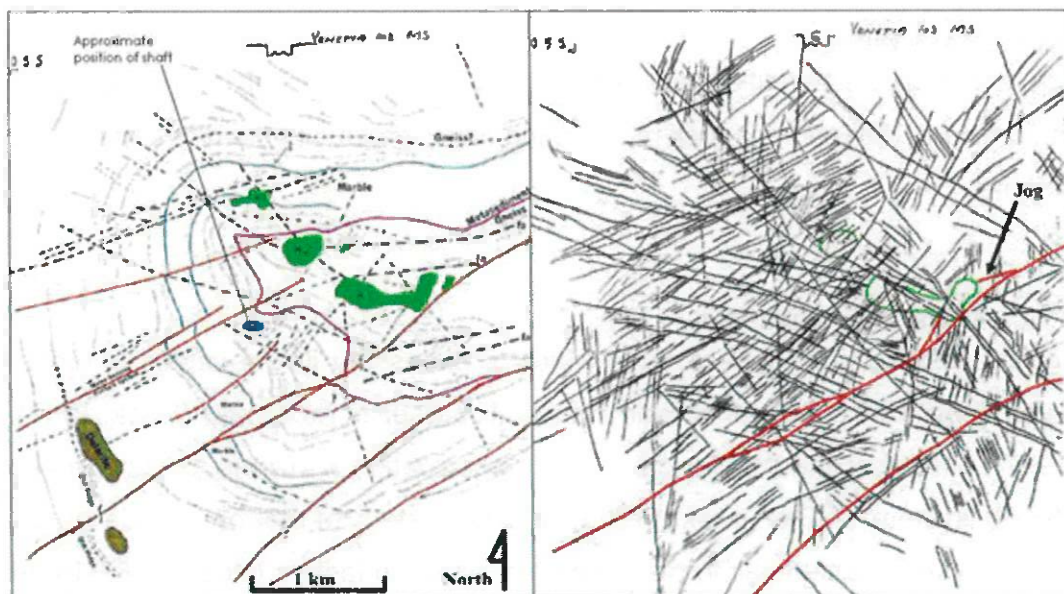


Figure 2-5: Aerial photograph (Barton, 1974)

Barnett (2007) stated that rock formation at Venetia Mine has gone through numerous phases of shearing and folding. Figure 2.6 illustrates the structural model with five

defined domains built in Gemcom (Barnett, 2007). The five domains illustrated are; Tina-Lezel shear zone, north, south, east, and west. Barnett (2007) has described the properties of such domains in details.

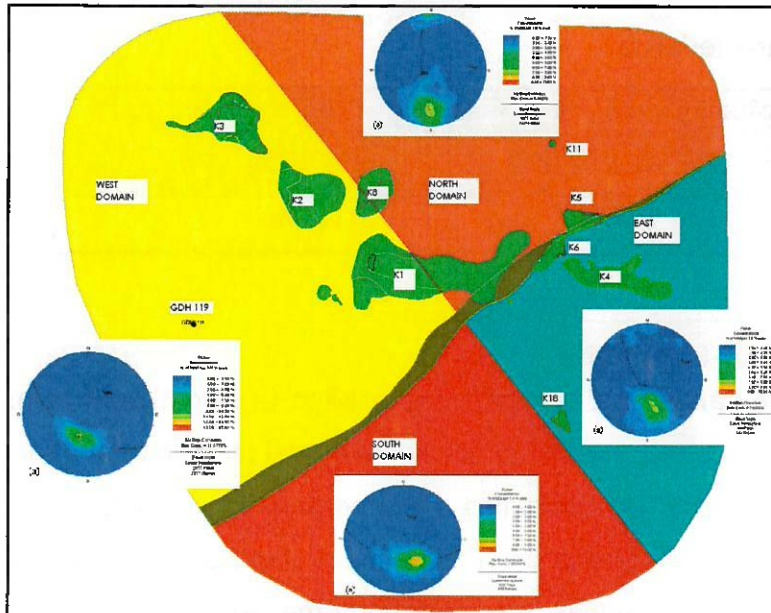


Figure 2-6: Structural domain (Barnett, 2007)

2.4 Geotechnical Considerations

2.4.1 Rock Mass Model

The rock mass model signifies all geotechnical properties including intact rock strength, structural defects, and rock mass rating of the various kimberlite and country rock domains.

2.4.2 Rock Mass Classification

Rock mass classification is a useful tool which is necessary in the design of tunnel support and, they force the users to examine the properties of rock mass (Hoek, 1987). Hoek, 1987 stated that the most widely used classification known is the rock mass rating (RMR) system of Bieniawski (1989) and Q system of Barton, Lunde et al., (1974). Each system develops information on the following factors (Hoek, 1987) listed below:

- Spacing;
- Surface properties and structural discontinuities;

- Intact rock material;
- Subsurface ground water;
- In-situ stress;
- Orientation;
- Inclination of the support requirements.

Tables 2.4 to 2.9 illustrates how the RMR system is determined from the list describe by Hoek (1987). The common rock mass classification system which are often used in rock engineering to assist with designing underground structures are, Rock Mass Rating (RMR), Geological Strength Index (GSI) and Rock Mass Quality (Q). Below listed are common systems used in various operation in Table 2.3.

Table 2.3: Rock Mass Classification Systems (Cosar, 2004).

Rock Mass Classification System	Originator	Country of Origin	Application Areas
Rock Load	Terzaghi, 1946	USA	Tunnels with steel Support
Stand-up time	Lauffer, 1958	Australia	Tunneling
New Austrian Tunneling Method (NATM)	Pacher et al., 1964	Austria	Tunneling
Rock Quality Designation (RQD)	Deere et al., 1967	USA	Core logging, tunneling
Rock Structure Rating (RSR)	Wickham et al., 1972	USA	Tunneling
Rock Mass Rating (RMR)	Bieniawski, 1973 (last modification 1989-USA)	South Africa	Tunnels, mines, (slopes, foundations)
Modified Rock Mass Rating (M-RMR)	Ünal and Özkan, 1990	Turkey	Mining
Rock Mass Quality (Q)	Barton et al., 1974 (last modification 2002)	Norway	Tunnels, mines, foundations
Strength-Block size	Franklin, 1975	Canada	Tunneling
Basic Geotechnical Classification	ISRM, 1981	International	General
Rock Mass Strength (RMS)	Stille et al., 1982	Sweden	Metal mining
Unified Rock Mass Classification System (URCS)	Williamson, 1984	USA	General
Communication Weakening Coefficient System (WCS)	Singh, 1986	India	Coal mining
Rock Mass Index (RMI)	Palmström, 1996	Sweden	Tunneling
Geological Strength Index (GSI)	Hoek and Brown, 1997	Canada	All underground excavations

Table 2.4: RMR parameters and their ratings (Bieniawski, 1989).

PARAMETER		RANGE OF VALUES						
STRENGTH OF INTACT ROCK MATERIAL	POINT-LOAD STRENGTH INDEX (MPa)	>10	4-10	2-4	1-2	For this low range – UCS test is preferred		
	UNIAXIAL COMPRESSIVE STRENGTH (MPa)	>250	100-250	50-100	25-50	5-25	1-5	<1
RATING		15	12	7	4	2	1	0
DRILL CORE QUALITY RQD (%)		90-100	75-90	50-75	25-50	<25		
RATING		20	17	13	8	3		
JOINT SPACING (m)		>2	0.6-2	0.2-0.6	0.06-0.2	<0.06		
RATING		20	15	10	8	5		
JOINT CONDITIONS (See Table 2.8)		Very rough surfaces, Not continuous, No separation, Unweathered wall rock	Slightly rough surfaces, Separation <1mm, Slightly weathered walls	Slightly rough surfaces, Separation <1mm, Highly weathered walls	Slickensided surfaces or Gouge <5mm thick or Separation 1 to 5mm Continuous	Soft gouge >5mm thick or Separation > 5mm Continuous		
RATING		30	15	10	8	5		
GROUND WATER CONDITIONS	Inflow per 10m tunnel length (l/minute)	None	<10	10-25	25-125	>125		
	(Joint water pressure)/ (Major Principal Stress)	0	<0.1	0.1-0.2	0.2-0.5	>0.5		
	General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
RATING		15	10	7	4	0		

Table 2.5: Adjustment rating for discontinuity orientations (Bieniawski, 1989).

STRIKE AND DIP ORIENTATIONS		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
RATINGS	Tunnels and mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

Table 2.6: RMR determined from total ratings (Bieniawski, 1989).

CLASSIFICATION CRITERIA	CATEGORIES				
RATING	100-81	80-61	60-41	40-21	<21
CLASS NUMBER	I	II	III	IV	V
DESCRIPTION	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

Table 2.7: RMR rock classes (Bieniawski, 1989).

CLASSIFICATION CRITERIA	RANGE OF VALUES				
Class number	I	II	III	IV	V
Average stand-up time	20yrs for 15m span	1 year for 10m span	1 week for 5m span	10hrs for 2.5m span	0.5hrs for 1m span
Cohesion for rock mass (kPa)	>400	300-400	200-300	100-200	<100
Friction angle of rock mass (deg.)	>45	35-45	25-35	15-25	<15

Table 2.8: Classification of joint conditions (Bieniawski, 1989).

JOINT FEATURE	RANGE OF VALUES				
Discontinuity length (persistence)	<1m	1-3m	3-10m	10-20m	>20m
RATING	6	4	2	1	0
Separation (aperture)	None	<0.1mm	0.1-1mm	1-5mm	>5mm
RATING	6	5	4	1	0
Roughness	Very rough	Rough	Slightly rough	Smooth	Slicken-sided
RATING	6	5	3	1	0
Infilling (gouge)	None	Hard filling <5mm	Hard filling >5mm	Soft filling <5mm	Soft filling >5mm
RATING	6	4	2	2	0
Weathering	Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed
RATING	6	5	3	1	0

Table 2.9: Properties of discontinuity on the strike and dip orientation in the RMR system (Bieniawski, 1989).

STRIKE PERPENDICULAR TO TUNNEL AXIS		STRIKE PARALLEL TO TUNNEL AXIS	
Drive with dip: Dip 45-90°	Drive with dip: Dip 20-45°	Dip 45-90°	Dip 20-45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip: Dip 45-90°	Drive against dip: Dip 20-45°	Dip 0-20° Irrespective of strike angle	
Fair	Unfavourable	Fair	

2.4.3 Rock Quality Designation Index (RQD)

RQD is the total percentage of intact core pieces longer than 100mm and with diameter of 54.7mm (Hoek, 2007). Palmstrom (1982) stated that RQD can also be estimated from the number of discontinuities using the following

$$RQD = 115 - 3.3J_v \dots\dots\dots \text{Equation 2-1}$$

where J_v is a volumetric joint count and is defined as the sum of joints per unit length (Deere et al., 1967). RQD depend on borehole direction and volumetric joint is useful in reducing the directional dependence. Table 2.10 illustrates values of joint fracturing

and the greater the value of RQD, the better the rock mass value. Figure 2.7 illustrates procedure for calculation of RQD.

Table 2.10: Rock Mass Quality classification demonstrating the impact of RQD (Deere et al., 1967)

RQD	Rock Mass Quality
< 25	Very poor
25 – 50	Poor
50 – 75	Fair
75 – 90	Good
99 – 100	Excellent

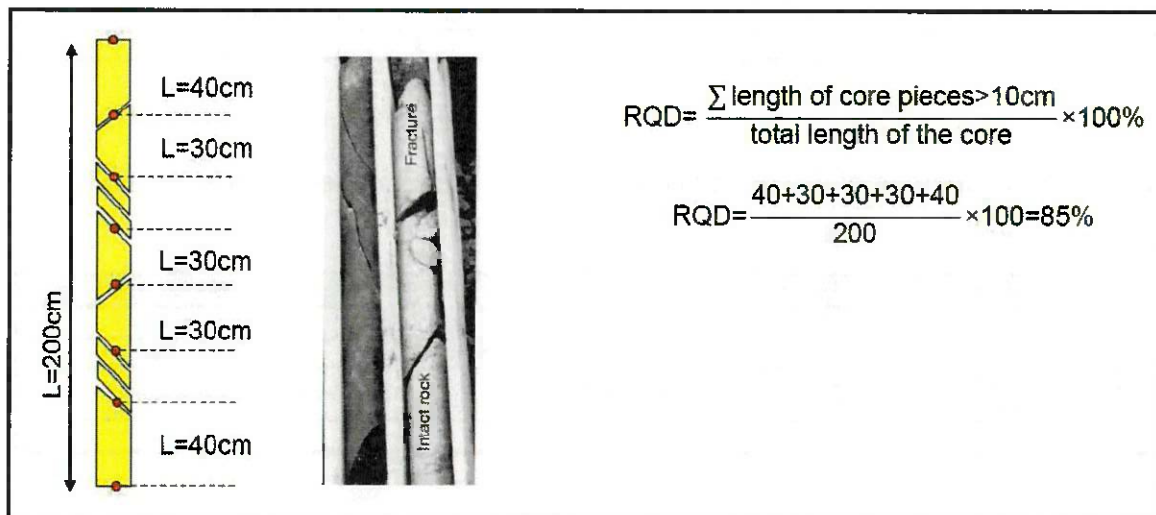


Figure 2-7: Measurement and calculation of RQD (Deere et al., 1967).

Palmstrom & Broch (2006) provide a brief account on the limitation on RQD which includes the following factors:

- It considers only the extent of rock fracturing;
- It doesn't account for rock strength;
- It depends on line orientation to preferential orientation of discontinuities;
- It cannot account for length of considered joint;
- It is insensitive when the frequency is greater than 3m^{-1} .

2.4.4 Rock Structure Rating (RSR)

Wickham et al. (1972) developed the RSR as a quantitative method based on the histories of tunnels supported with the steel sets. The technique used for describing the quality of the rock mass and selecting support for small tunnels. Hoek (2007) stated two factors influencing the rock mass behaviour in RSR system which are geological and construction parameters. Construction parameters includes size of the tunnel, direction of the drive, and method of excavation (Bieniawski, 1989).. Wickham et al. (1972) developed a numerical formula to compute the RSR.

$$RSR = A + B + C \dots\dots\dots \text{Equation 2-2}$$

$$RSR=A+B+C$$

Where:

- 1) Parameter A, Geology: General evaluation on geological structure:
 - a. Rock type;
 - b. Rock hardness;
 - c. Geologic structure.

- 2) Parameter B, Geometry: Consequence of discontinuity to the direction of the tunnel:
 - a. Joint orientation;
 - b. Joint spacing;
 - c. Direction of the tunnel drive.

- 3) Parameter C, Ground water: Evaluation on joint condition:
 - a. Amount of water inflow;
 - b. Joint condition;
 - c. Rock mass quality;

2.4.5 Modifications in RMR for Mining

RMR was proven based on civil engineering application (Bieniawski, 1989). and numerous of modifications have been successful applied (Laubscher, 1981). These modifications were done to ensure that the RMR system is relevant to the mining industry. The modified RMR is known as Laubscher's MRMR system (Laubscher,

1981). The modified RMR considers to cater for in situ and mining induced stresses, stress variations and the effects of blasting and weathering. Hoek (2007) stated that MRMR system is based on case histories in the caving operations. Therefore, the MRMR system was used in this study to define if there will be caving in the K01 kimberlite. Figure 2.8 highlights the graphical relationship derived by Diering & Laubscher (1987) between MRMR and the shape factor of the stope/tunnel (i.e. hydraulic radius). The hydraulic radius is a ratio of area to perimeter of an excavation's face.

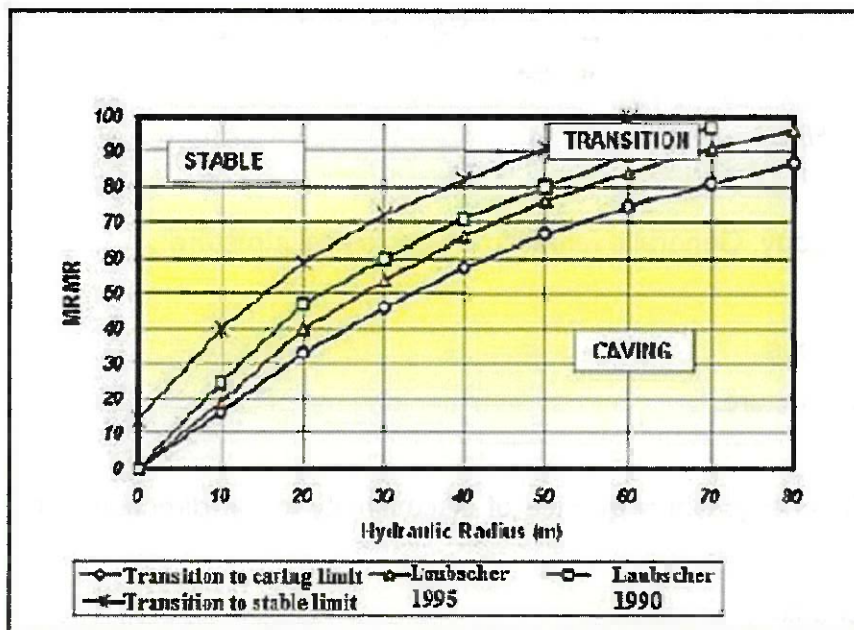


Figure 2-8: Laubscher's Stability Graph (Laubscher & Jakobs, 2001).

2.4.6 Rock Strength Model

The tests which were conducted for geotechnical samples includes; Brazilian tensile strength, base friction angle of major lithology, density measurements, uniaxial compressive strength, slake durability test, tri-axial compressive strength, and direct shear test on joints. The statistical summary of uniaxial compressive strength of all rock types presented in Table 2.11. Ratio of all mean standard deviation is below 25% (SRK Consulting, 2000). Table 2.12 illustrates summary of the UCS of the country rock type (SRK Consulting, 2000).

Table 2.11: Summary of the UCS of country rock (SRK Consulting, 2000).

Lithology Group	Rock Type - Code	Count	Min	Max	AVG	STD	(STD / AVG)%
Dolerite	DOL	22	32.5	42.3	39.21	2.31	6%
Shear Zone	SZ	0	0	42.3	39.21	2.31	6%
Gneiss Package	Biotite Gneiss - BG	89	28.4	40.8	37.11	3.23	9%
	Biotite Schist - BS	83	26.3	40.6	34.56	4.41	13%
	Amphibolite - AM	81	29.2	41.2	38.55	2.79	7%
	Average for the package				36.74		
Metasedimentary Package	Quartzite - QTZ	15	30.6	37.5	34.98	1.65	5%
	Metapelite - MP	2	29.4	29.9	29.65	0.35	1%
	Metacarbonate - MC	0	0	0			
	Average for the package				33.79		
Marble	Marble - MBL	10	19.60	40.8	35.00	8.34	24%
Kimberlite	Volcaniclastic Kimberlite - VK	0	0.00	0			
	Dark Volcaniclastic Kimberlite - DVK	0	0.00	0			
	MK	0	0.00	0			

There is a significant variation in the western side of the pipe. The weathered horizons signify the lower UCS values as shown in Figure 2.9 below.

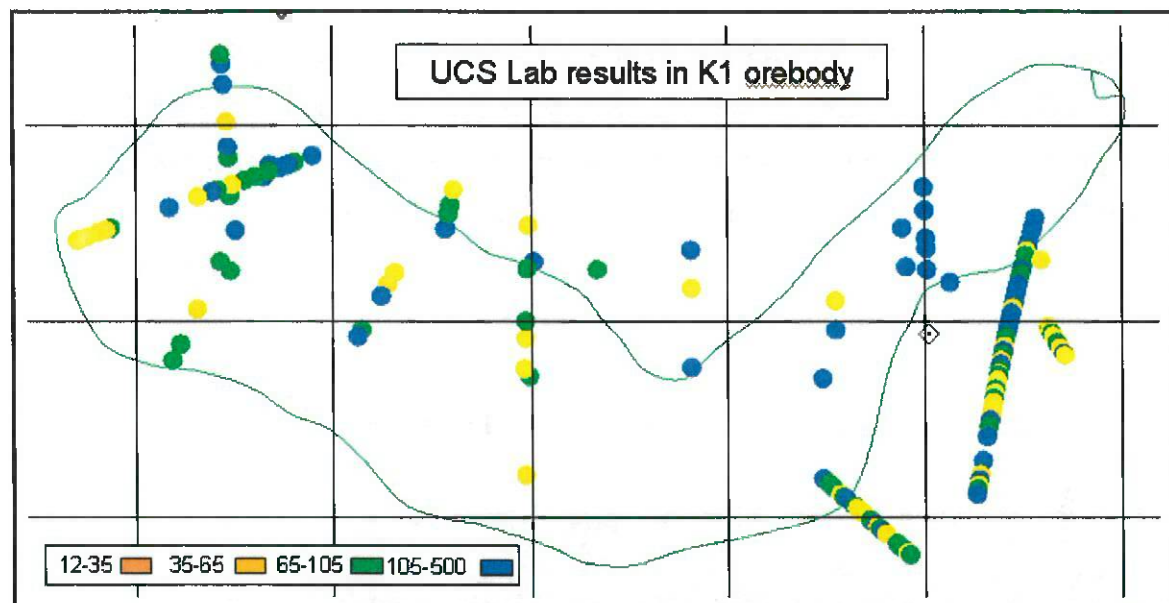


Figure 2-9: Spatial distribution for UCS for K01 below Bench 40 (SRK Consulting, 2000).

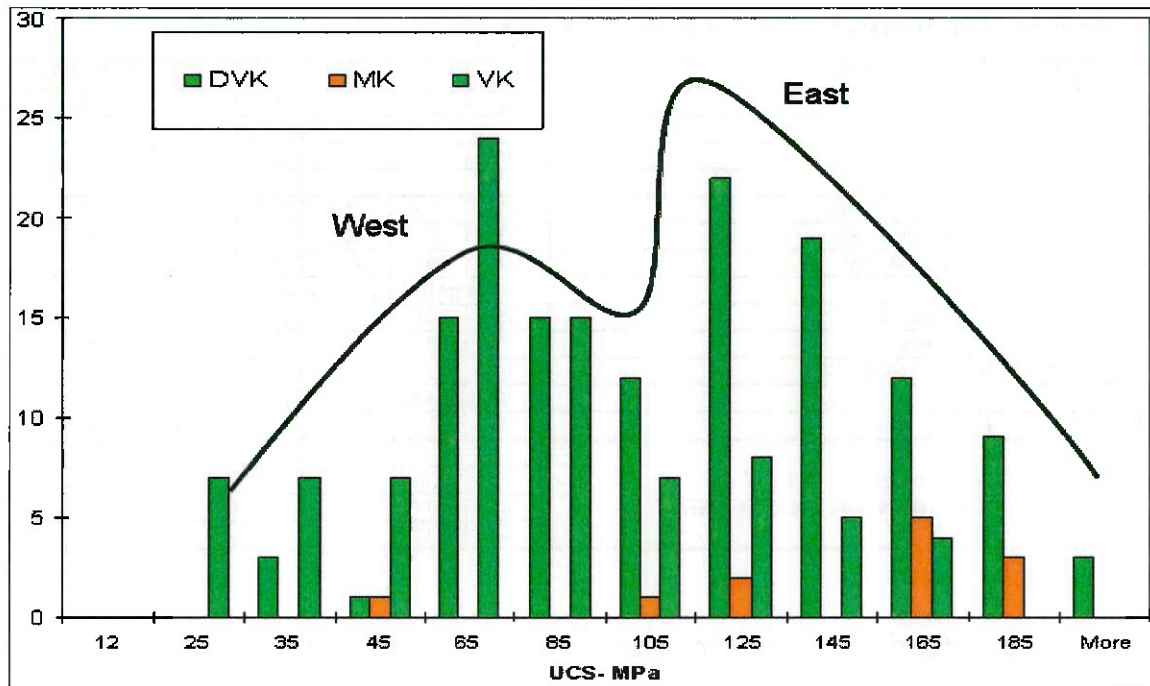


Figure 2-10: Distribution of UCS for various K01 lithologies (SRK Consulting, 2000).

Various lithologies are illustrated in Figure 2.10 and 50% of kimberlite will be mined have UCS value of over 65MPa. DVK has higher UCS compared to MK. The advantage of having 50% of MK deposit will be realised in the plant, as the system is designed for such UCS values. There is currently an ongoing project for plant optimisation to deal with DVK and VK. Figure 2.11 illustrates UCS data for different diamonds mines in South Africa (SRK Consulting, 2000).

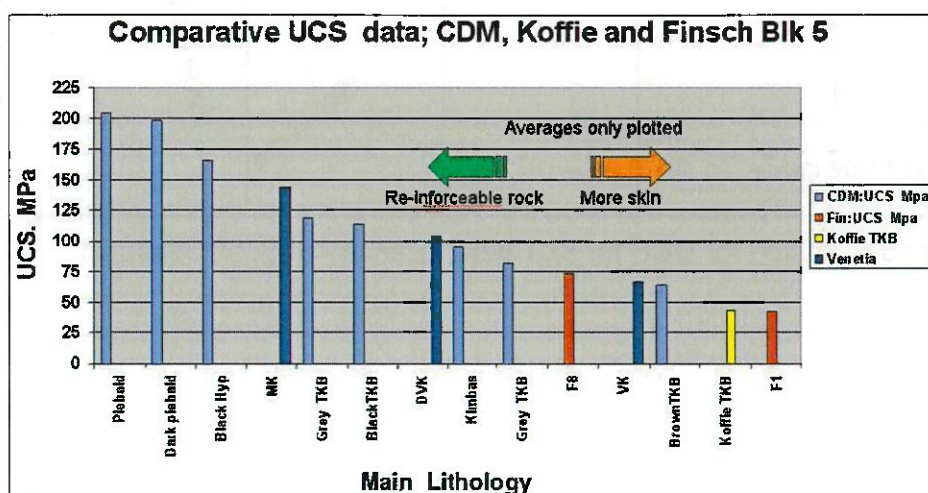


Figure 2-11: Comparative study on UCS (SRK Consulting, 2000).

Table 2.12: Average rock properties and UCS for country rock (SRK Consulting, 2000).

Lithology Group	Density (g/cm ³)	UCS(MPa)	Tensile Strength (MPa)	Elastic Modulus (GPa)	Poisson's Ratio
Gneiss Package	2.79	65	12	62.5	0.27
Marble Package	2.84	125	7	81.1	0.37
Metasediments Package	2.83	140	12.5	71	0.29
Dolerite	2.9	267	19	86.8	0.28
K01 DVK (Hard)	2.76	134	11	47.8	0.31

2.5 Selecting Mining Method Technique (MMT)

The purpose of this section is to present logical steps to select the mining method for K01 orebody which is the major deposit (averaging 100Mpa) for the success of the Venetia Underground Project (VUP). Part of the critical decision on selecting a mining method is to look at the rock properties under consideration (Chapter 1, section 1.5 and 1.6.1, and Chapter 2, section 2.2, and 2.3). Usually, merely one method may work best per each situation (Nejad et al., 2014), while others must be a compromise in safety, cost and productivity. Nicholas (1981) stated characteristics that have major impact on mining method selection (MMS) which includes the following:

- Geological and physical characteristics of the orebody;
- Ground conditions;
- Capital and Mining costs;
- Labour and availability (Nejad et al., 2014);
- Environmental consideration;
- Mining rate.

Carter (2011) stated that, the key influence of selecting a mining method is the style of mineralization, strength, and rock mass. Carter (2011), further insinuated that economic factors play a vital role in mining method selection. Similarly, Azadeh et al., (2010) noted that combining the economic and technical models into one model would lead into better trade-off for selecting the mining method. The optimum goal is to find a method that is efficient to use extracting the mineral resource to achieve better economic return with minimum impact on the environment (Siversten et al., 2010). Several qualitative and quantitative techniques have been developed to evaluate the

mining methods for an ore deposit based on physical characteristics of the deposit, plunge, depth and grade distribution (Bakhtavar, et al., 2015). Soderberg & Rauch (1968) came up with the concept of break-even stripping ratio, Lane (1964) study on optimum cut-off grade and Laubscher (1981) work on the concept of selecting MMS on RMR. Laubscher methodology on caveability was used in this project to determine the MMS for K01 deposit. Other research done on MMS are summarised in Table 2.13 below.

Table 2.13: Summary of research works on MMS (Dehghani et al., 2017).

Researcher	Year	Researcher	year
Peel and Church	1941	Hamrin	1998
Boshkov and Wright	1973	Tatiya	1998
Morrison	1976	Basu	1999
Nicholas and Mark	1981	Kahriman	2000
Loubscher	1981	Kesimal and Bascetin	2002
Karabeyoglu	1986	Clayton et al.	2002
Hartman	1987	Yiming et al.	2003
Bandopadhyay and Venkatasubramanian	1987	Samimi namin et al.	2003
Marano and Everitt	1987	Yiming et al.	2004
Agoshkov et al.	1988	Samimi namin et al.	2004
Camm and Smith	1992	Mihaylov	2005
Nicholas	1993	Bascetin	2005
Mutagwaba and Terezopoulos	1994	Miranda and Almeida	2005
Miller et al.	1995	Shahriar et al.	2007
Gershon et al.	1995	Karadogan et al.	2008

Several MMS tools have been developed over the years. This section will give an introduction in to the different MMS tools.

2.5.1 Morrison Method

Morrison (1976) approach was dividing underground mining methods into three groups, i.e. rigid pillar support; controlled subsidence; and caving. According to Carter, (2011) the factors influencing the selection are, ore width, support type, and strain energy accumulation. Morrison (1976) approach was proposed a qualitative technique which was specifically for underground mines. The factors considered are ore plunge, ore thickness, strength of ore and walls. Figure 2.12 illustrates Morrison selection criteria for MMS.

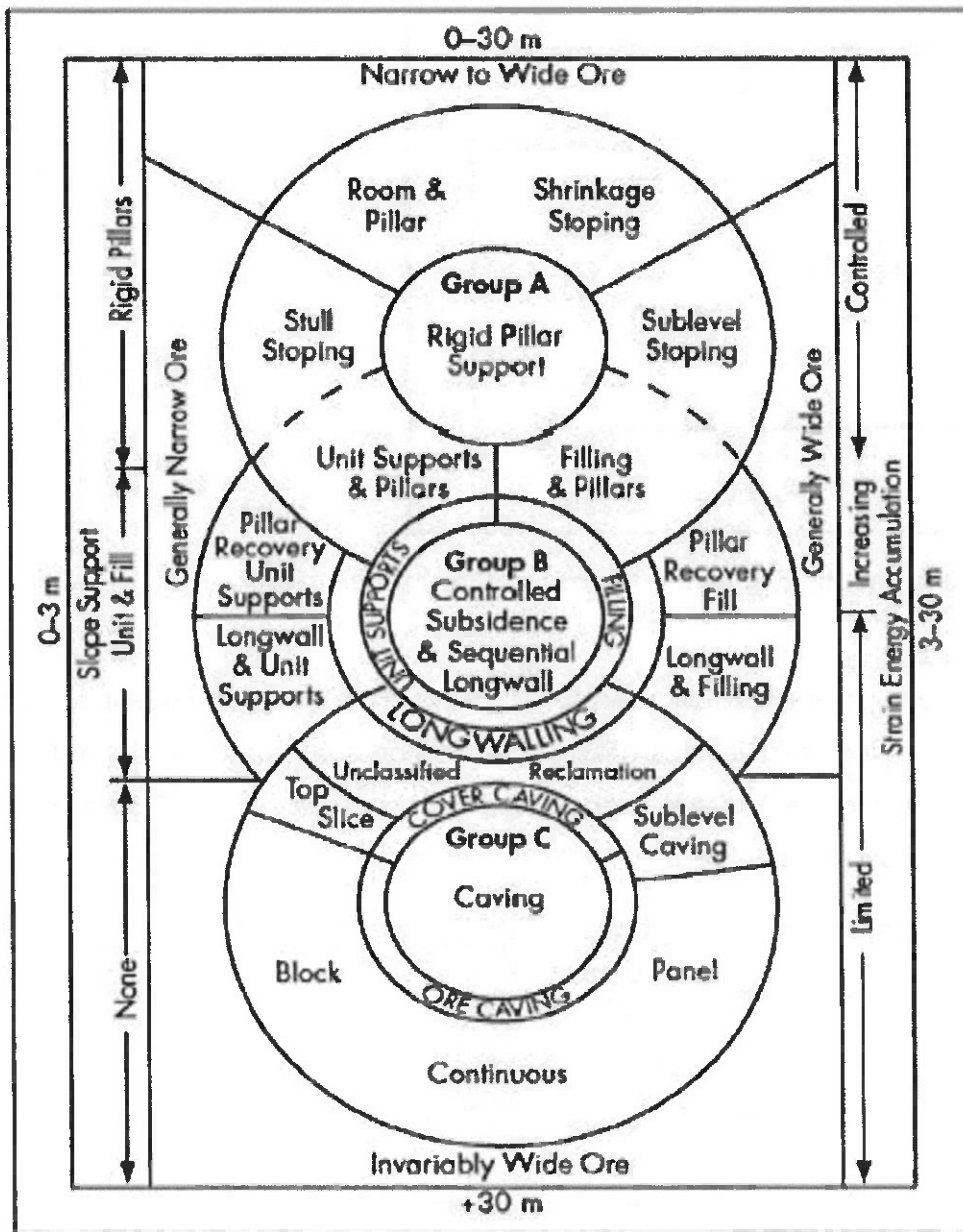


Figure 2-12: Morrison selection criteria for MMS (Morrison, 1976)

2.5.2 Laubscher Selection Method

Laubscher (1981) approach technique use RMR of a rock mass similar to UBC. The difference between Laubscher and UBC is that the latter method included hydraulic radius, Laubscher and Page (1990). This method is specifically for competent rock masses in mass mining, Carter (2011).

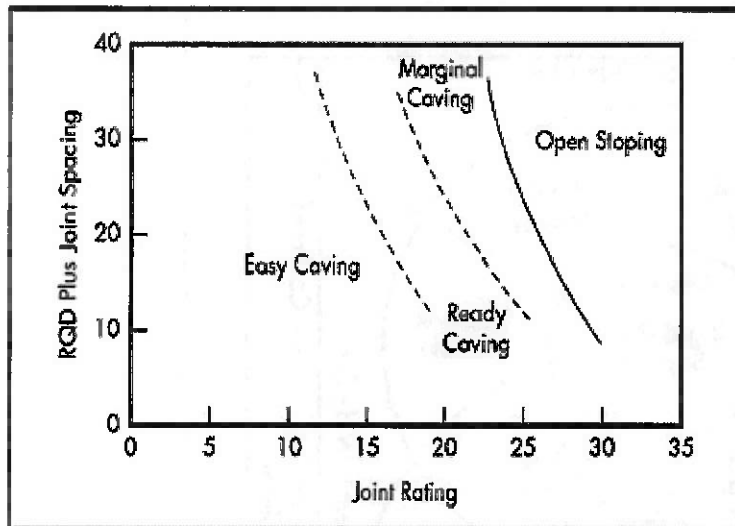


Figure 2-13: Laubscher mining method selection (Laubscher, 1981).

2.5.3 Hartman Method

Nicholas (1987) approach uses flow chart for MMS which is based on geometry and ground conditions of the ore (Nicholas, 1981). This method is qualitative, and the chart selects specific mining method for underground and surface mines. Figure 2.14 illustrate Hartman flow chart below.

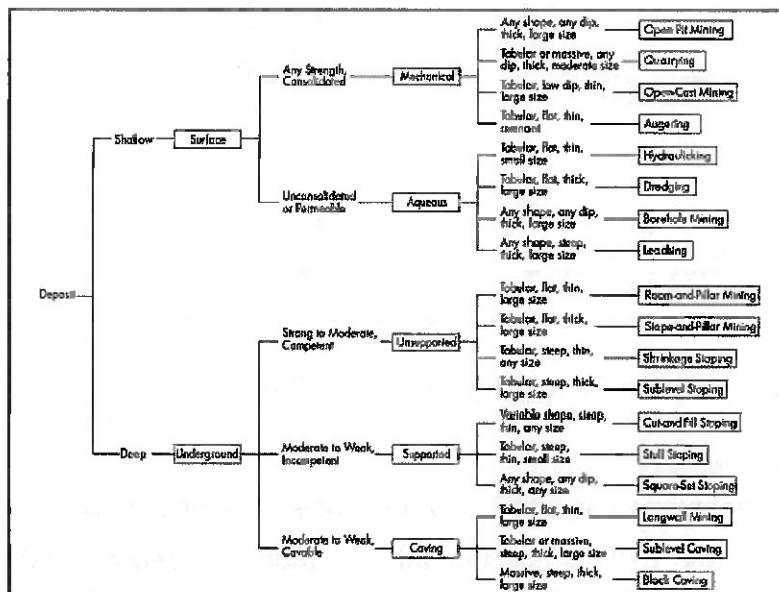


Figure 2-14: Hartman flow chart (Nicholas, 1981).

2.5.4 Nicholas Method

Nicholas (1981) technique is a quantitative MMS using numerical ranking tables to choose the most suitable method. The tables contain several factors which are

considered in MMS; rock mass of orebody; ore geometry and grade distribution (Carter, 2011).

Table 2.14: Nicholas criteria for MMS (Nicholas, 1981)

GENERAL SHAPE		ROCK SUBSTANCE STRENGTH (Uniaxial strength/overburden pressure)		
Equi-dimensional (E)	All dimensions are of the same order of magnitude	Weak (W)	<8	
Platy tabular (T/P)	Two dimensions are many times the thickness which does not exceed 100m	Moderate (M)	8 to 15	
Irregular (I)	Dimensions vary over short distances	Strong (S)	>15	
ORE THICKNESS		FRACTURE FREQUENCY	No. of fractures per m	%RQD
Narrow (N)	<10m	Very close (VC)	>16	0 to 20
Intermediate (I)	10 to 30m	Close (C)	10 to 16	20 to 40
Thick (T)	30 to 100m	Wide (W)	3 to 10	40 to 70
Very thick (VT)	>100m	Very wide (VW)	<3	70 to 100
PLUNGE		FRACTURE SHEAR STRENGTH		
Flat (F)	<20	Weak (W)	Clean joint with smooth surfaces or fill with material which has strength than the rock substance strength	
Intermediate (I)	20 to 55	Moderate (M)	Clean joint with rough surface	
Steep (S)	>55	Strong (S)	Joint filled with material that is equal to or stronger than the rock substance strength	
DEPTH BELOW SURFACE	Provide actual depth			
GRADE DISTRIBUTION				
Uniform (U)	Grade at any point in deposit does not vary significantly from mean grade of that deposit			
Gradational (G)	Grade values have zonal characteristics and the grades change gradually from one to another			
Erratic (E)	Grade values change radically over short distances and do not exhibit any discernible pattern in their change			

Nicholas (1981) Parameters in Table 2.14 were used on ten mining methods listed in Table 2.15.

Table 2.15: Rating for mining method per given parameter (Nicholas, 1981).

MINING METHOD	GENERAL SHAPE				ORE THICKNESS				ORE PLUNGE				GRADE DISTRIBUTION		
	M	T/P	I	N	I	T	VT	F	I	S	U	G	E		
Open pit	3	4	3	2	3	4	4	3	3	4	3	3	3		
Block caving	4	2	0	-49	0	2	4	3	2	4	4	2	0		
Sublevel stoping	2	2	1	1	2	4	3	2	1	4	3	3	1		
Sublevel caving	3	4	1	-49	0	4	4	1	1	4	4	2	0		
Longwall	-49	4	-49	4	0	-49	-49	4	0	-49	4	2	0		
Room and pillar	0	4	2	4	2	-49	-49	4	1	0	3	3	3		
Shrinkage stoping	2	2	1	1	2	4	3	2	1	4	3	2	1		
Cut and fill	0	4	2	4	4	0	0	0	3	4	3	3	3		
Top slicing	3	3	0	-49	0	3	4	4	1	2	4	2	0		
Square set	0	2	4	4	4	1	1	2	3	3	3	3	3		
MINING METHOD	ROCK SUBSTANCE STRENGTH			FRACTURE SPACING				FRACTURE STRENGTH							
	W	M	S	VC	C	W	VW	W	M	S					
ORE ZONE															
Open pit	3	4	4	2	3	4	4	2	3	4					
Block caving	4	1	1	4	4	3	0	4	3	0					
Sublevel stoping	-49	3	4	0	0	1	4	0	2	4					
Sublevel caving	0	3	3	0	2	4	4	0	2	2					
Longwall	4	1	0	4	4	0	0	4	3	0					
Room and pillar	0	3	4	0	1	2	4	0	2	4					
Shrinkage stoping	1	3	4	0	1	3	4	0	2	4					
Cut and fill	3	2	2	3	3	2	2	3	3	2					
Top slicing	2	3	3	1	1	2	4	1	2	4					
Square set	4	1	1	4	4	2	1	4	3	2					
HANGING WALL															
Open pit	3	4	4	2	3	4	4	2	3	4					
Block caving	4	2	1	3	4	3	0	4	2	0					
Sublevel stoping	-49	3	4	-49	0	1	4	0	2	4					
Sublevel caving	3	2	1	3	4	3	1	4	2	0					
Longwall	4	2	0	4	4	3	0	4	2	0					
Room and pillar	0	3	4	0	1	2	4	0	2	4					
Shrinkage stoping	4	2	1	4	4	3	0	4	2	0					
Cut and fill	3	2	2	3	3	2	2	4	3	2					
Top slicing	4	2	1	3	3	3	0	4	2	0					
Square set	3	2	2	3	3	2	2	4	3	2					
FOOTWALL															
Open pit	3	4	4	2	3	4	4	2	3	4					
Block caving	2	3	2	1	3	3	3	1	3	3					
Sublevel stoping	0	2	4	0	0	2	4	0	1	4					
Sublevel caving	0	2	4	0	1	3	4	0	2	4					
Longwall	2	3	3	1	2	4	3	1	3	3					
Room and pillar	0	2	4	0	1	3	3	0	3	3					
Shrinkage stoping	2	3	3	2	3	3	2	2	2	3					
Cut and fill	4	2	2	4	4	2	2	4	4	2					
Top slicing	2	3	3	1	3	3	3	1	2	3					
Square set	4	2	2	4	4	2	2	4	4	2					

2.5.5 Boshkov and Wright Method

Boshkov & Wright (1973) approach was a qualitative technique which is specifically designed for underground mines, see Table 2.16. Factors considered are, ore dip, ore strength, and ore thickness. Carter (2011) stated that the method principle works on the basis that the open pit mining is already eliminated.

Table 2.16: Boshkov and Wright criteria for MMS (Boshkov & Wright, 1973).

Type of ore body	Dip	Strength of ore	Strength of walls	Commonly applied mining methods
Thin beds	Flat	Strong	Strong	Open stopes with casual pillars Room and pillar Longwall
		Weak or strong	Weak	Longwall
Thick beds	Flat	Strong	Strong	Open stopes with casual pillars Room and pillar
		Weak or strong	Weak	Top slicing Sublevel caving
		Weak or strong	Strong	Underground glory hole
Very narrow veins	Steep	Strong or weak	Strong or weak	Resining
Narrow veins (widths up to economic length of stull)	Flat	N/A	N/A	Same as for thin beds
	Steep	Strong	Strong	Open stopes
		Weak	Weak	Shrinkage stopes Cut and fill stopes Cut and fill stopes Square set stopes
		Weak	Strong	Open underhand stopes Square set stopes Top slicing Square set stopes
Wide veins	Flat	N/A	N/A	Same as for Thick beds or Masses
	Steep	Strong	Strong	Open underhand stopes Underhand glory hole Shrinkage stopes Sublevel stoping Cut and fill stopes Combined methods Cut and fill stopes Top slicing Sublevel caving Square set stopes Combined methods Open underhand stopes Top slicing Sublevel caving Block caving Square set stopes Combined methods
			Weak	Top slicing Sublevel caving Square set stopes Combined methods Top slicing Sublevel caving Square set stopes Combined methods
			Strong	Open underhand stopes Top slicing Sublevel caving Block caving Square set stopes Combined methods Top slicing Sublevel caving Square set stopes Combined methods
		Weak	Weak	Top slicing Sublevel caving Square set stopes Combined methods
			Strong	Open underhand stopes Top slicing Sublevel caving Block caving Square set stopes Combined methods
Masses	N/A	Strong	Strong	Underground glory hole Shrinkage stopes Sublevel stoping Cut and fill Combined methods
	N/A	Weak	Weak or strong	Top slicing Sublevel caving Block caving Square set stopes Combined methods
Very thick beds	N/A	N/A	N/A	Same as for Masses

2.5.6 University of British Columbia (UBC) Method

UBC is the modification of the Nicholas (1981) approach which places more emphasis on the stoping mining method (Bitarafan & Ataei, 2004). The technique considers rock mass rating (RMR) and not the rock quality density (RQD) strength (Bitarafan & Ataei, 2004). It uses the ranking method between 1 and 5 with 5 being most appropriate. The ranking score of -49 represent a mining method which is inappropriate. Table 2.17 illustrates UBC method for selection of the MMS.

Table 2.17: UBC mining method selection criteria (Edumine, 1999).

General shape	
Equi-dimensional	All dimension are of the same order
Platy-tabular	Two dimensions are larger than the thickness
Irregular	Dimension vary
Orebody thickness	
Narrow	<10 m
Intermediate	10 – 30 m
Thick	30 – 100 m
Very thick	>100 m
Deposit plunge	
Flat	<20°
Intermediate	20° – 55°
Steep	> 55°
Grade distribution	
Uniform	Grade is equal to the mean grade throughout the orebody
Gradational	Grade has a zonal characteristic which gradually changes from zone to zone
Erratic	Grade can change quickly over distance without any pattern
Deposit depth	
Shallow	<100 m
Intermediate	100 – 600 m
Deep	>600 m
RMR	
Very weak	0 – 20
Weak	20 – 40
Medium	40 – 60
Strong	60 – 80
Very strong	80 – 100
RSS	
Very weak	<5
Weak	5 – 10
Medium	10 – 15
Strong	>15

2.6 Mining Method Assessment

At a conceptual phase of the mining project, it is not easy to choose a mining method but the technical flaws are easily eliminated (Siversten, et al., 2010). The driving criterion for the mining method assessment is to deliver the technical feasible option on selecting of the mining method.

There are Three General Principles for Underground Massive Mine:

2.6.1 Open Stope Methods

Excavations are un-caved and unfilled. The excavations must remain stable for the life of mine and the rock is used to an advantage, and the method is typically applied in hard rock requiring little or no support.

Mining methods: Open benching, open stoping with permanent pillars, and room and pillar mining methods.

2.6.2 Supported Methods

These methods typically use backfill, either cemented or un-cemented. The methods are typically applied to high value orebodies where resource loss to pillars is financially undesirable.

Mining methods: Stope and fill, post pillar cut and fill (used in wide orebodies), bench and fill, cut and fill, and drift and fill mining methods (the differences are in the size of the excavation prior to filling, the more competent the rock, the larger the excavation).

2.6.3 Caving Methods

Kimberlite pipes are suited to caving methods as they tend to be massive, vertical and circular in cross section. The key lies in having enough footprint to ensure that the hydraulic radius can be achieved thus delivering continuous caving. Historically, fragmentation has not been a problem with kimberlite.

- **Natural Caving:** The method relies on the rock mass to cave under gravity.
Mining methods: Block caving, incline caving, and front caving mining methods.
- **Induced Caving:** The method is used for competent orebody, Mining methods: Sub level caving, blast or hydro-fractured assisted caving, and open stope with mass blasting of pillars.

The known mining methods typically have fixed application criteria based on the quality of the rock and resource value. Based on the industry applications, it is usually easy to narrow down the applicable mining methods for an ore deposit. Though any orebody can be mined with any mining method, the challenge however is in recovery of the ore safely, economical and predictable. Thus, for caving mining principles, the rock properties determine which mining method is applicable.

2.7 Mining Method Description

Below is a brief overview of each mining method.

2.7.1 Open or Natural Support Methods

2.7.1.1 Open Benching

The method practised in kimberlite orebodies and use open benching method also known as blast hole open stoping Figure 2.15. Parallel horizontal drifts are blasted into the orebody and retreated to the contact using long hole. It's a daylight mining and loaders are remote controlled driven into the open void.

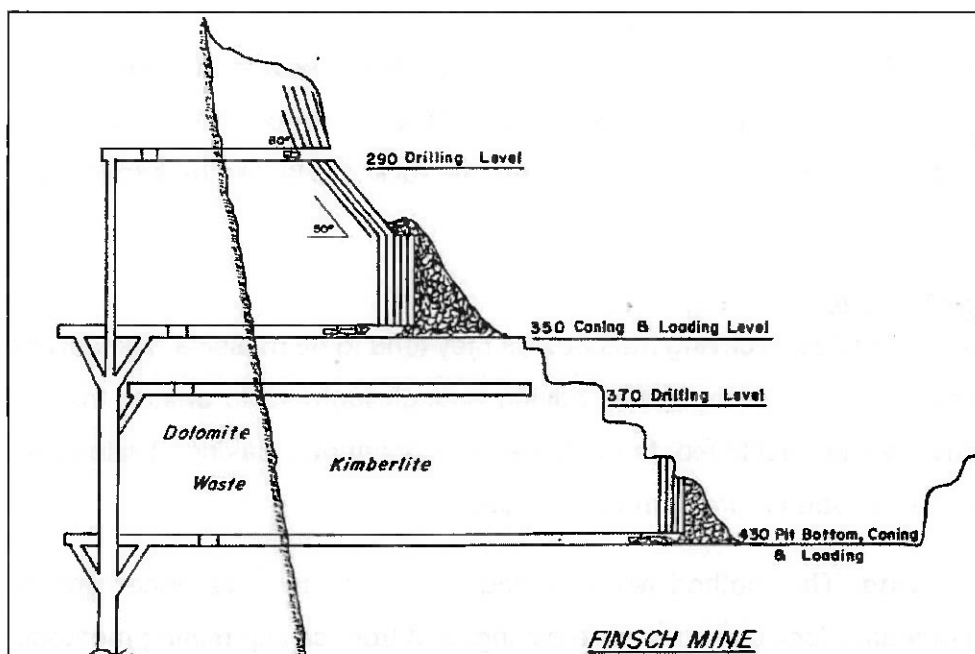


Figure 2-15: Blast Hole Open Stopping as practised at Finsch Mine (Preece, 2001).

Method application:

- It requires a stable country rock – permits for widely spaced drifts and large ring blasting thus high production & low cost with good recovery.
- Regional stability provided by a stable pit and exposed kimberlite faces. It is a non-entry method – use remote controlled loaders and secondary breakers. Improved safety is an advantage.

2.7.1.2 Open Stopping With Pillars

The method involves creating large underground excavations and leaving pillars behind to keep these excavations stable. It is typically done by developing parallel ring drilling drifts through the orebody and developing a loading level beneath them, see Figure 2.16. The stopes are self-supported. It is a non-entry and use of remote control LHD's and secondary breaking rigs. Orebody pillars are left in situ to provide regional stability.

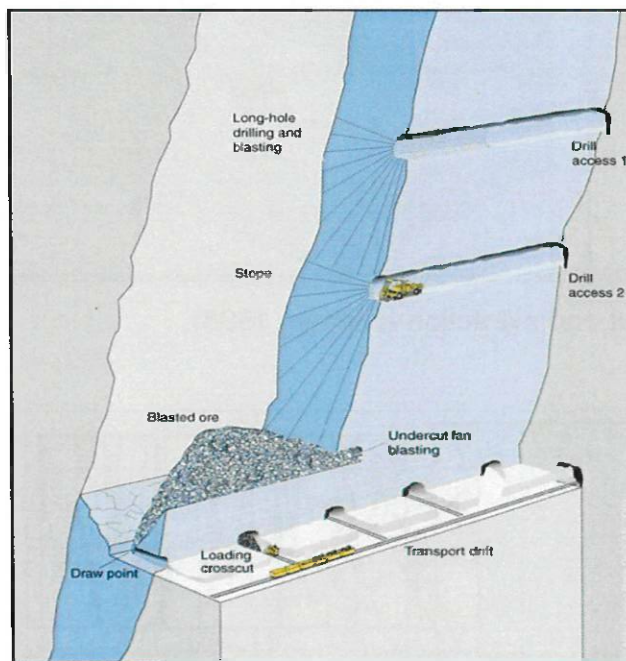


Figure 2-16: Typical Open Stopping layout and extraction (Hamrin, 1998)

2.7.2 Artificial Support Method

2.7.2.1 Open Stopping With Fill

Ore body subdivided on each sublevel into primary & secondary stopes. Primary stopes mined first and filled with consolidated fill (cemented). Secondary stopes mined once neighbouring primary stopes have cured then filled with unconsolidated fill. The method is also a pillarless, non-entry and use of remote controlled LHD's, see Figures 2.17, 2.18 and 2.19 below.

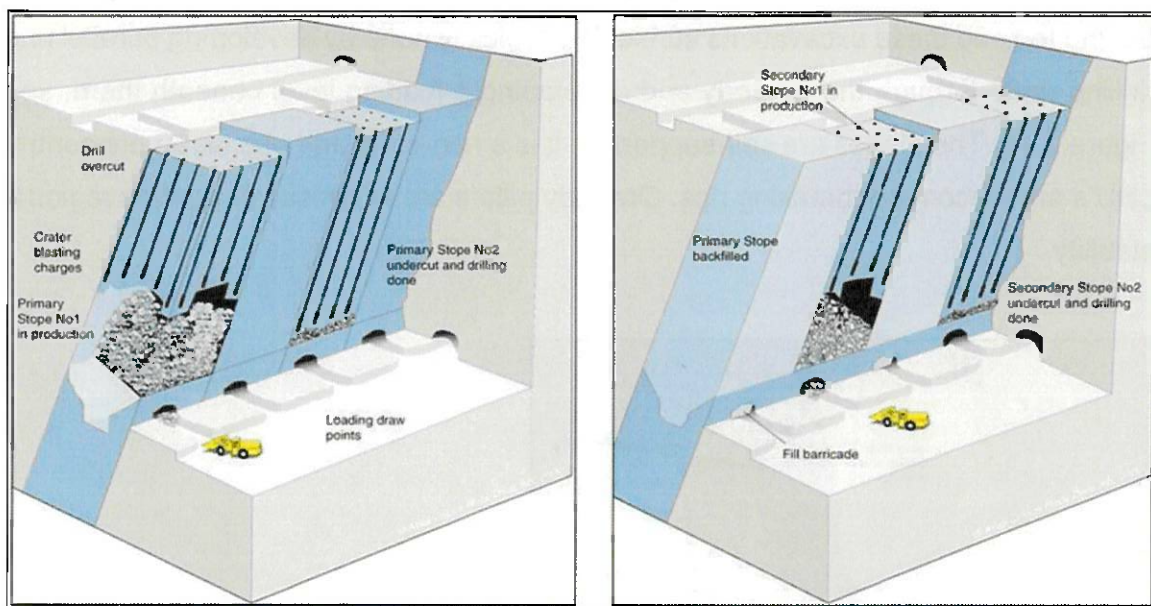


Figure 2-17: Typical Stope and Fill layout and extraction (Hamrin, 1998).

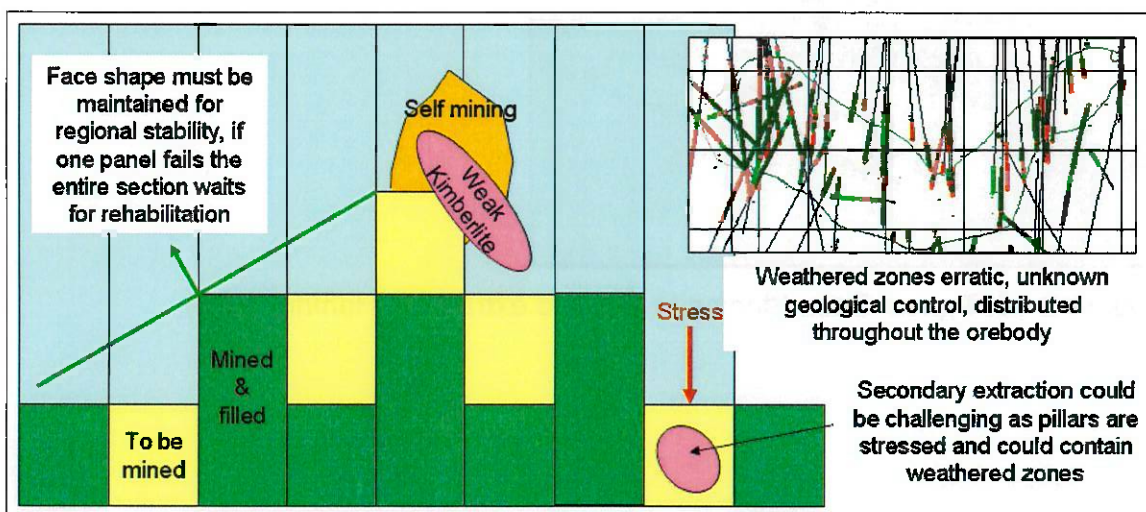


Figure 2-18: Challenges using the method in weak rock (Herselman et al., 2009).

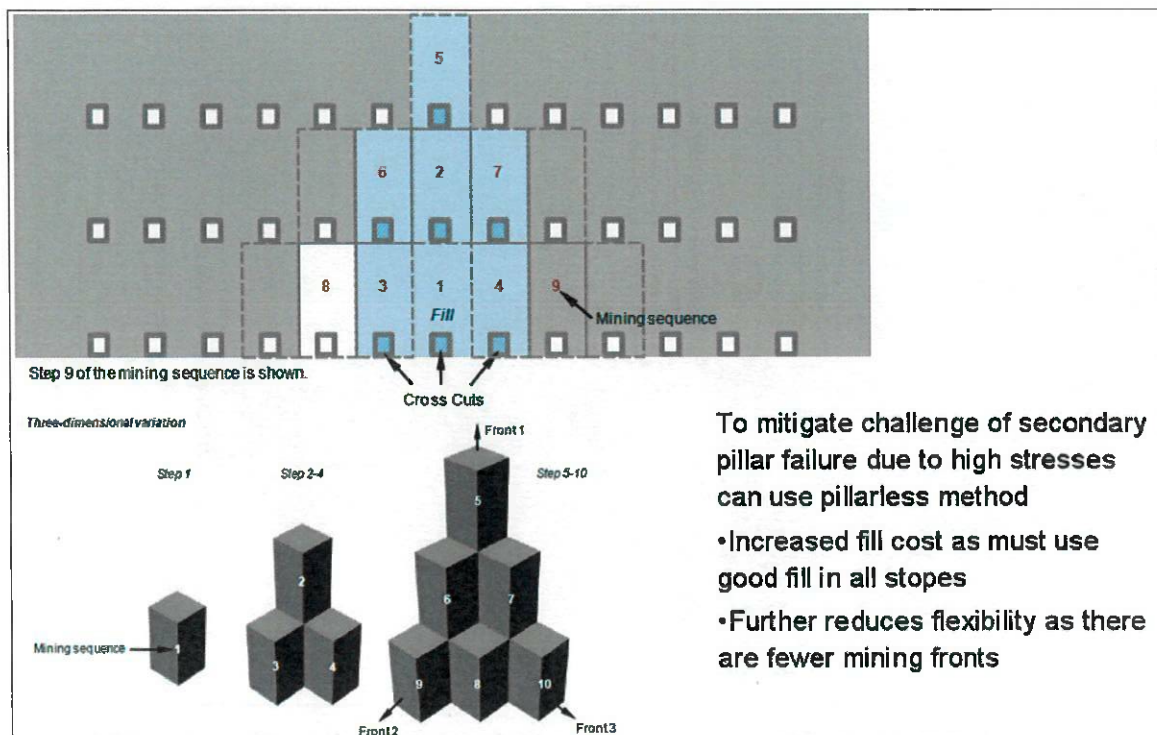


Figure 2-19: Pillarless Extraction layout and sequence (Herselman et al., 2009).

2.7.2.2 Post Pillar Cut and Fill

The method involves extraction of the orebody in horizontal slices leaving yielding stacked pillars behind and then filling the excavation for stability. It yields good recovery of the orebody within low production rate and cost as compared to traditional cut and fill (Figure 2.20). The significance of this method is that it separates the extraction from the filling cycle and the fill used can be of low strength as it is the pillars that do the work. It's typically applied in wide orebodies and it's a full entry method.

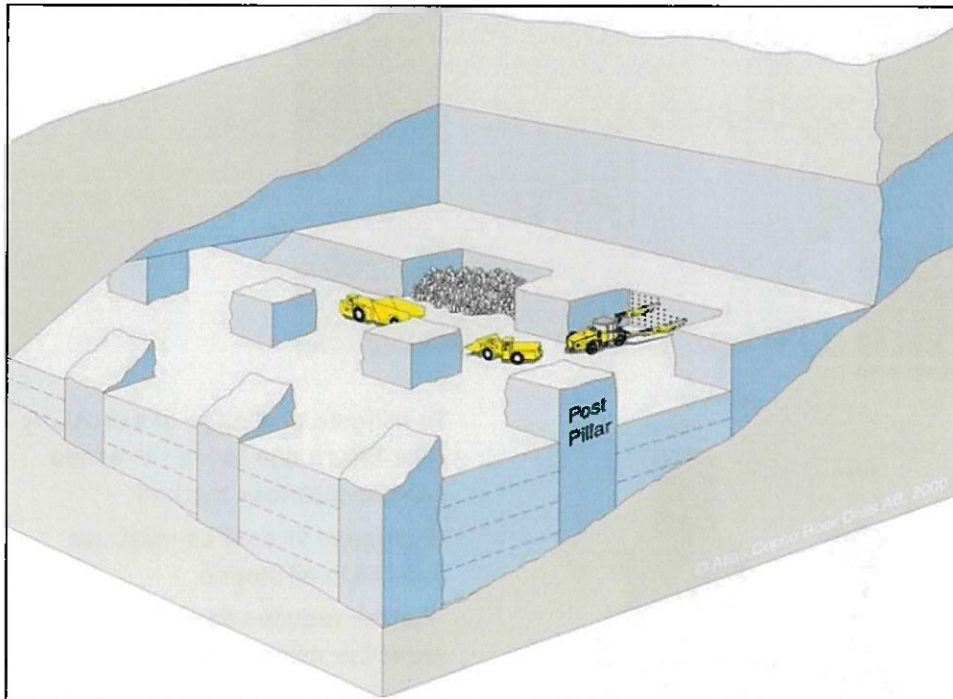


Figure 2-20: Typical Post Pillar Cut and Fill layout and extraction (Hamrin, 1998)

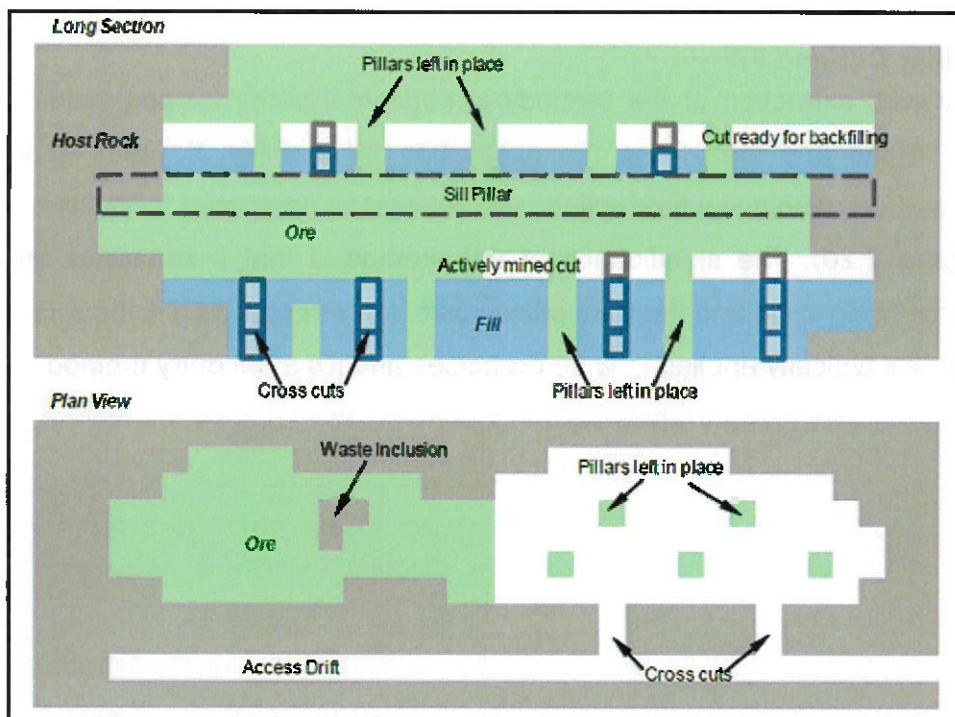


Figure 2-21: Typical Post Pillar Cut and Fill layout and extraction (Makhari, 2019).

Figure 2.21 illustrate the schematic view of the overhand post and pillar cut and fill method. Mining progresses upwards, on two mining horizons with alternating mining and filling cycles. Objective of the process is always to have a free face.

2.7.2.3 Cut and Fill

The method involves extraction of the orebody in horizontal slices and then filling them to keep the excavations stable, Figure 2.22. The method delivers a good recovery of the orebody at a low production rate and high cost.

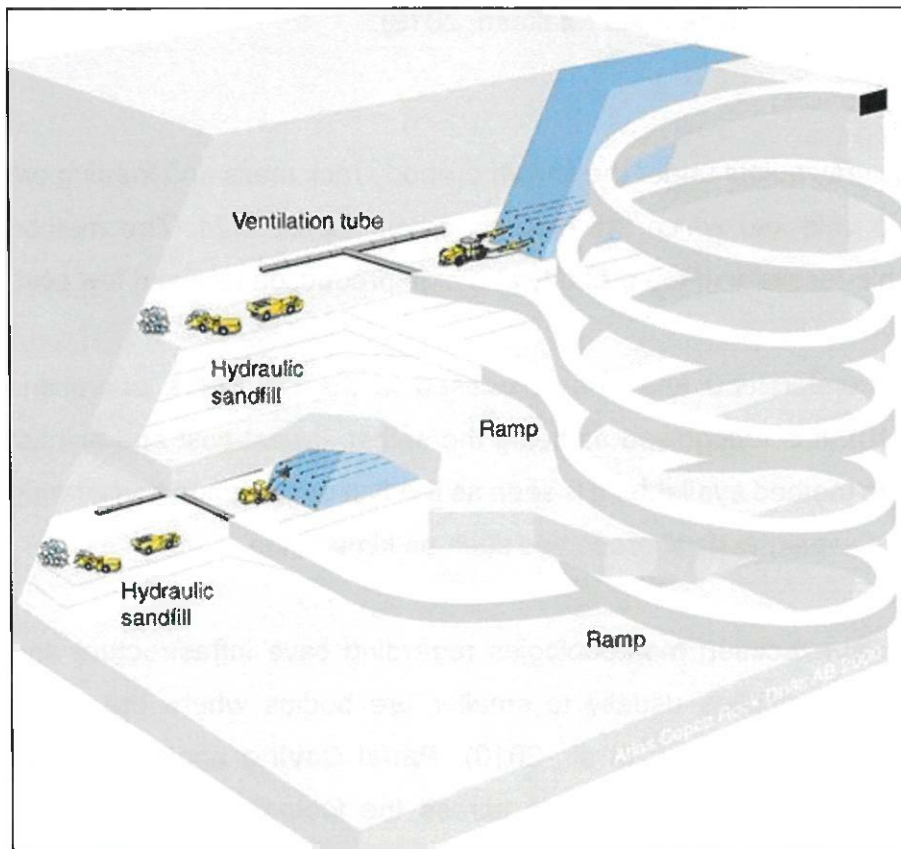


Figure 2-22: Typical Cut & Fill type mine (Hamrin, 1998).

2.7.2.4 Drift and Fill

The method permits the extraction of the orebody as shown in Figure 2.23 with horizontal drifting and then filling the drifts to keep the excavations stable. The method delivers a good recovery of the orebody at a very low production rate and very high cost.

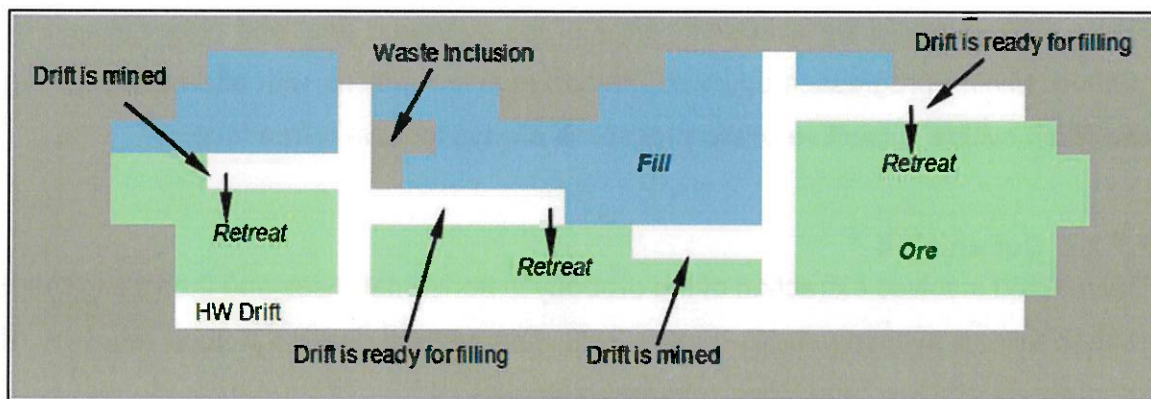


Figure 2-23: Typical Drift & Fill type mine (Makhari, 2019).

2.7.2.5 Block Caving

Block caving creates drawpoints under the shown orebody rock mass and loading ore from them creates a void into which the orebody caves Figure 2.24. The method results in a reasonable recovery of the orebody at a high production rate and low cost.

Block caving is well understood and often practised in the De Beers operations (Siversten et al., 2010). It is recognised as being the safest, lowest cost and highest production rate mining method available. It is seen as the first option when considering the mining of massive, weak, vertical orebodies such as kimberlites.

There are four general execution methodologies regarding cave infrastructure and depletion. **Block Caving** applies usually to smaller ore bodies where the whole footprint is caved at once (Siversten et al., 2010). **Panel Caving** applies to large footprints with a caving face that progresses across the footprint of the orebody (Siversten et al., 2010). **Inclined Caving** is applied where there is a significant dip or plunge in the orebody and the production level can be fitted to the plunging geometry with a "Sub-Level Caving" layout (Siversten et al., 2010). **Front Caving** is applied where the rock mass is too weak to maintain the production development over a large area, or in steeply dipping ore bodies (Siversten et al., 2010). In both cases, a single line of draw points is developed, caved, and exhausted before the next line is started.

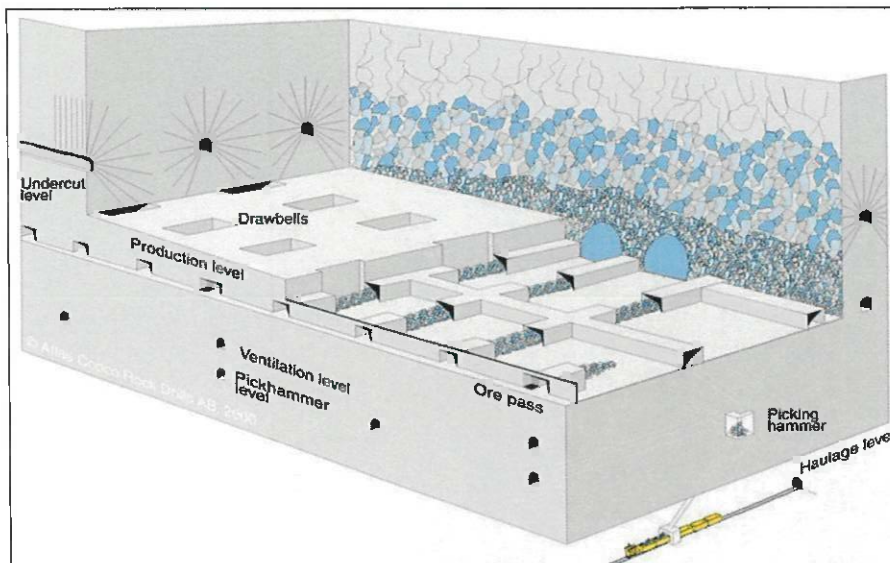


Figure 2-24: Block Cave layout showing undercut and production tunnels (Hamrin, 1998)

2.7.2.6 Sub Level Caving

Sub level caving is applied to competent orebodies which are narrow and inclined with a waste rock hanging wall that will cave. Typically, parallel drifts that are close together are developed along the short axis of the orebody and then long hole drilled and blasted on retreat to the rim tunnel as shown in Figure 2.25. The production rate is high due to the relatively simple mining cycle and large tonnage released per blast.

The challenge with the method lies in dilution management and resource recovery. Tunnel spacing, extraction sequencing and draw control is vital to ensure an acceptable recovery. The method is also associated with mud rush risks when the muck pile contains clay rich material and water ingress (rainfall or groundwater) occurs.

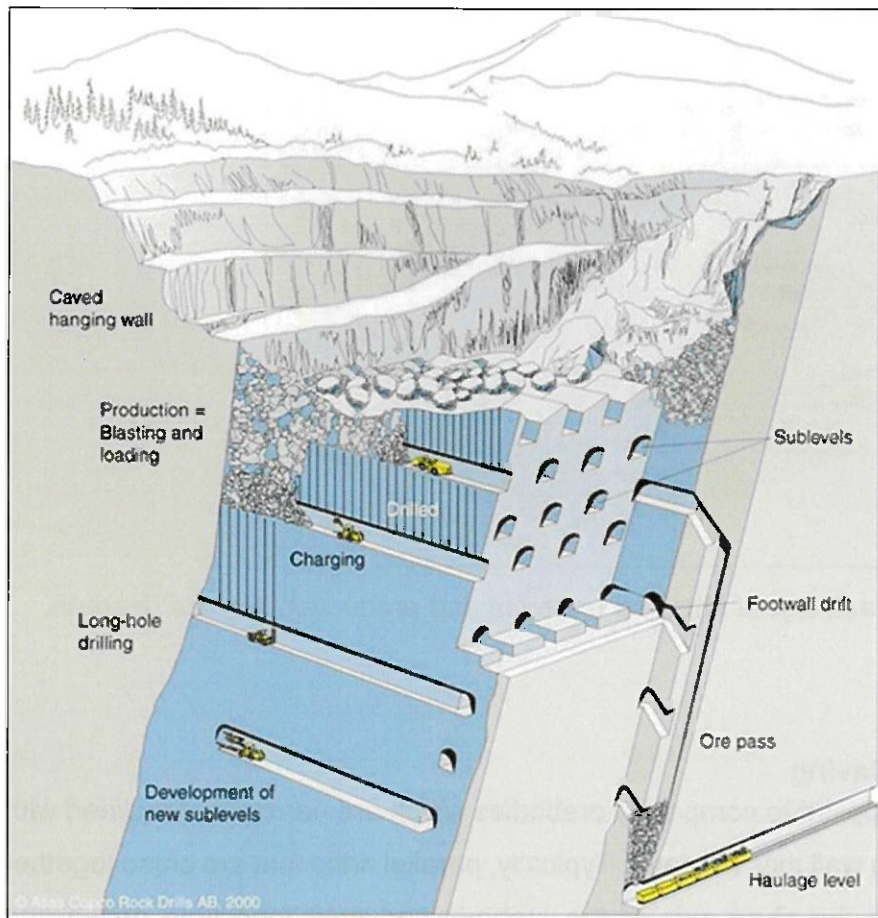


Figure 2-25: Sub Level Caving (Hamrin, 1998).

2.8 Previous Studies on Caving Mining Method

2.8.1 Ekati Diamond Mine

Ekati Diamond Mine has adopted a sub level cave mining method for their Koala pipe. The orebody was 200m long by 100m wide but tapers-off with depth. The RMR is 60 ± 11.5 for the SLC but the 160m of cave material above the SLC was considerably weaker see Figure 2.26. The small orebody footprint and severe taper with depth eliminated the possibility of a block cave type operation as the caving radius could not be achieved for the main high value ore. Open pit mining was stopped 250m below surface on the low-grade cave material contact.

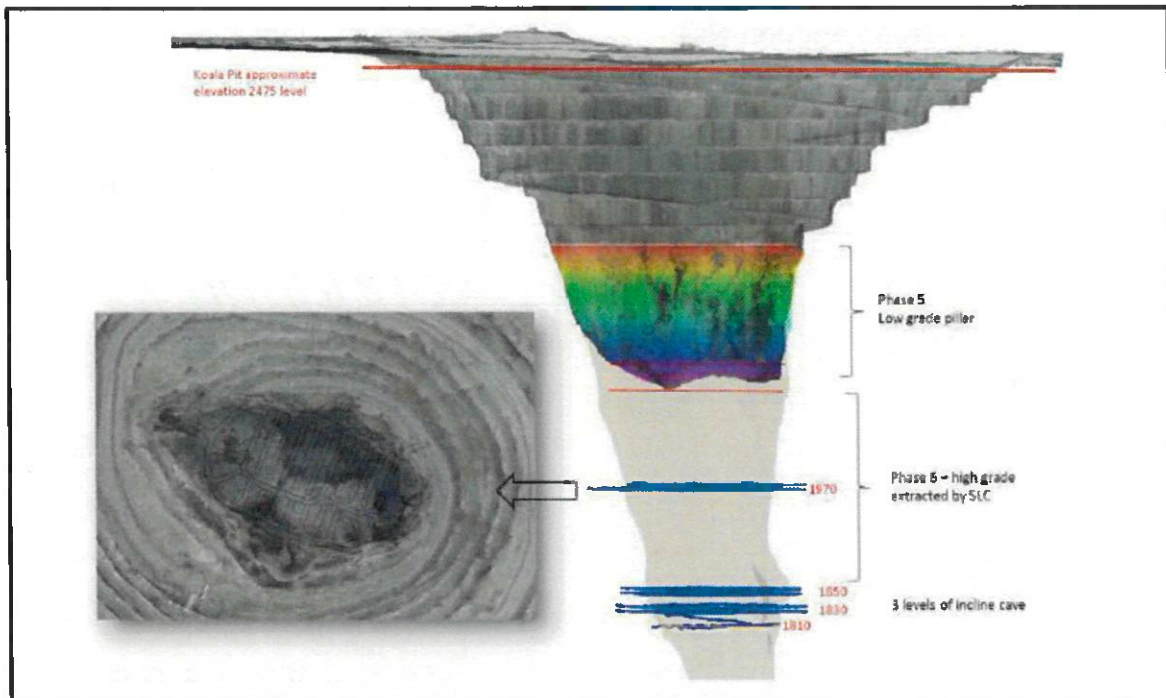


Figure 2-26: Three-dimensional isometric view of Koala pipe (Jakubec et al., 2018).

The sub level cave design consisted of 5m x 5m tunnels on 14.5m horizontal and 20m vertical spacing as shown in Figure 2.27. A total of 10 levels were planned with every fifth level being converted to a ring pull level where drawpoints are established right around the orebody and loaded to recover the caved material. Five levels have been mined out to date. The production rate was planned at 3,300t/day, or 1.2Mt/year (Jakubec et al., 2018). This was achieved within a two-year ramp-up and subsequently increased to 2Mt a year once the cave had broken through to surface (Jakubec et al., 2018). Draw control consisted of loading material from the drawpoint well more than 120% to recover cave material.

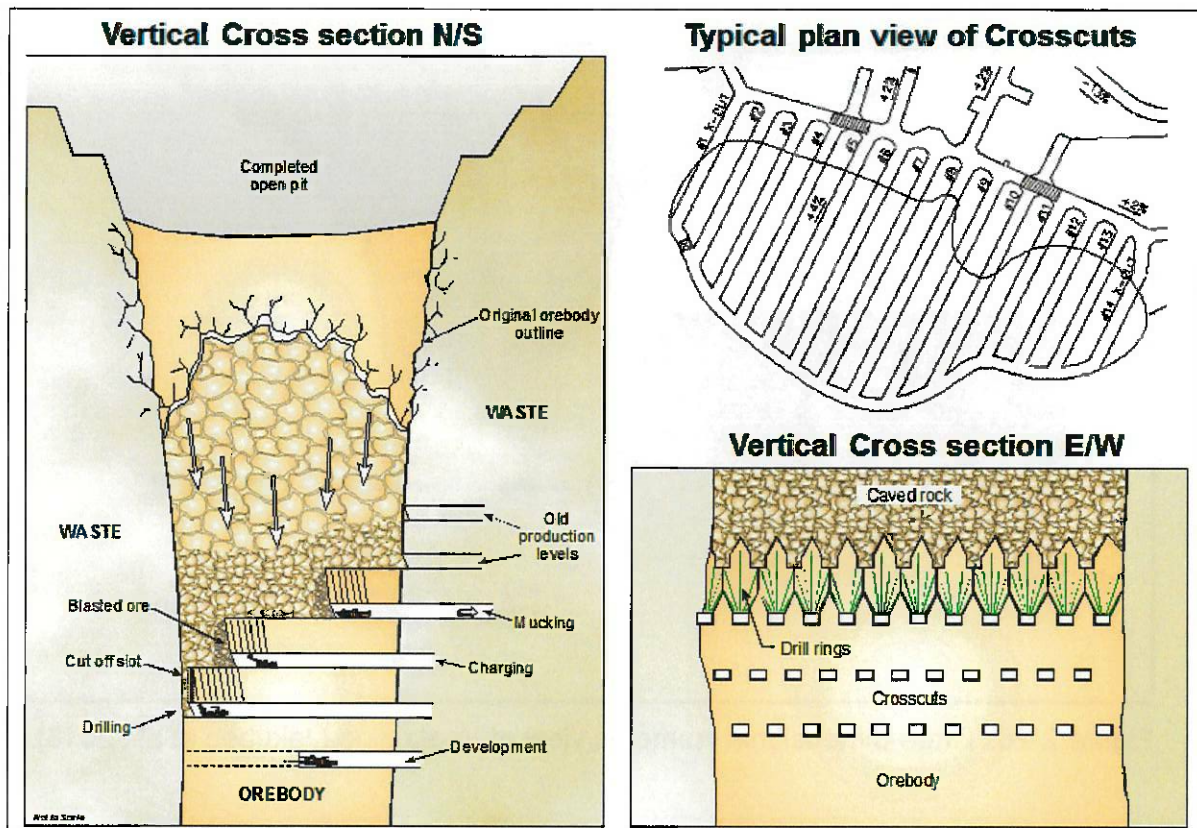


Figure 2-27: Section and plan views of Ekati Diamond mine (Jakubec, et al., 2018).

In Ekati Mine, a 160m low grade pillar was left above the SLC with the intention of drawing the pillar down using an overdraw strategy and ring pull levels (rim loading levels established around the orebody on every fifth level) (Jakubec, et al., 2018).

2.8.2 Ridgeway Mine

Ridgeway SLC closed in late 2009 and is widely recognised as the most successful modern era sub level cave mine (Brunton et al., 2009). The mine produced between 4Mtpa and 5Mtpa, with recoveries of more than 90% based on an overdraw strategy (loading more than 120% of blasted tons). The mine started with a sub level spacing of 25m and increased it to 30m shown in Figure 2.28. Horizontal tunnel spacing was always 14m. Tunnel size varied over time with an average tunnel being 6m wide by 4.7m high. Levels were approximately 200m in width by 200m in length.

Ridgeway SLC was placed in a high value mineralisation with the grade decreasing as the cave progressed to surface. Production was ramped up to full production within two years.

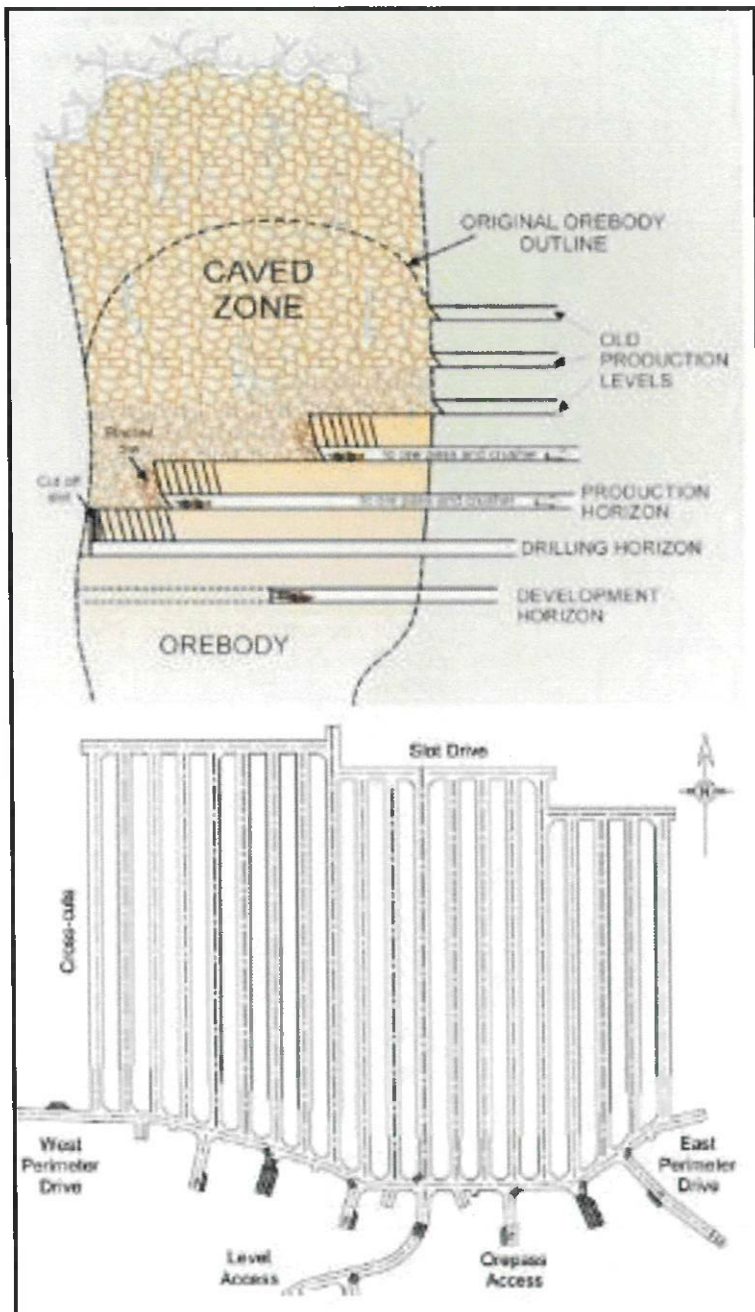


Figure 2-28: Ridgeway SLC mining method (Brunton et al., 2009).

In general, operating conditions involved two levels at full production with the mine requiring 40 drawpoints to maintain steady state production. Draw control was based

on a strategy of 120% (Figure 2.29) overdraw of the blasted tons which was done to ensure recovery of the mineralised zone above the SLC (Brunton et al., 2009).

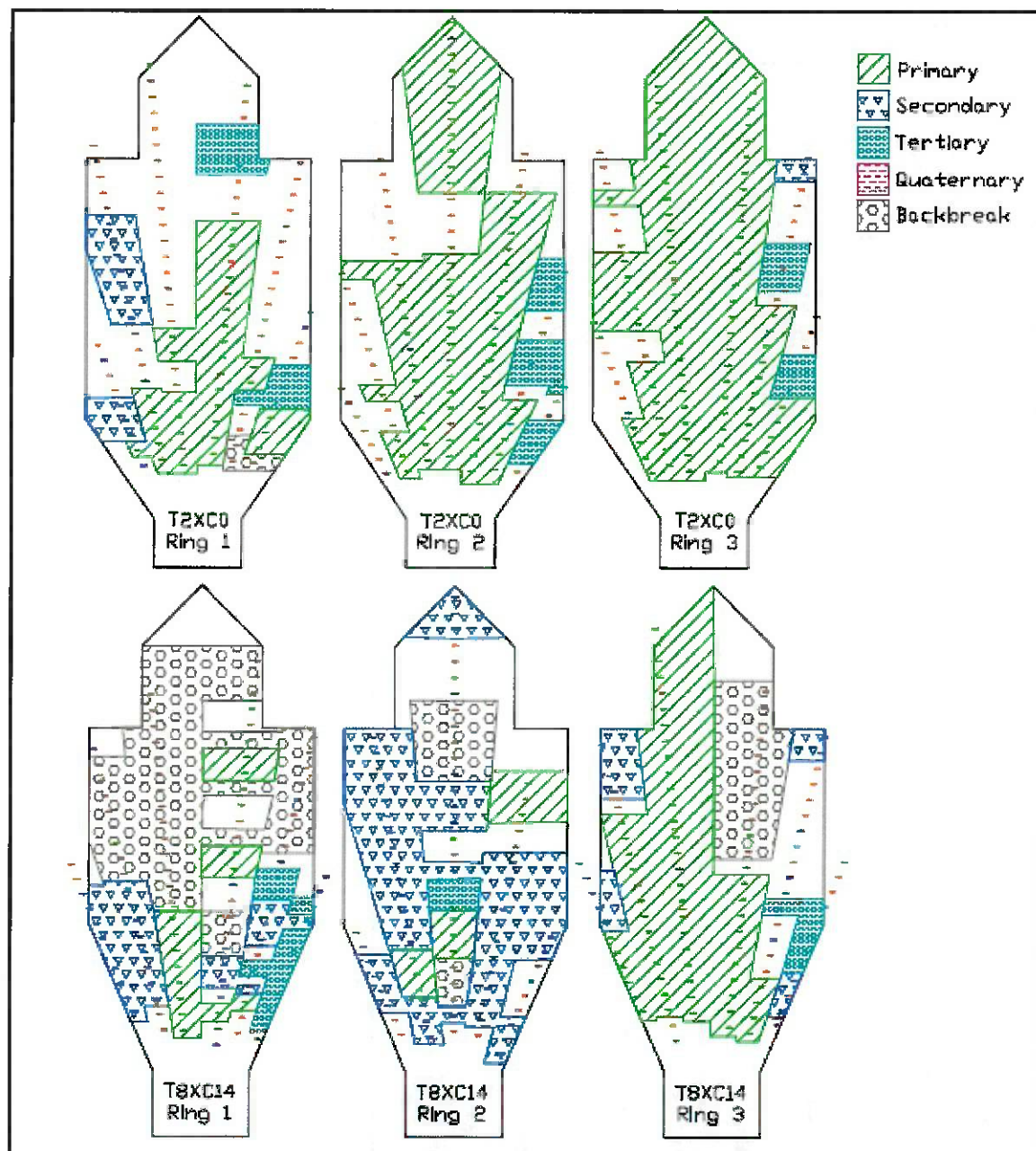


Figure 2-29: Section and plan view of Ridgeway production tunnels (Brunton et al., 2009).

2.9 Chapter Summary

The literature study highlighted the importance of geotechnical consideration in evaluating the selection of an underground mining method. The history of transition from open pit to underground, geology, mineral reserves, geo-consideration, mining methods, mining method options, and economic viability options are discussed as recorded in literature, highlighting the existence of literature gap in optimising the selection of a viable underground mining method selection. The evaluation tool and technique were chosen in the literature review for the implementation in the data analysis. The next chapter formulates the basic methodology adopted on selection of the mining method.

3 RESEARCH METHODOLOGY

3.1 Chapter Overview

This chapter discusses the scope of research methodology used in this project. The chapter starts with brief overview case study for Venetia Mine. This is followed by the research approach and design, methods employed in the study, and data collected and analysed. The chapter also looks at the limitations of the study.

3.2 Venetia Mine Case Study

The Venetia underground project is designed as a green field project executed in a brown field project consisting of two vertical shaft and a decline. The mine has been operating as an open pit since 1992 and is comprised of two orebodies, K01 and K02 kimberlite deposit. Figure 3.1 shows a plan view of the Venetia kimberlite pipes with cut 5 pit shell limits.

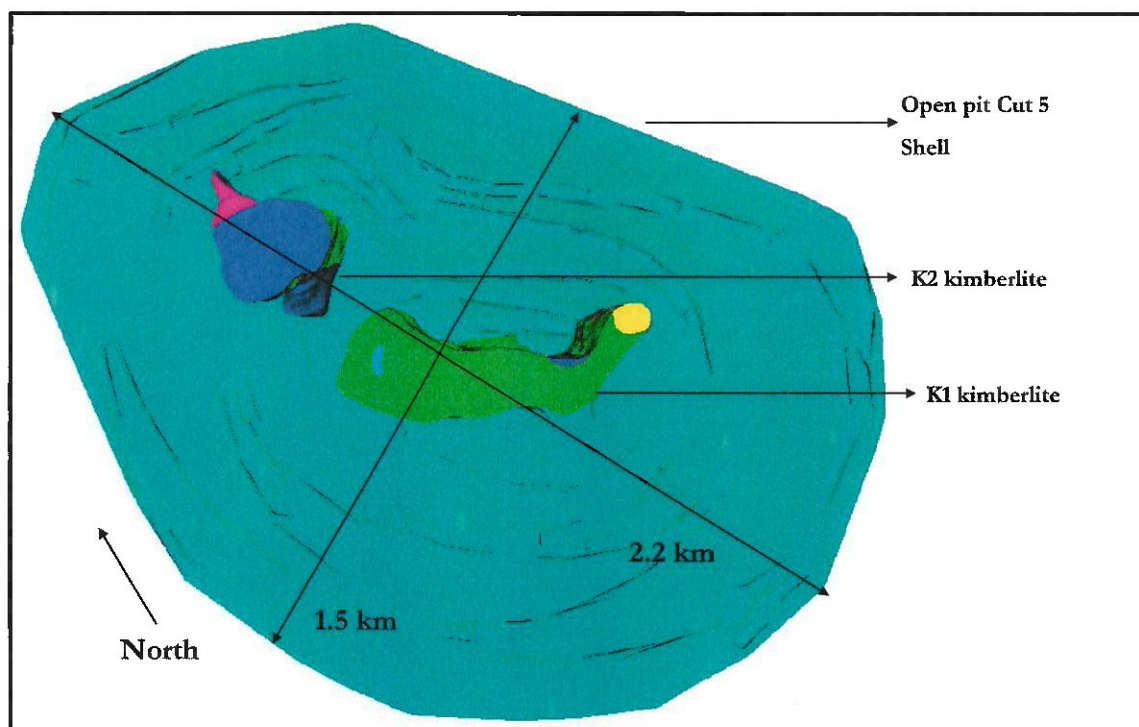


Figure 3-1: Plan of K01 and K02 with Cut 5 Pit Shell (VUP-CSR, 2006).

Over the years, advances in block cave technology have shown that with an understanding of the orebodies, appropriate mine design and management, block

caves can be mined at a lower cost than by conventional drill and blast methods. It is for this reason that DeBeers technical team chose to use block caving mining method to commence Venetia underground K01 orebody (VUP-CSR, 2006). The open pit will mine both kimberlite pipes K01 and K02 to a depth of 600m and 450m respectively as shown in Figure 3.2.

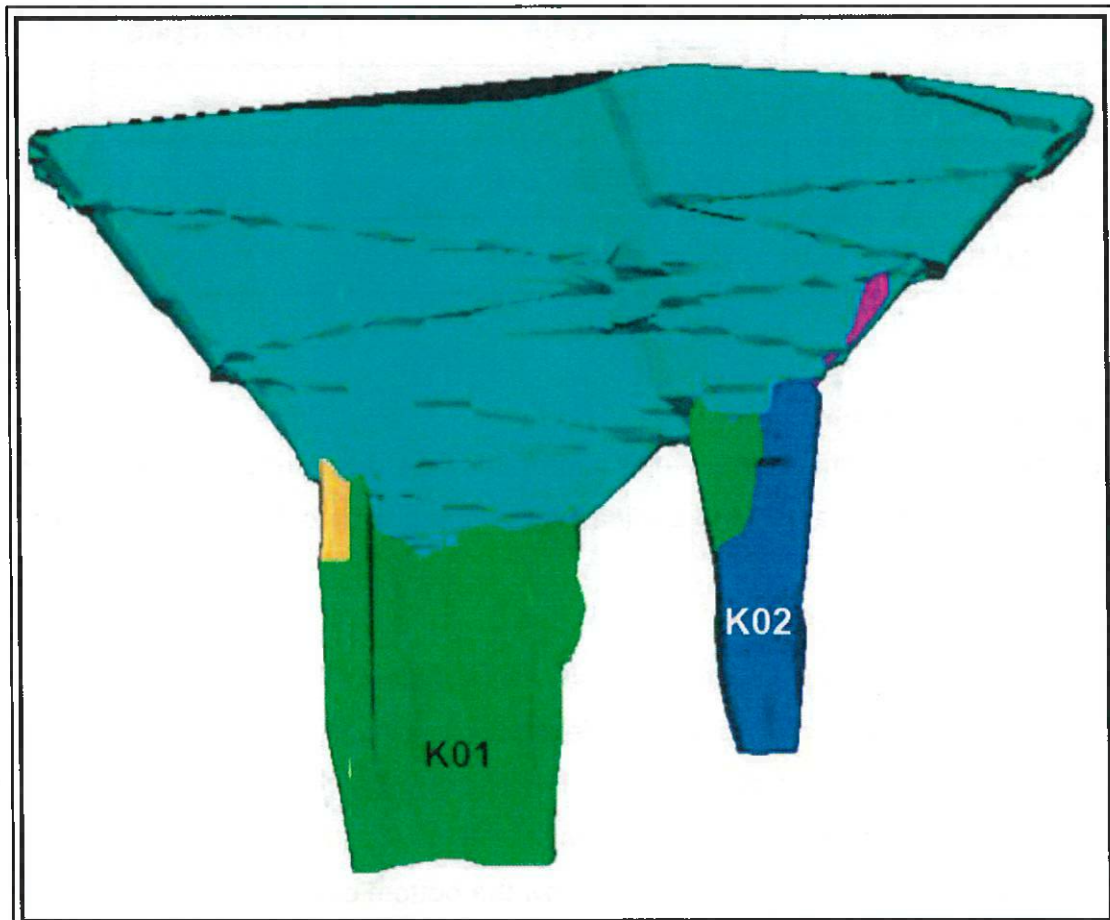


Figure 3-2: Cut 5 shell and underground extrapolated ore resource (VUP-CSR, 2006).

The underground ore resource has been classified as very low inferred. Table 3.1 below illustrate the in-situ tonnage and average grades resources

Table 3.1: In situ tonnage for K01 and K02 underground ore resource

In Situ Tons - Venetia Underground		
Block	Tons	Grade (cpht)
Total K1	51,315,759	140
Total K2	37,057,819	27
Total K1 & K2	88,373,577	

3.3 Block Cave Design

The assumptions were made and largely informed by the recent underground project, i.e. Finsch Mine Block 4 and 5 and Cullinan centenary cut projects. The assumptions were as follow:

- K01 mining method will be;
 - Blast hole open stoping (BHOS) for remainder of K01 open high walls;
 - K01 to be mined up to 600m level below surface;
 - First block cave directly below BHOS to the height of 150m;
 - Second block cave to the height of 150m.
- Ground infrastructure to be placed below the bottom caves.
- Tunnel size and support as per current Finsch standards.

The block cave mining method has been designed according to the parameters, development and scheduling rates based on Finsch block 5. Table 3.2 illustrates design parameters for the concept study.

Table 3.2: Block cave layout parameters (VUP-CSR, 2006).

Block cave layout parameters	
Column Height (both K1 and K2 Kimberlite pipes)	150m
Drawpoint area	225m ²
Layout geometry to ensure that the maximum drawpoint area is not exceeded	
Undercut level horizontal tunnel spacing (centre to centre)	15m x 15m
Extraction level horizontal tunnel spacing (centre to centre)	30m
Vertical tunnel spacing (grade to grade)	15m
Drawpoint spacing along production tunnel	15m
Drawpoint layout system	Herringbone
Drawpoint orientation	45°
Rim tunnel to contact middling	30m
Contact support length at contact position (undercut level)	5m
Contact support length at contact position (production level)	10m
Undercut slotting position from contact	20m
Undercut height	10m

The design parameters have resulted in mining blocks that have short life span and are shown below in figure 3.3

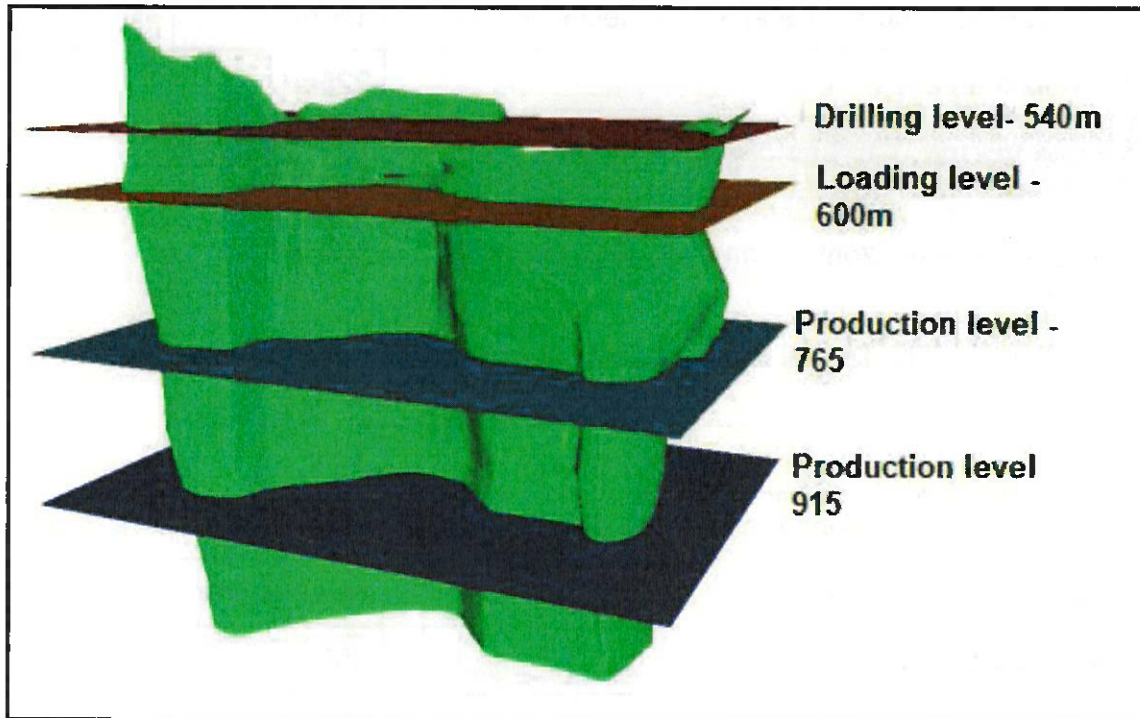


Figure 3-3: K01 Pipe mining blocks (VUP-CSR, 2006).

3.4 Undercut Level for BC

The design for the undercut level is based on the Finsch block 4 undercutting methods. The tunnels are oriented at the short axis of the orebody and it allows a numerous working area. This also give the best distribution of production draw points from which wedge tonnes can be loaded. Figure 3.4 illustrate the undercut layout on K01 orebody.

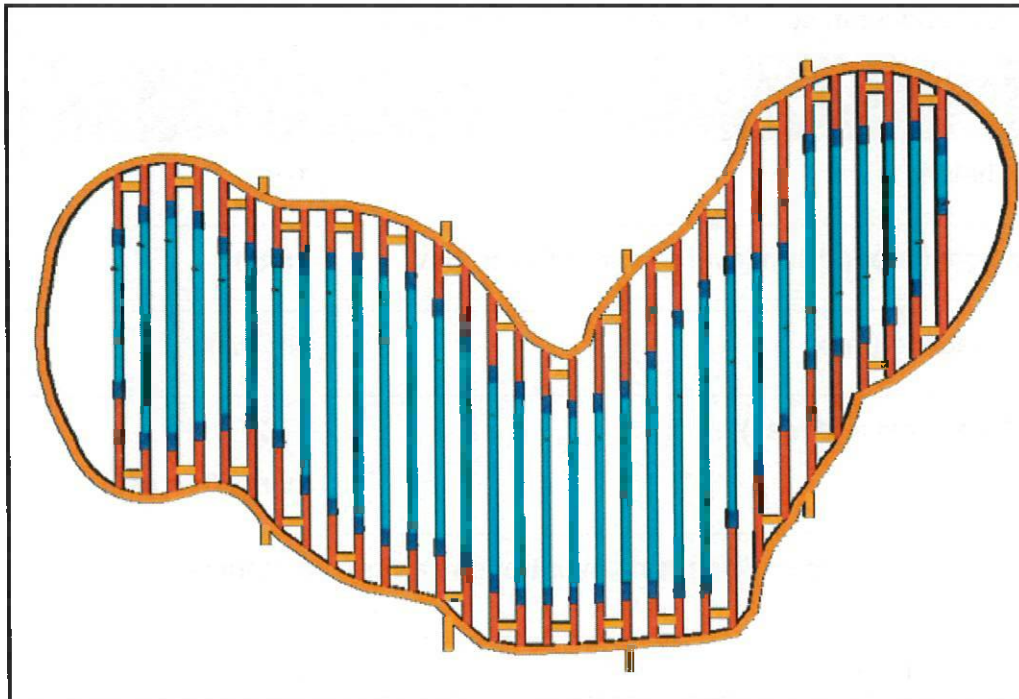


Figure 3-4: Plan view of the undercutting level of the BC (VUP-CSR, 2006).

The undercut will be initiated from north-western part of the orebody using 10m high raisebore (RB) holes. The slot will be cut in 1m diameter RB holes, 24m inside the northern contact from where the undercut rings will be blasted. Table 3.3 illustrate the BC undercutting parameters.

Table 3.3: BC undercutting parameters (VUP-CSR, 2006).

Undercutting parameters	
Undercut height	10m
Trough height (depends on level spacing) FW to HW	15.8m
Undercut lead/lag (max)	5m - 7m
Undercut rate (min and max)	8-12 m/month/tunnel
Re-compaction time (from u/c passing over trough centre)	6 months
Stress shadow angle	45°
Resource recovery factor	85%
Tunnel availability	70%
Initial draw rate	50mm/day
Production draw rate	200mm/day

3.4.1 Extraction Level

The extraction level is based on a single sided loading with ground handling infrastructure on the northern of the orebody. Figure 3.5 and 3.6 illustrate level layout and trough layout respectively.

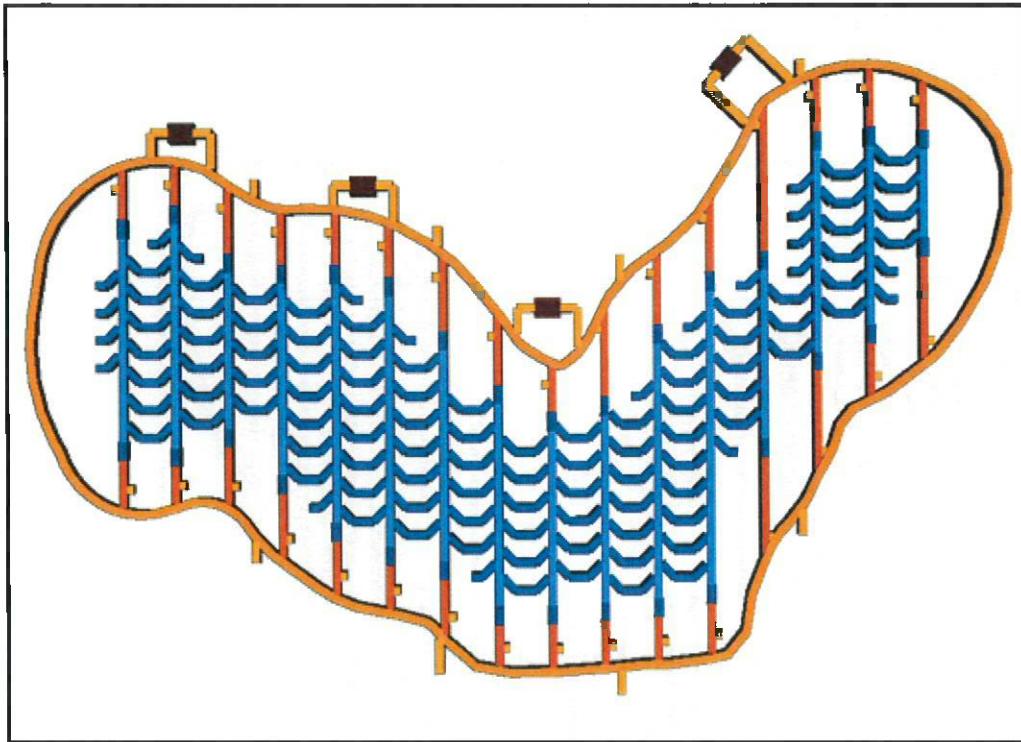


Figure 3-5: Plan view of extraction level (VUP-CSR, 2006).

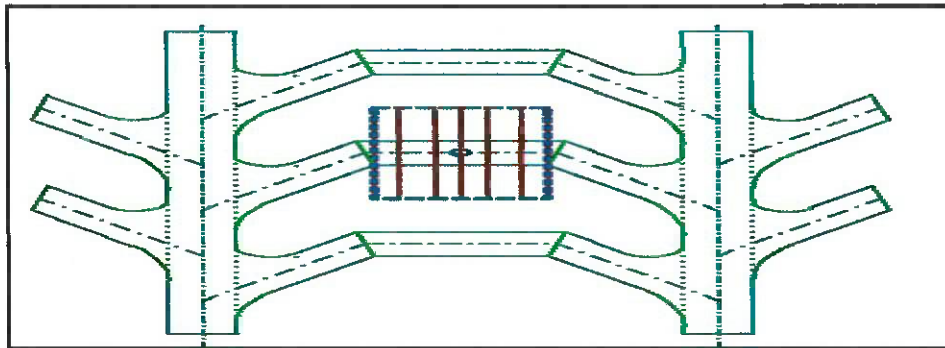


Figure 3-6: Plan view of the trough layout (VUP-CSR, 2006).

3.4.2 Blast Hole Open Stope Level

The BHOS has drilling level at 540m and loading level at 600m (Figure 3.3). This was laid out to take advantage of the extensive open pit perimeter and utilise the free face. A 'V' shape retreat concept was adopted, with the loading tunnels laid along the axis of the ore body with most of the tunnels having the potential to produce as soon the open pit operation has terminated. The loading tunnels were spaced 15m horizontal while drilling tunnels were spaced at 30m so that the two troughs collect the fragmented ground from a full ring blast. Figure 3.7 illustrate the loading level of the

BHOS. A north/south slot was then blasted to create a free face for the balance of the loading tunnel on the level.

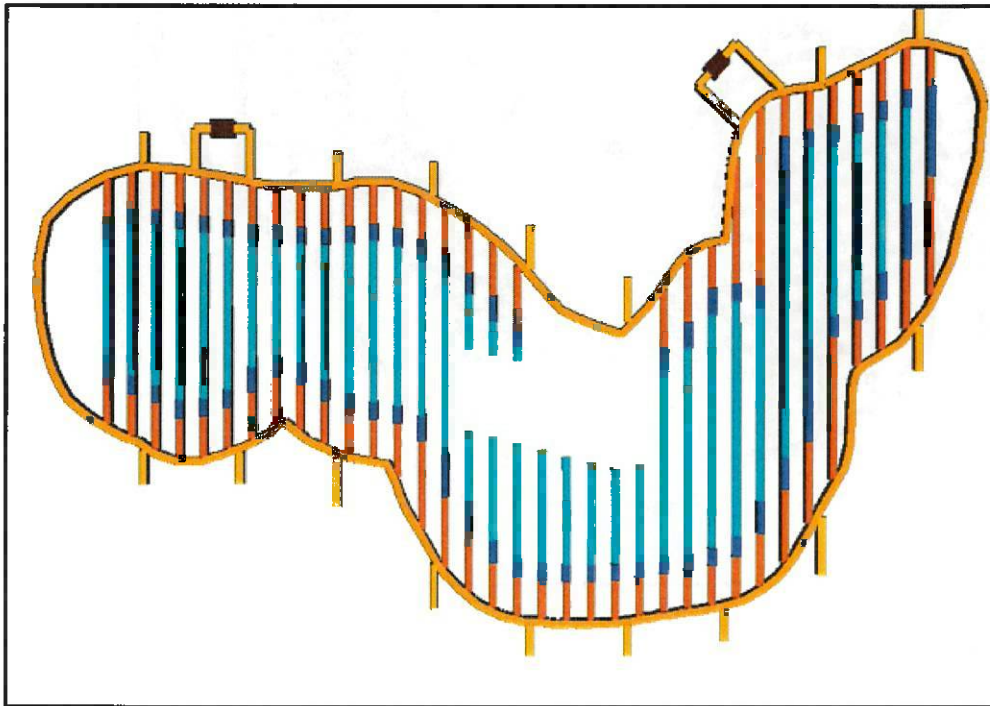


Figure 3-7: Plan view of the loading level of the BHOS (VUP-CSR, 2006).

The major risk for this project is a rigid assumption based on the industry norms. The data used for the case study was based on the previous projects from DeBeers with minimal consideration on the geotechnical and geology of the country rock in Venetia mine.

3.5 Research Approach and Design

The research is quantitative in nature and methodology used is applicable to the current problem. The Laubscher methodology is chosen because it is quantitative and have enough data on site to analyse the data. The challenges presented in Chapter 2 requires the need to further investigate, develop, and evaluate the mining method selected. Methodology attempts to re-evaluate accurately the geotechnical data that using RMR and MRMR ratings to validate the independency and dependency of mining method with respect to geotechnical considerations.

The data provided in the project report is based on the number of drilling projects conducted in Venetia Mine. The research has two sets of objectives to achieve, the first being assess the importance of technical shortcoming with selecting mining method based on previous report for mining method selection, and the second being the influence of geotechnical system in mining method selection decision.

3.6 Sources of Data

One class of data used in this study was the drilling data available in the mine and with the intention of accomplishing the study with high accuracy. The drilling data used is of higher degree of accuracy because it was directly collected from the data source (Anglo American Technical Division , 2008).

3.7 Data Inputs Variance

The data input variables are illustrated in table 3.4. A detailed description of data variance has been dealt in Chapter 2.

Table 3.4: Data used in the research

Analyses of data for mining method selection
Ore-body shape deposit
Deposit plunge
Ore-body thickness
Grade distribution
Depth of orebody (K01)
RMR/ MRMR
UCS
Lu Laubscher caveability graph

3.8 Sampling Process

Saunders et al., (2014) coined sampling as a process of selecting number of units (sample) from a collection unit and used to determine distinctive characteristics and can be generalised as the whole of population. Stratified sampling was implemented

in this study because it provides the best representation of the study's population for rock mass classification tests.

3.9 Data Collection Process

The study used data collection process as illustrated below:

- Review of mine history records,
- VREP project, and
- Underground site visits and ground water sampling (current developed faces).

3.10 Data Analyses

The data was initially organised in standard manner aligning it according to different rock facies. Following the completion of data collection, the tools listed below are used to process and analyse the data.

- GEMS version 6.2.2 to determine the major domains.
- Laubscher (1977) caveability graph.
- Microsoft Excel software.

3.11 Data Quality Assurance

Data was received at various stages through the study of this project. Main assumptions and historical data sets used during the analysis are as follows:

- 1) B. Venter Phase 1 (Geotechnical):
 - Early 1993 delineation drilling;
 - Only RQD;
 - Assumed joint condition rating = 26.24;
 - Fracture Frequency FF=JF;
 - 3 Joint sets.
- 2) B. Venter Phase 2 (RMR L90):
 - 1993 to 1997 delineation drilling;
 - RQD, JF, Total Core Recovery (TCR) available;
 - Assumed 3 Joint sets.
- 3) Anglovaal:
 - 1981 copper drilling program;

- Only TCR, RQD and FF;
 - Assumed Joint condition rating =26.24;
 - FF=JF;
 - 3 Joint sets.
- 4) W. Barnett Geotechnical (RMR L90):
- 1998 to 2002 geotechnical drilling. Drill holes GDH001 to GDH053;
 - Inclusion of structural geology;
 - Limited by grouping of drill hole intervals into equivalent domains.
- 5) VREP Drilling program:
- 2004 to 2006 Geotechnical drilling.
- 6) VREP 2 Drilling program:
- Post 2006 Geotechnical drilling program;
 - Data collection for Underground (UG).

3.12 GEMS Modelling Methodology

All drill logs were imported into GEMS. The drill logs were then interpreted into three-dimensional space to interpret the position of all the major domain boundary structures. The macro-structural logs, geotechnical logs, and the position of lithology units were used to determine the most likely structure positions. The process of modelling methodology is a process of clipping out various domains from original country rock.

3.13 Laubscher Graphs (Empirical)

Geotechnical rock properties were illustrated in Chapter 2. Data analyses mainly allows the interpolation of RQD, RMR and MRMR such that the following items are established:

- Hydraulic radius
- Caveability of the kimberlite

3.14 MMS Using MRMR Selection Tool

The MMS criteria data, highlighted in Table 3-4, was gathered and the input were used in the caveability graph to determine the hydraulic radius for caving.

3.15 Use of Microsoft Excel

Microsoft office excel was utilised to produce charts for the input data.

3.16 Ethical Considerations

The study did not require any ethical consideration due to the nature of it. All material used for the study were respectfully requested and acknowledged.

3.17 Limitation of The Study

The complexity of the data storage in the mine lead to confusion and time delays. This compromise the investigation which need to be carried out since most of the historical drill hole data are stored in separate places. Poor stacking and weathering of core samples posed a great challenge.

3.18 Chapter Summary

This chapter explained the entire research design that was implemented in this study. The mining method selection was reviewed, and the design process was discussed. Furthermore, all the design parameters, RQD, RMR, MRMR, and Laubscher caveability process were discussed with the purpose of placing an emphasis on research approach adopted in this study. In conclusion, limitation of the study and ethical considerations were conferred.

4 RESULTS AND DISCUSSION

4.1 Chapter Overview

Chapter 4 presents the primary data set which was collected and discussed in Chapter 3 for further elaboration on recommendations. The geotechnical consideration technique has improved the process of mining method selection. The variation which is caused by assumptions are dealt with objectively by using caveability linear graph which determine if the orebody will cave or not. The data from Chapter 3 were sorted out to ensure all required information is available and gathered.

4.2 GEMS Modelled Domain

This section explains specific work done with regards to changes in the interpretation of lithological domain boundaries as illustrated in Figure 4.1. Data used in Gems modelling are listed in Table A.1 and A.2 (Appendix A). The bounding fault for the Lezel-Tina (L-T) fault shear zone was updated and there were not any significant changes which were observed. It is noted that the shear zone contains anastomosing late brittle ductile faults that cannot be modelled in depth. The L-T fault on the eastern boundaries of the shear zone is the most prominent fault and was modelled as a major domain. All drill hole logs and mapping show numerous shear zones and weak rock mass. In this study, weak rock is interpreted as to be part of a separate domain, confined to weak rheology of metasediments.

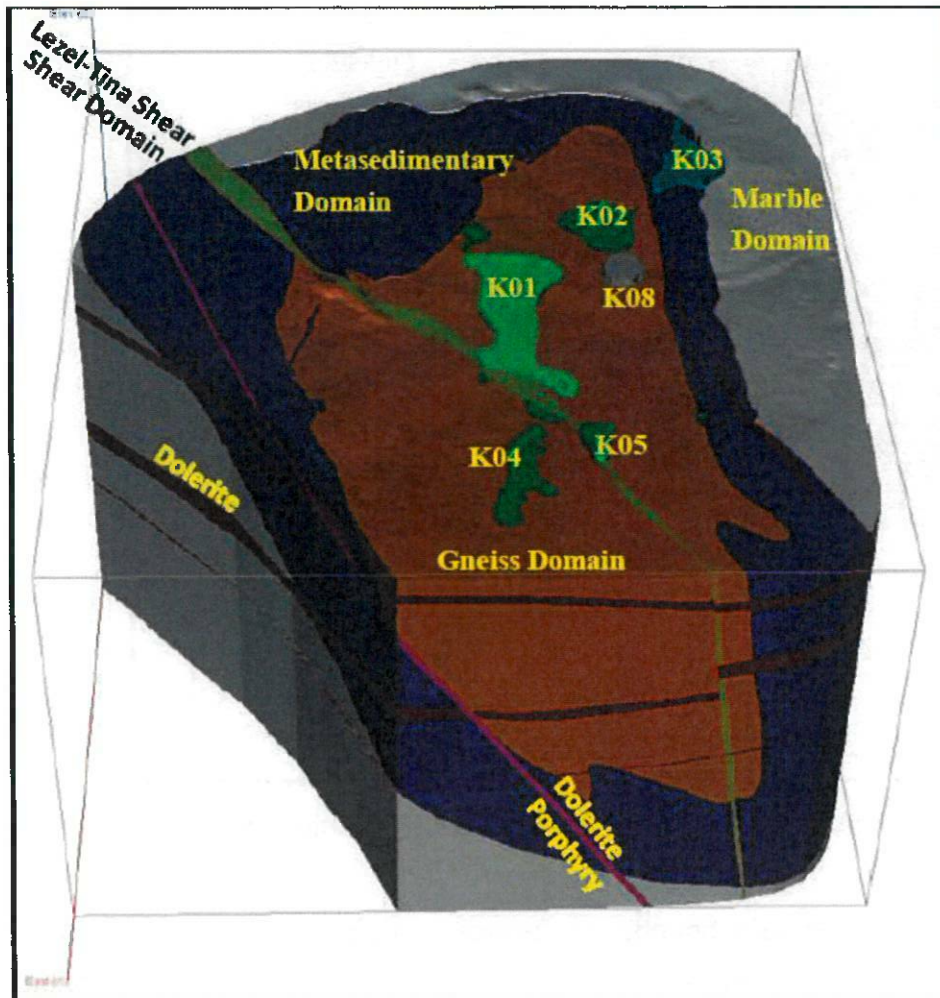


Figure 4-1: Isometric view northwards of completed lithological domains

Borehole GDH115 shows double intersection of both dolerite and L-T fault and the bottom intersection cannot fit east sill elevation. Therefore, the bottom elevation is interpreted to be in unusual dip in sill elevation as seen in Figure 4.1. The top intersection thereby serves to illustrate complexity of shear zones. The L-T fault is interpreted to be on the east of the top section and the primary displacement on the dolerite must be on the west of the top section. The position of L-T fault does not necessarily represent the position of the maximum brittle - ductile displacement through the zone.

There is additional dolerite in GDH085 included in the model and can be interpreted in many possible ways. Figure 4.2 illustrates porphyritic dolerite dyke.

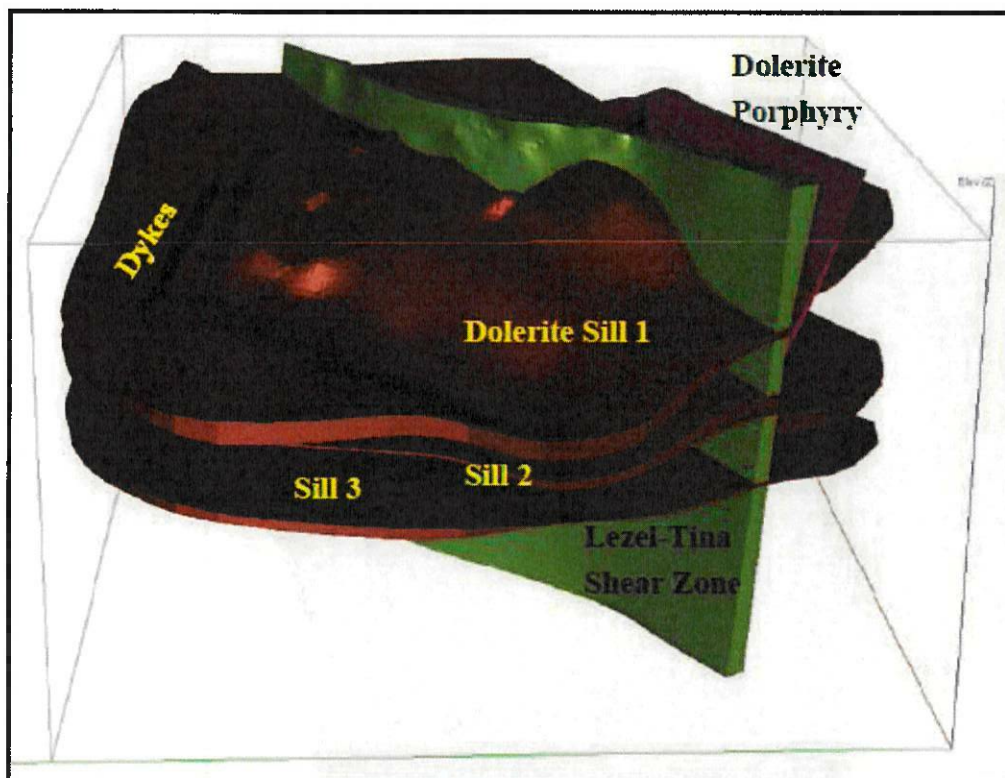


Figure 4-2: Modelled cross-cutting lithological domains, Dolerite dykes and sill

The major lithological domain boundaries between gneiss and metasedimentary package was updated. Only the minor contact adjustment needed to the contact position to get the contact to fit new drill hole data. In all cases the geometric lineation trends was maintained. The variation in techno stratigraphy is likely to be associated with L-T shear zone.

4.3 Country Rock Domain

The rock mass classification data is reduced into manageable engineering domains to simplify analysis and engineering work. Spatial definition of the domains must happen prior to analysis to tag drill hole with domain name for use in the analysis. In some cases, domains are speculative or negated as significant feature during analysis. Visual inspection of the IRS and FF was undertaken to identify additional geotechnical domains that cannot be determined from completed lithological and structural domains. There were no additional domains identified.

The report for slope design indicates a reduction in RMR from about 60m below surface. It is estimated an average of 5m reduction on the RMR in the upper weathered zone and there is a gradational transition that occurs anywhere from around 100m below surface. The visual spatial inspection of the data tentatively suggested two additional geotechnical domains (Figure 4.3). The first being Metasedimentary domain and the second is Metasedimentary domain between slimes dam fault zone and L-T fault shear zone. The two domains are modelled as solids to tag the rock data.

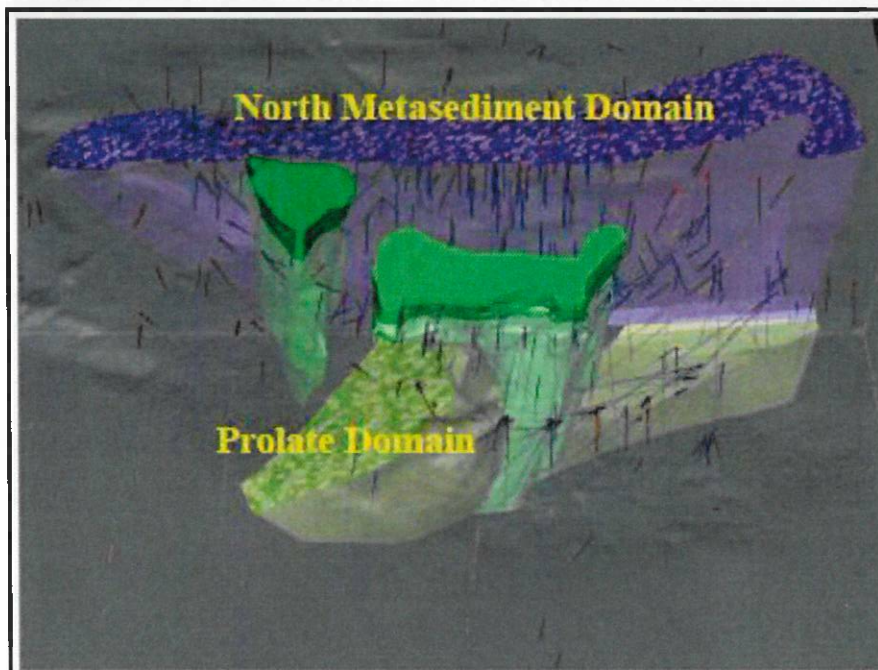


Figure 4-3: Additional possible geotechnical domains in the country rock tested in the study

4.4 DVK Kimberlite Domains

Analyses were also undertaken to define possible geotechnical domains in the kimberlite pipes. One of the most critical geotechnical domains to define was the altered and weathered DVK facies and ADVK facies (Figure 4.4). The available UCS data for DVK and ADVK was analysed to determine if statistical strength domain could be identified. The data clearly show that both ADVK and DVK rock strength bimodal in distribution (Figure 4.5) with higher strength component and a lower strength component. Two facies UCS approximated somewhere between 80MPa and 100MPa respectively. Although the UCS data is accurate measurement, it is also typical biased

towards higher quality rock samples. Samples below 40 MPa are not likely to be easily sampled and analysed.

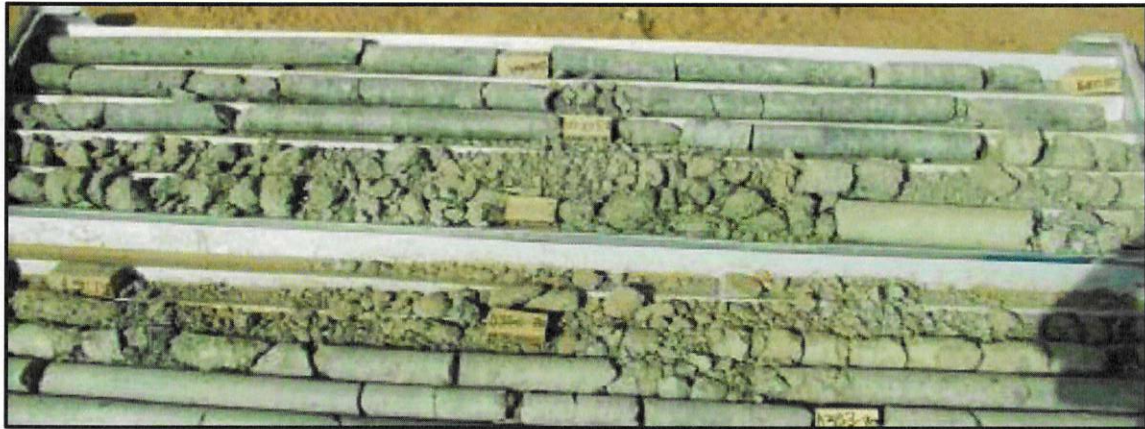


Figure 4-4: DVK Kimberlite in drill core, showing the contrast between competent, weak, and altered kimberlites (VUP-CSR, 2006).

Drill core logging field estimates IRS were also studied to see variation in rock strength (Figure 4.4). These estimates are more abundant than the laboratory data and are more representative of the distribution of rock strength. The recorded values fall into seven categories as represented in Table 4.1.

Table 4.1: IRS estimation categories used by Venetia drill core loggers

	Category	Minimum (MPa)	Maximum (MPa)	MPa value used for RMR
0	Extremely Weak Rock (0.25-1.0)	0.25	1	0.625
1	Very Weak rock (1.0-5.0)	1	5	3
2	Weak rock (5.0-25.0)	5	25	15
3	Medium Strong Rock (25-50)	25	50	37.5
4	Strong Rock (50-100)	50	100	75
5	Very Strong Rock (100-250)	100	250	175
6	Extremely Strong Rock (>250)	250	600	425

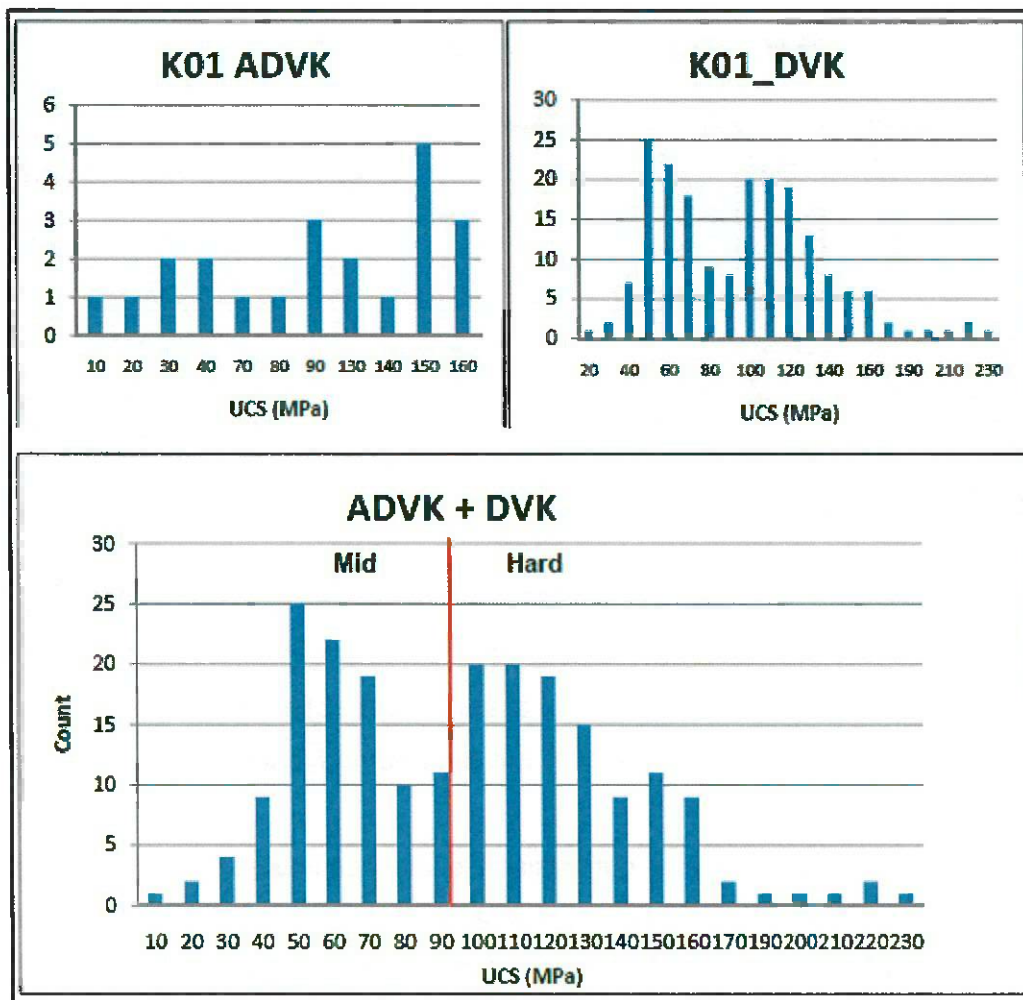


Figure 4-5: Distribution of laboratory UCS results for DVK RMR classification.

Histograms showing the distribution of laboratory UCS results for DVK (Figure 4.5) RMR classification value for each drill hole interval is determined by using the mid-point IRS value (MPa) in each category in Table 4.1. Note, the MPa's value assumed is the midpoint value in each category as per Table 4.1. Figure 4.6 shows the actual histogram of IRS for DVK kimberlite with MPa values more broadly distributed than just the category mid-points.

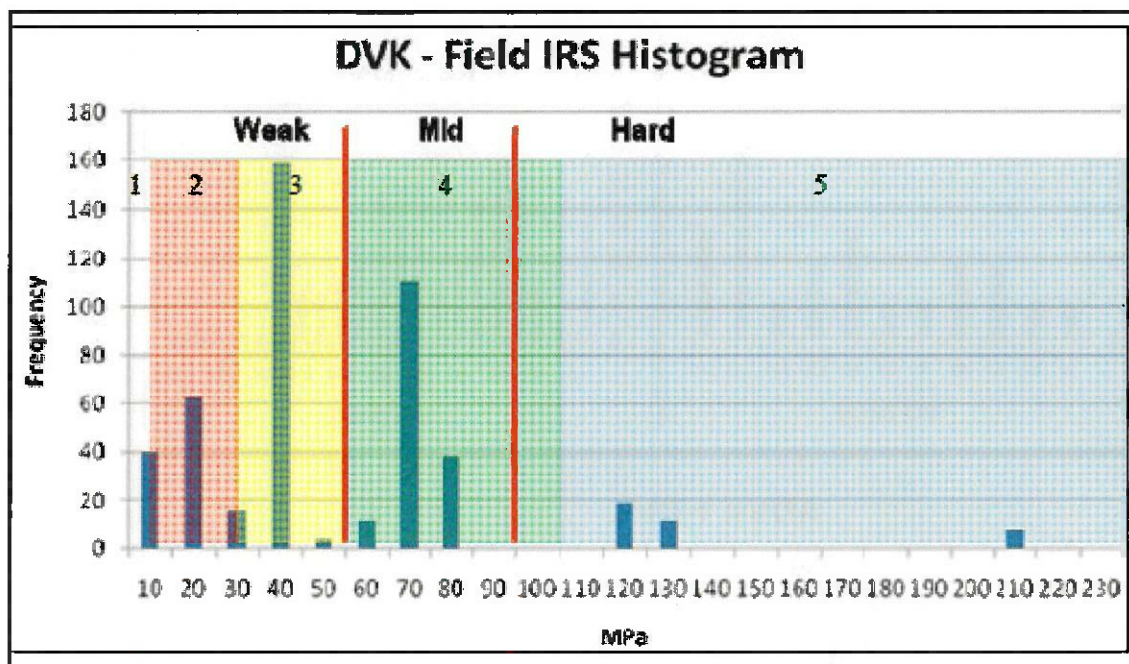


Figure 4-6: Histogram showing the distribution of intact rock estimates for DVK from drill core logging. The logging categories (0-6) are defined and labelled.

The percent abundance of the weak component is also estimated. The final rock strength used therefore takes variability in rock strength into account. The adjustment has been done according to Laubscher & Jakobs (2001). The spread of data in Figure 4.6 is biased by the experience of the logger and this fact should be understood during interpretation. The IRS distribution from drill core suggests that most of DVK intervals have strength values between 25MPa and 50MPa, and between 50MPa and 100MPa. DVK stronger than 100MPa is rare while lower than 40MPa is common. This interpretation is quite different from UCS determined in the laboratory. Based on analysis of both UCS and field IRS data distribution (Figure 4.5 and 4.6), it was decided to define three statistical division as follows:

- 0 – 50MPa Weak division;
- 0 – 100MPa Mid strength division;
- 100 – 240MPa Hard strength division.

ADVК is formed from DVК that has undergone clay alteration by metasomatic fluids. The percentage of clay is variable and appears to be gradational range in clay between DVК and ADVК. DVК and ADVК is mainly mineralogical and they should be

considered as lithological domains. The similarity in the distribution of strength values between DVK and ADVK in Figure 4.6 supports the interpretation that these two domains are not easily separated as geotechnical domains with distinct properties.

Two surfaces were used to split DVK and ADVK lithological solids into three geotechnical (Table 4.2).

Table 4.2: Summary of domain definition and solids created for K01 DVK facies

Lithological Domains	ADVK	DVK	Geotechnical IRS or RMR Domains
Solids Created	ADVK_Weak	DVK_Weak	K01DVK_Weak
"sub-domains"	ADVK_Mid	DVK_Mid	K01DVK_Mid
	ADVK_Hard	DVK_Hard	K01DVK_Hard
MRMR Domains	Assume All Weak	Use As-Is	

The accuracy and robustness of defined domains are negatively influenced by logging delays caused by the weathering of the kimberlite after it was removed from the drill hole. The kimberlite core starts to degrade as soon as it is drilled. For the calculation of MRMR it is recommended to down grade ADVK. The assumption is these sub domains would be exposed to ground water during mining and would reduce strength. The DVK domains used Figure 4.7 shows the solids generated by GEMS and illustrated 50MPa and 90MPa of the DVK and ADVK respectively.

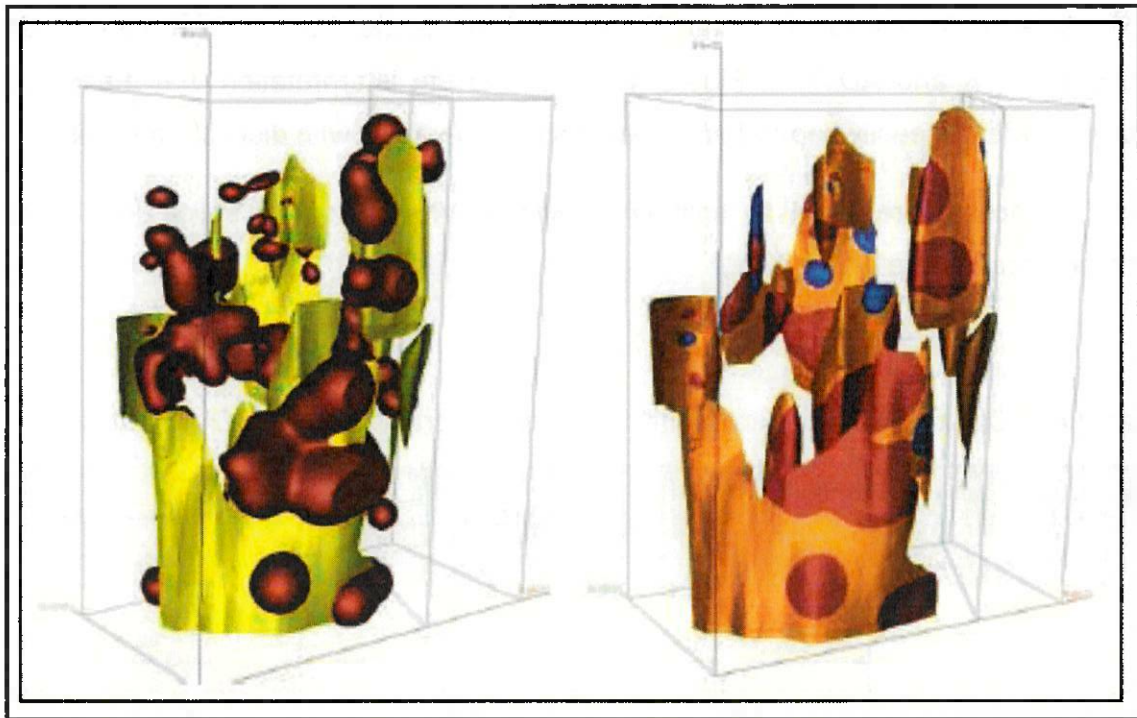


Figure 4-7: GEMS clipping of DVK and ADVK lithological domains by IFR shells

4.5 Kimberlite Contact Domain

An attempt was made during this study to evaluate the drill core contacts statically to determine if there is a significant change in RMR near K01 pipe contact zone. The lack of drill core data analysis at the contact zone obviate calculation of RMR. In conclusion, no detailed analyses were done because of the lack of drill core information. Mokele (2019) stated that, this should not be a challenge, because in situ RMR at the contact doesn't change, only the MRMR that is reduced during mining due to stress was affected.

4.6 Geotechnical Data Analysis

4.6.1 Rock Mass Properties

The rock mass properties have been assessed from the triaxial laboratory tests by using the Hoek-Brown failure criterion (Hoek et al., 2002). The input parameters used for rock tests are the average UCS value (Table 4.3), triaxial test results Geological Strength Index (GSI, calculated from average RMR where $GSI = RMR - 5$, (Hoek, 1994) (Hoek, et al., 2002)) and the density. The triaxial data was used in RocLab to determine the best fit m_i value (Appendix A, Table A.1). From the RocLab finding the triaxial data was limited and inconsistent. Therefore, a certain degree of subjectivity was required to remove suspicious values and get m_i values that are sensible.

Table 4.3: Average UCS values used in RocLab

Rock Type	Average of Uniaxial Compressive Strength (MPa)
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

The Hoek-Brown parameters (m_b , s and a ; defined in Hoek et al., 2002) are produced in RocLab. Also determined are the Mohr-Coulomb equivalent parameters, rock mass cohesion (c) and rock mass friction (σ) to the Hoek –Brown parameters. This determination is based on assumptions about the magnitude of the maximum value of confining stress σ_3 above which failure cannot occur, and we used the “Tunnel” assumption in RocLab with a depth of about 700 m Additional parameters determined

are the rock mass compressive (σ_{cm}) and tensile strength (σ_{tm}), and the deformation modulus (E_m).

Table 4.4: Rock mass properties determined by RocLab (Tait, 2007)

Rock Mass Mechanical Properties							
Lithology	GSI (g/cm ³)	m_b (-)	c (MPa)	ϕ (degrees)	σ_{cm} (MPa)	σ_{tm} (MPa)	E_m (GPa)
Gneiss Package	60	3.1	3.9	46.1	14.3	0.5	32.5
BS only	60	2.8	3.2	42	9.3	0.36	26.7
Dolerite	52	3.5	4.9	52	17.5	0.35	30
Marble	55	3.4	3.6	46	9.5	0.23	33.1
Metasediment Package	52	1.5	3.6	42.7	12.4	0.57	24.6
Shear Zone	50	1.3	3.1	40.7	9.8	0.46	21
K01_CK	58	1.7	3.3	40.3	11.4	0.65	23
K01_DVK_Mid	60	3.1	2.7	40.5	6.7	0.24	13.7
K01_DVK_Hard	69	4.8	4.8	48.7	21.8	0.82	34.1
K01_DVK_Weak	43	1.3	1.5	29.8	1.5	0.05	3.7
K01_MVK	60	2.3	2.47	37.6	6.2	0.29	11.8
K01_CRB	55	2.1	4	44.8	35.4	0.56	29.1

4.6.2 Joint Statistics

Barnett (2007) summarizes the joint patterns at Venetia Mine (Figure 4.8). The brittle tectonic history is not well constrained but is likely to be complex with many reactivations. In summary, the primary joint sets are as follows:

- J0 is parallel to the gneiss or schist foliation planes. These joints often have biotite on their surface since the thin (<1cm) concentrated foliation of biotite define the direction of weakness of the rock;
- J1 is a pervasive sub- vertical joint set in the pit that strikes between NNW and NNE. In aerial photograph, these circa N-S lineaments appear to form discrete concentrations, sometimes en-echelon in pattern. The lineaments trend is common throughout the Limpopo Belt and Zimbabwe craton;
- J2 is the most well developed sub-vertical joint set and strikes NE-SW. The orientation is parallel to the Lezel shear system and other parallel faults identified in aerial photographs and is very likely related to the brittle reactivation of these shear zones;

- J4 is set at a sub – vertical set that strikes *circa* WNW-ESE. This is a common and very strongly developed Karoo aged dyke emplacement trend. The trend is known as Okavango dyke trend (Reeves, 2000);
- J6 is the intermediate to shallow SW dipping joint set. Mapping shows it, to define a strong fracture zone passing SE of the pit, partially through K01 kimberlite and north of K01 pipe;
- J5, J7, and J8 are intermediate to shallow dipping fractures. They are typically poorly developed and occasional appeared in some domains;
- J9 is typically found in dolerite sills and kimberlite.

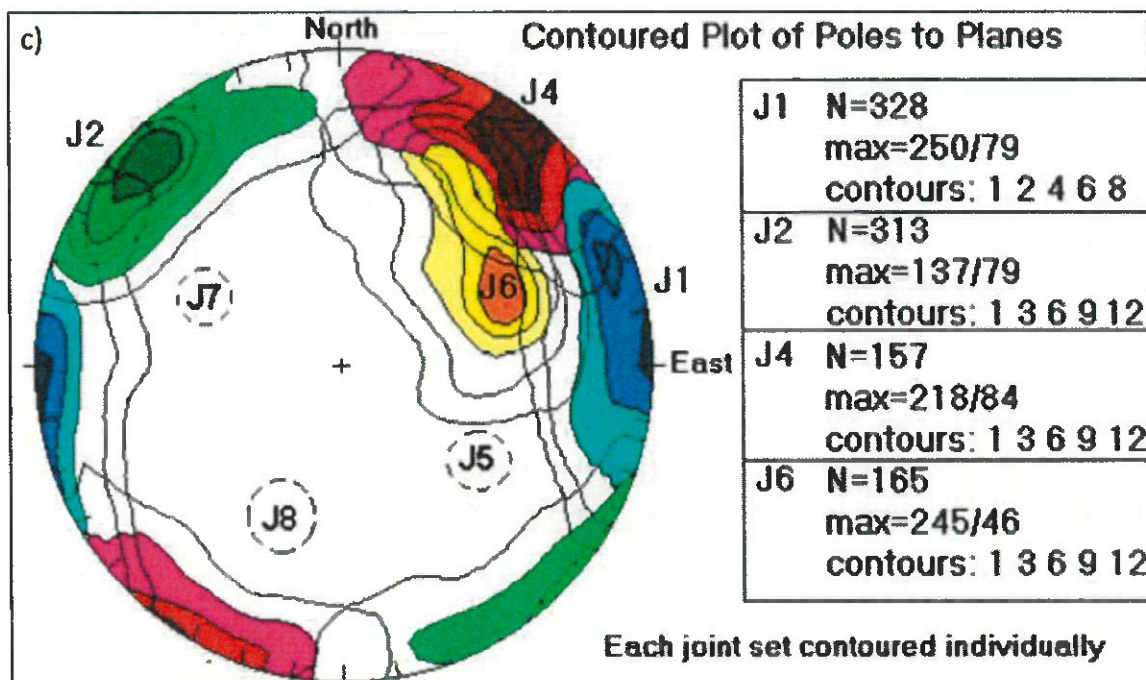


Figure 4-8: Contoured plot of poles to the 4 dominant joint sets (Barnett, 2007).

The mapping database was used to get a range in joint spacing, trace length and characteristics in some Domain. Not all joints set sampled in every part of mine. Table 4.5 gives indication of where joint sets are located and should be understood that the estimated frequency is not applicable in certain domains. Gneissic Domain joint set are included in the data base (Figure 4.9). Mapping was done in the kimberlite and

joints set are illustrated in Table 4.6. DVK kimberlite Domain joints sets are illustrated in Table 4.7.

There are over 100 measurements per primary set (J1 to J4) and between 29 and 78 measurements for each of the other sets, except J9 that has only 2 measurements.

Table 4.5: Gneissic domain joint set summary

JOINT SET	SPACING (m)		LOCATION	TRACE LENGTH (m)		ORIENTATION	
	Ave	Range		Ave	Range	DDIR	DIP
J1	0.91	0.01 – 10	All	1.33	0.05-30	249	84
J2	1	0.01 – 10	All	1.11	0.05-20	135	78
J3 / J0	0.45	0.01 – 7	All (Not intrusive)	2.61	0.05-30	Foliation parallel	
J4	1	0.01-15	6, 9, 10	1.47	0.01-20	219	84
J5	1.32	0.01-5	8	1.38	0.01-15	290	36
J6	1.06	0.05-12	3, 4, 6 (best)	1.45	0.05-15	240	48
J7	1.62	0.1-10	8, 9	1.21	0.1-15	128	48
J8	0.97	0.05-15	2, 3	1.14	0.05-10	60	48
J9	0.51	0.1-3	2, Dolerite?	1.72	0.1-20	Sub-horizontal	

Table 4.6: Kimberlite joint set summary

JOINT SET	SPACING (m)		LOCATION	TRACE LENGTH (m)		ORIENTATION	
	Ave	Range		Ave	Range	DDIR	DIP
J1	0.77	0.02 – 2.5	All?	0.37	0.01-2	234	77
J2	0.95	0.01 – 10	All?	0.51	0.01-6	145	82
J4	1.6	0.2-3	Least common	0.47	0.05-15	195	73
J6	0.59	0.05-4	Least common FZ1	0.88	0.2-3	225	25
J7	0.88	0.2-3	Least common	0.75	0.2-2	124	43
J9	0.61	0.5-4	Least common	1.2	0.05-20	Sub-horizontal	

Table 4.7: Summary of the joint spacing determined from drill hole data in DVK

JOINT SET	TRUE SPACING (m)	STD. DEV.	COUNT
J1	5.5	6.1	13
J2	1.1	2.3	107
J4	1.2	2.7	55
J5	2.5	5.4	53
J6	0.4	1.2	167
J7	0.7	1.2	65
J8	3.2	5.6	43

4.6.3 RMR Calculation and Description

The drill holes are imported into spread sheet for RMR calculations. Figure 4.9 illustrates the RMR values for K01 orebody. Bieniawski (1989) produce a set of guidelines for selection of tunnel support in which the RMR has been determined. These guidelines have been used for several years and are also applicable for this study. For the case considered previously, an assumption was made to choose block cave mining as a preferred method for K01 orebody. Table 4.8 suggests that the RMR values determined indicate the rock mass which is fair to good rock. Therefore, this implies that the rock condition is stable for caving. The value of RMR of 48 represent a fair rock which can still cave at normal condition with enough hydraulic radius. The spreadsheet calculates RMR based on Laubscher (1990) as well as in-situ rock mass rating (IRMR) of Laubscher & Jakubec (2001). However, the RMR deals with a variety of assumptions that are different in each historical logging at the mine.

Additional assumptions made during RMR calculations are:

- Rubble zone were assigned a JF = 40/m and joint condition rating = 0;
- Large joint geometry assumed 95% in all cases;
- It was assumed that there is no joint sampling bias and JF was not adjusted;
- Rock strength is derived from average UCS for rock type, except where logs indicate a reduced strength.

Adjustment were also made to the RMR calculation depending on structural geology class observation. These adjustments prevent overestimate of rock mass strength. Geology adjustments observations are as follows:

- Class 1a: strongly sheared. Adjusted IRS to equal that of shear zone laboratory UCS strength;
- Class 1b: Breccia. Adjusted to micro fracturing estimate to 15/, if not already estimated;
- Class 2: Displacement or striations. No adjustment, discrete macro structure is assumed;
- Class 3: Rock mass is weakened by alteration or strong fracturing. Adjusted IRS equal to shear zone laboratory;
- Class 4: Core is completely broken. Adjusted FF to be 40/m;
- Class 5: Core is completely altered. Adjusted weak rock strength to equal 2.5 MPa.

Depending on the assumptions made during RMR estimate, that estimate is given a confidence level. For example, logs that do not have joint property conditions have assumed and resulting RMR is given low confidence. The calculation of the RMR has indicated a ground condition which has a good rock condition. This entails a caving method wholly dependent on pre-fracturing to reduce the RMR to allow for caving to occur.

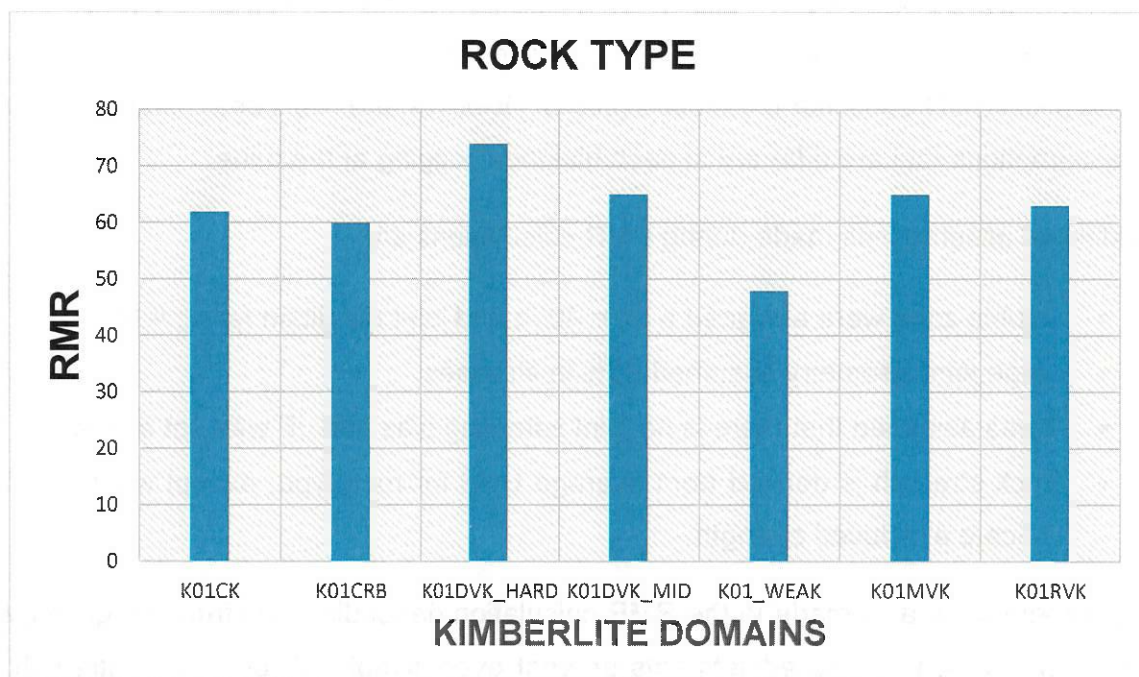


Figure 4-9: RMR values for K01 kimberlite orebody

Table 4.8: Guidelines for excavation and support of 10m span rock tunnels in accordance with the RMR system, (Bieniawski, 1989).

Rock mass class	Excavation	Rock bolts (20 mm diameter, fully grouted)	Shotcrete	Steel sets
I - Very good rock RMR: 81-100	Full face, 3 m advance.	Generally no support required except spot bolting.		
II - Good rock RMR: 61-80	Full face, 1-1.5 m advance. Complete support 20 m from face.	Locally, bolts in crown 3 m long, spaced 2.5 m with occasional wire mesh.	50 mm in crown where required.	None.
III - Fair rock RMR: 41-60	Top heading and bench 1.5-3 m advance in top heading. Commence support after each blast. Complete support 10 m from face.	Systematic bolts 4 m long, spaced 1.5 - 2 m in crown and walls with wire mesh in crown.	50-100 mm in crown and 30 mm in sides.	None.
IV - Poor rock RMR: 21-40	Top heading and bench 1.0-1.5 m advance in top heading. Install support concurrently with excavation, 10 m from face.	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh.	100-150 mm in crown and 100 mm in sides.	Light to medium ribs spaced 1.5 m where required.
V - Very poor rock RMR: < 20	Multiple drifts 0.5-1.5 m advance in top heading. Install support concurrently with excavation. Shotcrete as soon as possible after blasting.	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert.	150-200 mm in crown, 150 mm in sides, and 50 mm on face.	Medium to heavy ribs spaced 0.75 m with steel lagging and forepoling if required. Close invert.

4.6.4 Modified Rock Mass Rating

MRMR by Laubscher (1984), Laubscher and Taylor (1976) and Laubscher and Page (1990) was used to assess the rock mass rating for K01 orebody against the hydraulic radius and if caving will be successful or not. The value generated from the calculation of the RMR were used to determine the MRMR. Appendix B illustrates the calculation of MRMR and the adjusted assumptions. To derive the MRMR, the basic adjustment is required from RMR given by Laubscher (1990) and summarised in Table 4.12. Table 4.13 illustrates the MRMR value for each kimberlite domains.

Table 4.9: MRMR mining adjustment (Laubscher & Jakebc, 2001)

Parameter	Possible adjustment%
Weathering	30-100
Orientation	63-100
Stress	60-120
Blasting	80-100

Table 4.10: MRMR value for each kimberlite domain

Domain (K01)	CK	CRB	DVK_HARD	DVK_MID	DVK_WEAK	MVK	RVK
MRMR	45	44	54	47	35	47	46

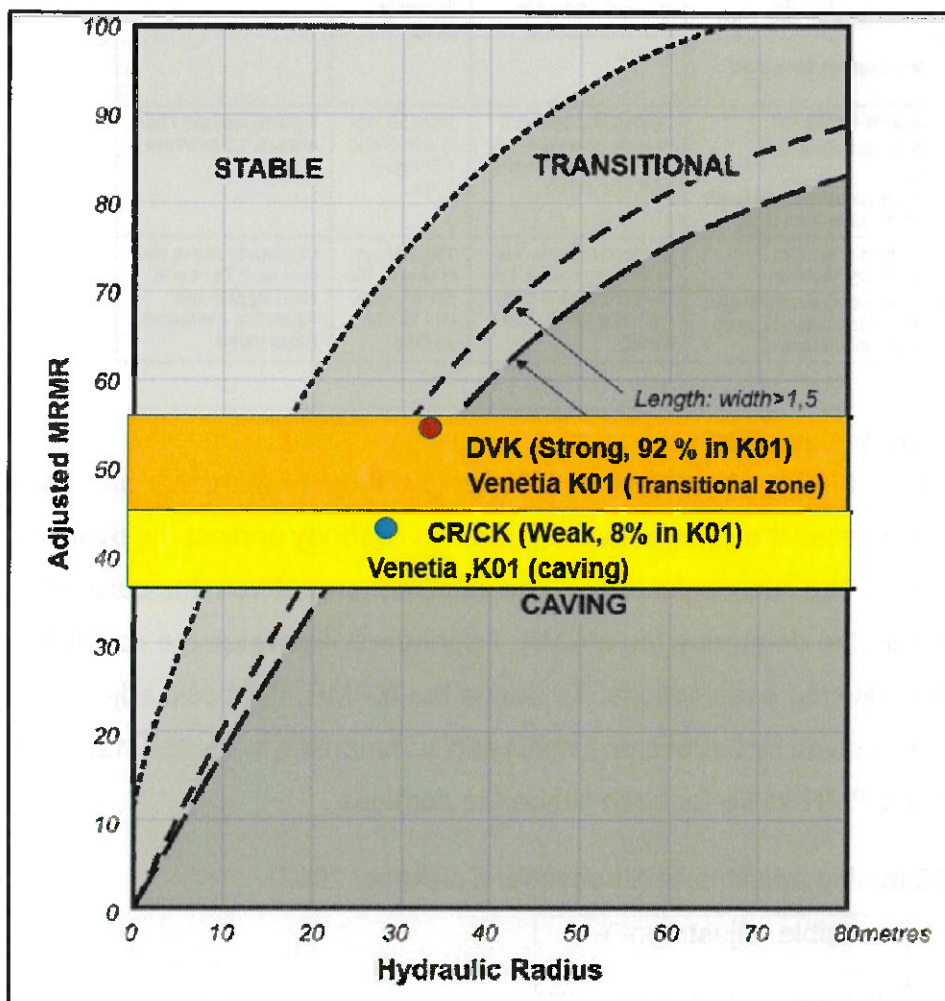


Figure 4-10: Hydraulic radius for K01 caveability (Suorineni et al., 2014).

4.7 Chapter Summary

The geotechnical data analysis and Gems modelling brought an improvement in the understanding of the country rock lithology for the kimberlite domains. The biggest concern remains with respect to the spatial variability of the sub-facies of DVK and ADVK that is susceptible to weathering, and whether that susceptibility can be quantified in terms of rate of weathering and strength reduction. The contact zone of the kimberlite remains a concern since very limited information is available.

The geotechnical data used to analyse the seven facies studied. The DVK which is considered strong and CK which is considered weak were plotted in the Laubscher caveability graph. The results indicate DVK sitting in stable condition while CR sitting in caving position. The results further insinuate that K01 may not be mined by single mining method. To evaluate this further, the area was calculated for k01 and found that, 92% of kimberlite is DVK which helps to decide. As a result, the mining method employed shall cater for DVK with the least of material coming from the weak zone.

The MRMR for DVK is 54 which allows caving for material to occur at the hydraulic radius of 35m. This characteristic poses a serious challenge in K01, since it is irregular in shape and the narrow width limits the caving radius to maximum of hydraulic radius of 30m with the narrowest area in east at 20m. While the CR require hydraulic radius of 29m to cave, to overcome such challenge a sub level caving mining method should be applied. The SLC will cater for both the weak and strong orebody. Some country rock has same MRMR rating with kimberlite, care must be taken to control dilution when applying SLC. Finally block caving is only economic with substantial blocks for CR/CK. The other challenge for block caving is when having multiple lift transitioning from one lift to other, block caves have long ramp up and ramp down profiles as the lower caves will destabilise the upper cave resulting in production gap.

5 CONCLUSION AND RECOMMENDATIONS

5.1 Chapter Overview

This chapter discusses the results from Chapter 4 and seek to address the way forward. Gems software was used for modelling kimberlite facies in Chapter 4 and it deals with the key component of lithological facies to address the significance of geotechnical engineering in selection of mining method. Venetia Mine is currently mining an open pit, it is certain that a cave method will be implemented when transitioning to underground. However, as discussed in Chapter 4; section 4.3 and 4.4; the Venetia orebody tend to be irregular with strong zones, weak zones and this tend to pose a serious challenge of selecting a single mining method for the whole orebody. This chapter summarizes the project report findings, give conclusion and recommendations.

5.2 Key Design Area

Technically an orebody can be mined with any mining method, the challenge however is extracting the ore safely, predictably and economically. The known mining methods typically have fixed application criteria based on the rock quality and resource value. Based on the typical industry applications, it is usually easy to narrow down the suitable mining methods to any ore body deposit. Venetia deposit is typical vertical weak orebody with weak country rock.

5.3 Design of The Undercutting

Taking into consideration that the MRMR for kimberlite domains range from 35 to 54 and hydraulic radius range from 20m to 35m, the undercut length has been measured using the Microstation for levels 46 and 54, respectively. These two levels were chosen as there are the current waste development tunnels in the Decline Shaft. Figure 5.1 illustrates the kimberlite profile at 46 and 54 level respectively. The profile depicts a narrow, irregular and undulated orebody on the eastern part. To calculate the dimension of the undercut, the following equation was used below:

$$\text{Hydraulic radius} = \frac{\text{Area}}{\text{Perimeter}} \dots\dots\dots \text{Equation 5-1}$$

$$= \frac{L \times W}{2(L+W)}$$

The calculation for the hydraulic radius are shown in Appendix B since the hydraulic radius range between 20m and 35m and the undercut length range between 65m and 141m. The minimum undercut width is 69m with the maximum undercut range 202m. This is expected as the DVK facies shown a maximum hydraulic radius of 35m. It is not feasible to induce an undercutting with the dimension of 71m x 160m for DVK_Hard facie. This is clear indication that the K01 orebody will not cave and as a result another alternative method has to be chosen. Figure 5.1 shows the layout of 46 level and 54 level lifts as would be planned for Venetia Mine block caving. The Red area would be actual caving foot print for hydraulic radius to be achieved.

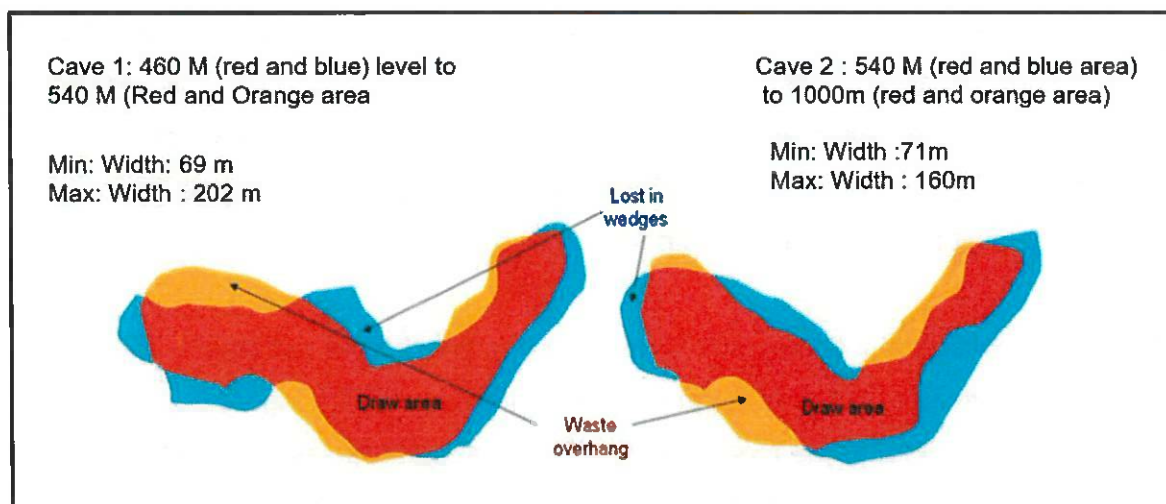


Figure 5-1: 46 level (on the left) and 54 level (on the right) as would be planned for Venetia.

Layouts show overhang and wedge material in orange and blue, respectively. Red area is the actual caving foot print in which hydraulic radius must be achieved. The challenge with applying block caving to Venetia is due to the narrow orebody width and highly discontinuous nature of the contacts. The narrow width is limiting the caving radius to a maximum of 30m with the narrowest area in the east at 20m. This puts Venetia in the transitional zone thus not stable but not definitely caving. To overcome this small block heights and some form of caving assistance are required. There are however areas in the east that are simply too narrow to ever cave and will remain

stable. It is difficult to try and apply two mining methods simultaneously – one in the east and one in the west as block caving will be destabilising the rock mass in the areas where people will be working so the mine would have to extract the east first and then the west. This limits the production rate and allows waste to migrate from the east to the west when caving starts.

A further challenge is that, there are significant overhanging waste areas which will cave as the waste has a similar RMR to the kimberlite and this will bring in early dilution. There are also significant amounts of kimberlite in the wedge areas which will be lost as the waste from the slope failure rushes in and cuts off the wedges.

Finally block caving is only economical with substantial block heights due to the high capital cost thus using low lift caves will be uneconomic, will have poor fragmentation thus further increasing cost and reducing production. Block cave has significant challenges when applying multiple lifts. The transition from lift to the other lift results in long ramp up and down profiles. The lower cave will destabilise the upper cave which will result in a production gap between the caves.

Central to caving methodology and current industry practice on average, a 240 - 250m undercut length is regarded as a benchmark (Chitombo, 2010). Figure 5.2 and Figure 5.3 illustrate the total areas measured for each kimberlite facies. The distinct results consist of small patches of facies which may results in two separate mining method. For block caving to succeed see Figure 5.4 for 46 level, number of tunnels to be developed for each facies differs and current industry practice of 11 tunnels develop for a block caving (Glenn, 2020).

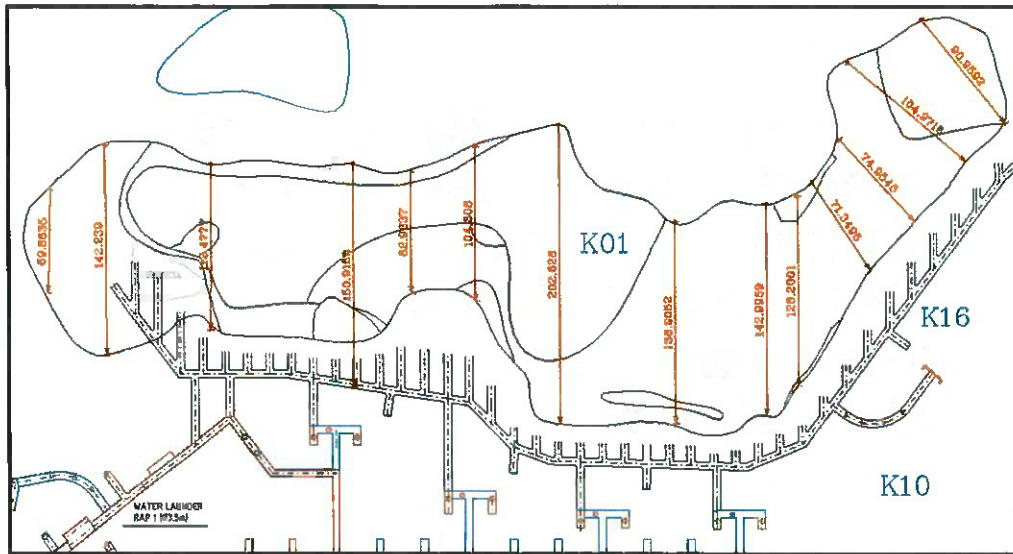


Figure 5-2: 46 level undercut design measurement.

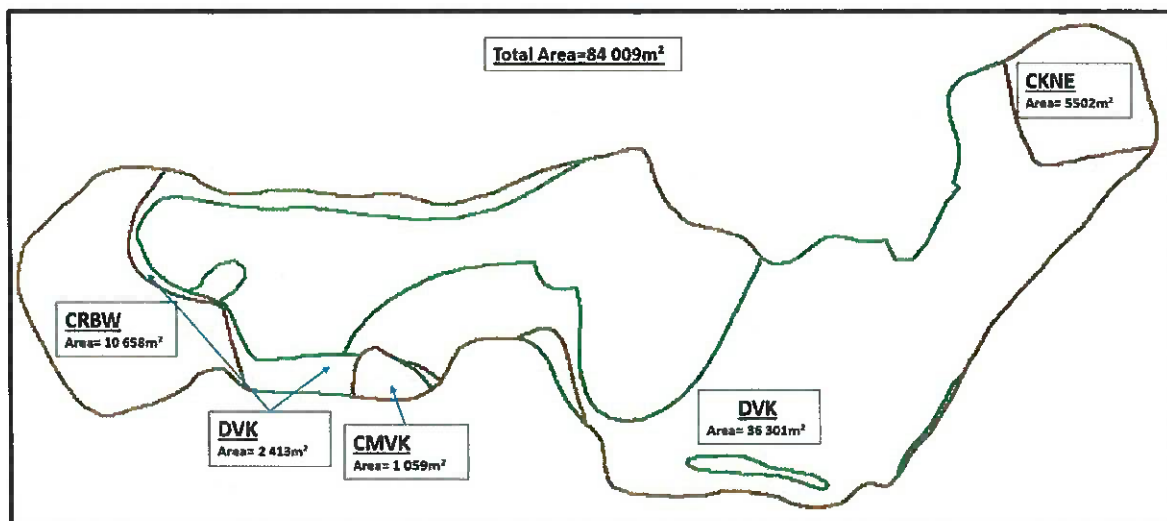


Figure 5-3: 46 level kimberlite measured area.

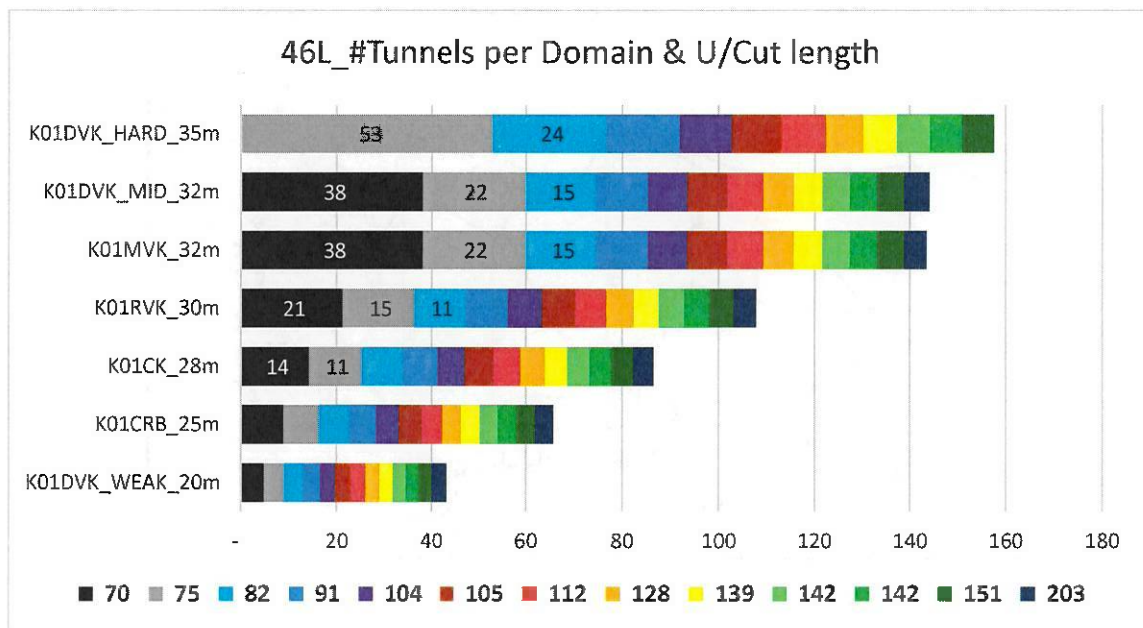


Figure 5-4: Number of tunnels by undercut length and domain type 46 level

The challenge in applying block caving to K01 see Figure 5.3 and 5.4 is due to the narrow orebody and the highly discontinuous contacts. The width of the orebody limits the achievable caving radius to a maximum of 35m with the narrowest area in the east at 20m. On Laubscher's caving prediction chart, K01 requires hydraulic radius of 35m for DVK, thus it requires at least a 140m by 140m square to be undercut to achieve continuous caving. However, as the orebody is on average of 120m wide this is a significant challenge. K01 thus lies in the transitional zone, thus Figure 5.7 for 54 level illustrates number of tunnels to be developed for caving to occur. There is significant number of tunnels to be developed before caving starts which will create a production gap.

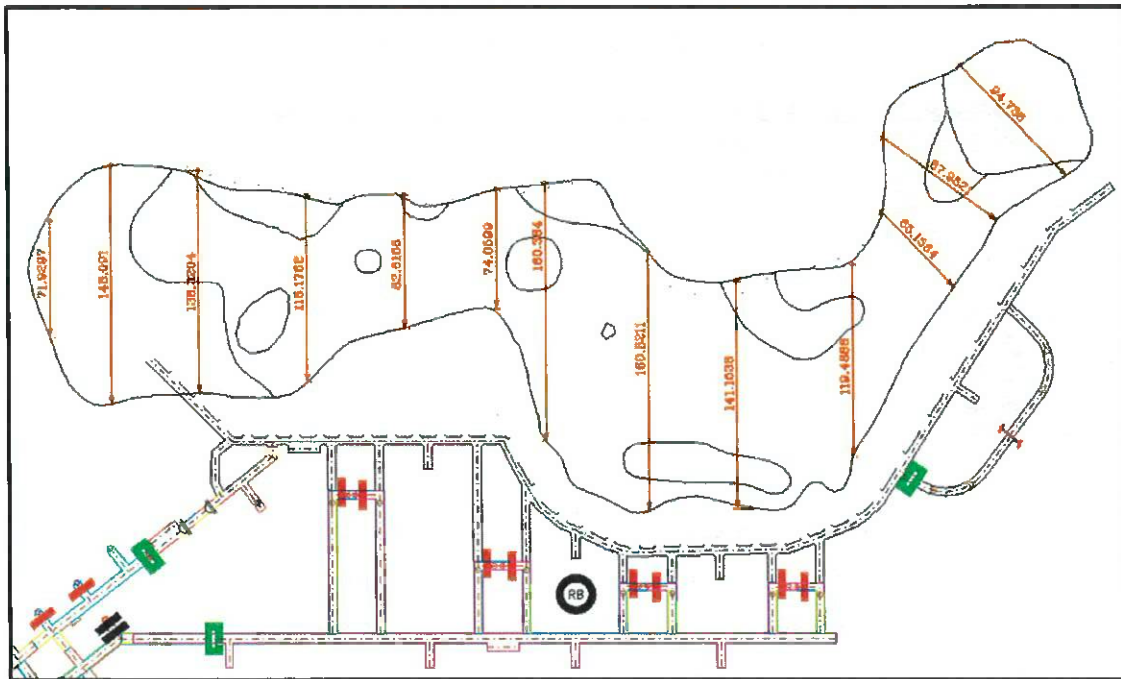


Figure 5-5: 54 level undercut design measurement.

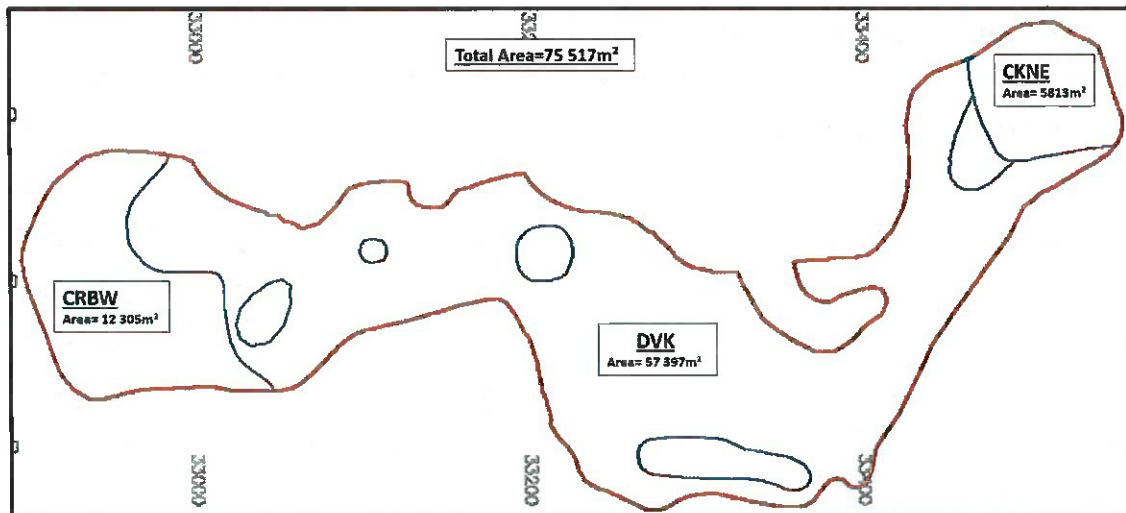


Figure 5-6: 54 level kimberlite measured area.

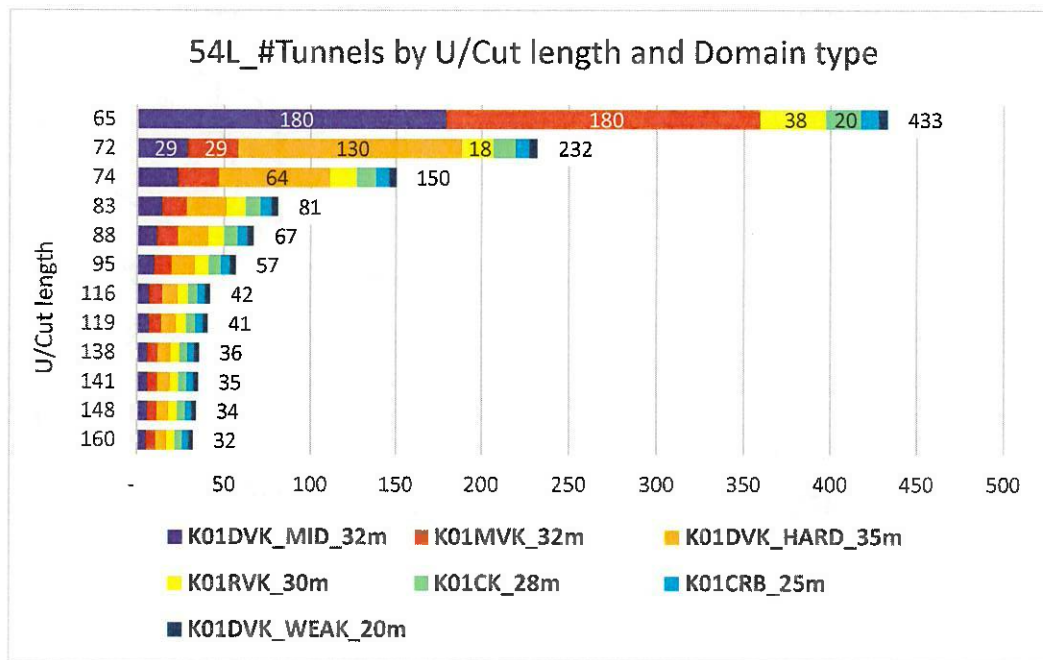


Figure 5-7. Number of tunnels by undercut length and domain type 54 level.

5.4 Recommended Mining Method

Caving Mining Methods were discussed in Chapter 2 in details and the application of the methods were also investigated. Venetia orebody tends to be strong, country rock is weak, and the weathering of kimberlite is expected as the mine develop to the lowest levels. Based on the applicability assessment only sub-level caving and drift and fill are applicable to Venetia orebody. Table 5.3 below illustrates the mining method assessment for K01 orebody. The distinction between block caving and sub level caving is the ore recovery with the latter shows good improvement in the recovery. High cost and low production rate for Drift and Fill, classify it as the least preferred mining method.

Table 5.1: Mining method assessment for K01.

Principle	Method	Safety	Consistency	Profitability	Risk	Consequence	Mitigation
Natural Support	Open Benching				Country rock failure Slope stability	Poor recovery Consistency Safety	Change method
	Open Stoping With pillars				Unstable pillars & Unstable rooms	Poor recovery Consistency Safety	Reduce excavation size - Change method
Artificial Support	Open Stoping or Post Pillar cut & fill				Unstable pillars & Unstable rooms	Poor recovery Consistency Safety	Reduce excavation size - Change method
	Cut & Fill Drift & fill				High cost Low production rate	Unprofitable Low cash flow	Good design Good management
Caving	Sub Level Caving				Poor recovery Mud rush	Cash flow Safety	Good design Good management
	Block Caving				Orebody will not cave Waste dilution	Air blast Poor recovery	Change method

Dyson (2004) explain (SWOT), Strengths, Weakness, Opportunities and Threats methodology as illustrated in Table 5.4. According to Dyson (2004) the use of SWOT analysis is to identify opportunities and threats in an environment by identifying the strengths and weaknesses. By identifying the factors, strategies are established which may eliminate weaknesses and build strength (Dyson, 2004). The concern of the exercise is to establish possibility of applying Caving Mining Method in K01 deposit. The analysis provided by SWOT matrix, suggested Venetia mine should utilize SLC as compared to C&F. For example, the production strength coupled by low operating cost sees SLC as the best fit for K01 deposit. Hence, C&F is recognized as unknown method to any DeBeers operation and it has high cost coupled by low production.

Table 5.2: Mining method SWOT analysis.

Method	Strengths	Weaknesses	Opportunities	Threats
Cut & Fill	•Non subsidence method •Low dilution	•Low production rate •Entry method – greater exposure to safety risk •Access through pit – LOM pit stability •Unknown to De Beers	•Potential to step up to cut & fill or stope & fill if ground conditions are better than expected	•Unable to achieve production targets •High cost method – unviable in a suppressed market •Significant safety & resource recovery risk •Pit slope failure
Sub Level Caving	•Non entry method •High production rate •Low operating cost •Flexible – cost & production •Known to De Beers	•Waste dilution •Mud rush/ flooding	•Dilution could be less than planned •Ability to increase level spacing reducing costs and improving production rate •Able to ramp up in a booming market •Low cost to survive a suppressed market	•Dilution could be worse than planned •Mudrush/ food risk management could fail

Yuksel (2012) explain PESTEL (Political, Economic, Socio-cultural, Technological, Environment and Legal) analysis as illustrated in Table 5.5. The adoption for PESTEL analysis for this project is to understand the general idea of environmental condition and legal impact into the business solution. Yuksel (2012) stated that the basic strategy for management is to involve the analysis of the current factors relevant to the environmental within which the company is operating. It is for this reason that an approach like PESTEL utilised for the mining method selected. Yuksel (2012) describes the macro environment consists of Socio-cultural, technological, ecological, political, economic, and legal factors.

The other factor included in the PESTEL (Table 5.5) is safety, as the industry strives to achieve zero harm. The Block cave and Sub level cave mining method require high skilled level personnel. The skill shortage and remote location of the Mine pose a significant challenge. The closest Mine is Palabora Copper Mine which is 400 km away. This part presents a challenge in selecting Mining Method in an area whereby skill requirements is critical. It is of this reason that a decision like this hamper the process of selecting a mining method. The technical coupled by economic factors exonerates C&F as a least mining method because of its nonexistence within DeBeers

Company and with high capital expenditure. SLC is regarded from this analysis as the best mining method for K01, because its background within the company, risks are well understood, and having an improved cash flow.

Table 5.3: PESTEL analysis for mining method.

Method	Safety	Political	Economic	Social	Technical	Environmental	Legal
Cut & Fill	Entry method		•Low capex •High opex •Low cashflow •Inflexible/ sensitive •complicated	High skill level	•Never been done •Unknown risk •Stability?	Non subsidence method	•Entry method. •Compliance with MHSA
Sub Level Caving	Mud rush		•High capex •Low opex •High cash flow •Flexible/ insensitive •uncomplicated	High skill level	•Done before •Risk is known •Dilution	subsidence method	•Non entry method (Minimum number of people working in production area. As the ring retreat , no one allowed behind the ring

5.5 Cut and Fill as a mining method for K01

As this method is technically feasible a business case has been considered based on the following parameters as shown in Table 5.6.

Table 5.4: Input parameters (assumed).

Scenario	Base Case	Parameter determination
Extraction methodology	Primary and secondary or <u>pillarless</u> drift & fill extraction	Stress modeling
Tunnel height - span (modeling to confirm)	5m Range: 4m – 10m	Stress modeling
Tunnel width (modeling to confirm)	5m Range: 4m – 10m	Stress modeling
Geological losses (un-mineable loss due to contacts, weathered/ weak kimberlite & in <u>stope</u> + regional pillars)	To be determined Dependant on pillar requirements	Stress modeling
Production levels in operation	2	Stress modeling
Production rate	1Mt/year	Arena simulation

Assumption were considered with limited time; a detailed study needs to be done to ensure all the parameters are included in the modelling. Following the PESTEL and SWOT analysis a detailed modelling shall be done to determine:

- Stress analysis on drift dimension;
- Back behaviour as we undercut the pipe;
- Analysis of primary/secondary vs pillar less mining sequence;
- The role and performance of regional barrier pillar;
- Fill design for stable working platform;
- Room and tunnel support;
- Production rate determination.

To create a working scenario for the method, consider two scenarios with the first being a primary secondary extraction drift scenario which has the advantage of maximising the available mining faces for the production cycles and second scenario is a pillarless mining.

Both scenarios assume mining 100m high blocks on two levels leaving a crown pillar between the pit bottom and first production section and a sill pillar between the two production sections. A flat back face configuration has been assumed.

Production rate for this method is determined by the drill rig capability and number of faces available. Each production section has been split into 3 production stopes with one drill rig each. Thus, these 6 production drill rigs, working contops (2 x 12hrs) shift arrangement can produce 155kt a year per drill rig. Tunnels will be cycled through a production (drilling, charging, loading, blasting and support) and fill cycle. Thus, a minimum of 10 ends are required a day per stope to maintain a continuous cycle.

Figure 5.8 and 5.9 illustrate the primary and secondary extraction using parallel drifting. From the analysis of the method with production constraints, the alternative is to include a third production level, but logistics now become a challenge. The method may require trucking solution over a vertical height of 200m to 250m to achieve a 1-1.5 Mt a year. In addition, a twin access declines are also required for men and material transport. Congestion becomes a significant risk. Current scenario depicts the following;

- 6 drills;
- 6 rock bolters;
- 3 LHD's (17 tons);
- 5 dump truck (60 tons);
- Number of LDV's.

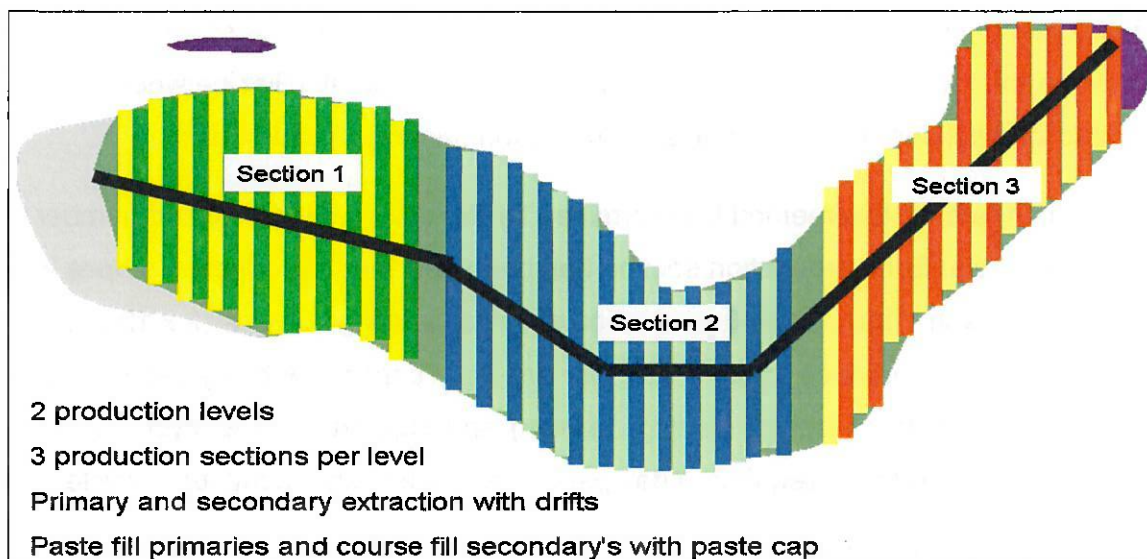


Figure 5-8: Primary and secondary extraction using parallel drifting.

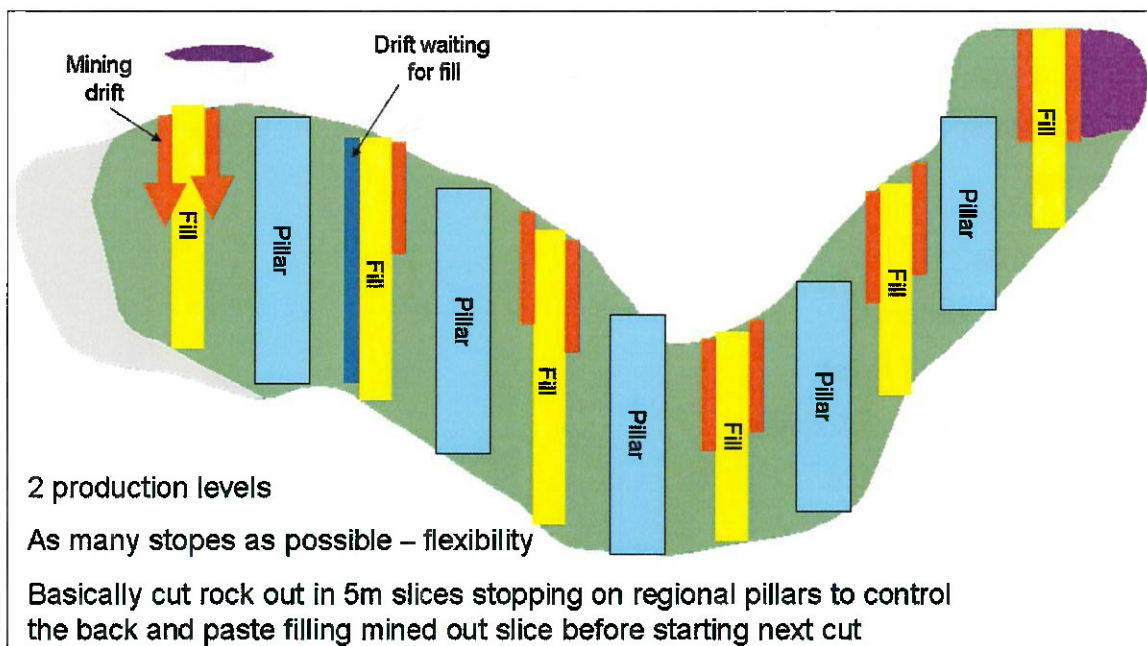


Figure 5-9: Pillarless extraction using rib pillars.

5.6 Sub-Level Caving as a Mining Method for K01

As this method is technically feasible, a business case has been considered based on the following parameters as shown in Table 5.7.

Table 5.5: Sub-Level parameters (assumed).

Parameter	Base Case	Parameter determination
Level spacing	20m Range: 15m – 25m	Review history, <u>Ekati</u> performance, trial at Finsch, blast simulation modeling
Tunnel spacing	14m	Stress modeling
Tunnel width x height	5m x 4.25m	Stress modeling
Geological losses (un-mineable loss due to contacts)	5%	Depletion modeling
Production drilling kimberlite recovery (due to waste dilution)	75%	Depletion modeling
Steady state load % of ring tons blasted	110% (40% waste)	Depletion modeling
Max ROM waste % treated (due to waste dilution)	35%	Depletion modeling
Pillar extraction %	50%	Depletion modeling
Loaded tons @ 110%	3.8Mt/year	Arena simulation

A detailed study needs to be done to ensure all parameters are considered for study. The detailed modelling shall be done to determine the following:

- Stress analysis on drift layout;
- Caving pillar behaviour as we undercut the pipe;
- Analysis of the mining sequence;
- Long hole ring blast design;
- Analysis of the mining sequence;
- Stress analysis on infrastructure;
- Waste failure and dilution ingress estimation;
- Tunnel support;
- Production rate determination.

The challenge with sub level caving is in selecting the correct layout geometry. Essentially, the closer the tunnels the better the draw, but if they are too close there could be pillar failure between the tunnels. Thus, compromise is often made between optimal draw spacing and optimal pillar and tunnel size. A further challenge is in setting the production rate. The general trend is that the higher the level spacing the bigger the ring tonnes and the higher the production rate.

Tunnel size and spacing is easily determined using stress modelling but ring design is based on industry practice and on mine experience. DeBeers has successfully developed undercuts in 15m spacing tunnel and 15m ring heights on block caves. As the rings are in abutments, they would have been under considerably more stress than SLC rings which are always in a distressed environment. The key challenge for SLC is to drill accurate holes and charge to the correct explosive density. The next challenge is hole closure due to stress, damage from previous blasting, weathering of kimberlite, and water. The general rule is to re-drill collapsed holes, which is a problematic as the muck pile is often in the way and it is impossible to know if there are any explosives left from the previous blasted rings.

For the exercise, a base case of 20m high level spacing, 5m wide by 4.5m high tunnels on 14m centres layout is used with a tunnel retreat rate of 10m (5 rings) per month. This delivers a production rate of 3.2 Mt a year assuming 2 levels in production retreating a flat face with a 110% loading factor. Figure 5.10 illustrates two levels in production with staggered tunnels. Figure 5.11 illustrates ring comparison of 15, 20 and 25 level spacing.

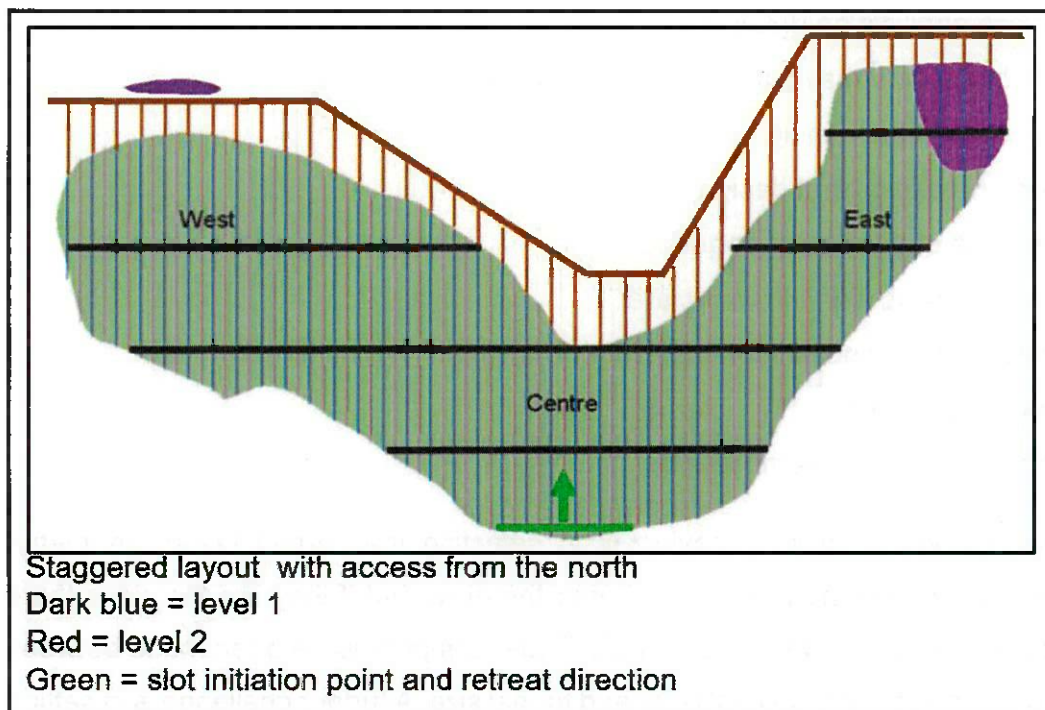


Figure 5-10: Typical level layout and retreat face shape

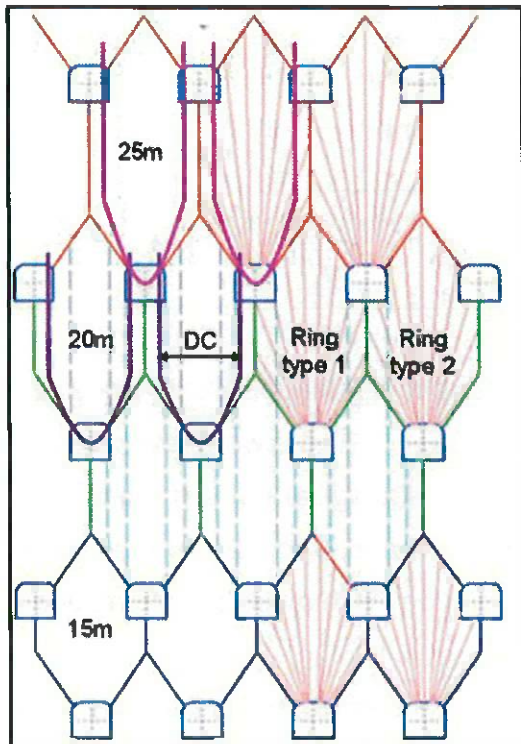


Figure 5-11: Ring comparison of a 15, 20 and 25m level spacing.

With ring comparison, there are few factors which needs to be taken for consideration while modelling the design:

- Tunnels spacing and size determined by geotechnical modelling which is a function of rock strength and stress;
- Must ensure a stable tunnel but want tunnels as close as possible to optimise recovery;
- Level spacing function of drilling capability;
- Require good fragmentation to maximise recovery and minimise production delays;
- Challenge to minimise holes losses from blasting;
- Need to manage high swell characteristics, draw over the apex and varying rock condition;
- From the ring comparison, below is the comparisons of ring metres drilled and long hole which can be drilled as per ring.

	<u>25m</u>	<u>20m</u>	<u>15m</u>
Ring metres	225m	197m	128m
Longest hole	30m	25m	20m

5.7 Conclusion

The objective of the project has been achieved by updating the GEMS lithology and geotechnical risk block model based on the data available. All data were reviewed and validated as correctly as possible. The country rock lithological domain model was updated, and additional geotechnical domains were created. The country rock solids are considered the best possible interpretation of geology with available information.

Additional effort went into analysis and definition of kimberlite domains that may have a significant impact on the selection of mining method, such as weathered DVK domains and contact zones. Data analysis was done using RMR and MRMR classification systems.

The results of the geotechnical analysis were used in the calculation of the MRMR system. The hydraulic radius for all kimberlite domains were calculated with the least being 20m and maximum 35m. The analysis as expected shows ADVK_Hard with maximum hydraulic radius. This indicates stable ground condition which require a pre-drilling for caving to occur. The SWOT and PESTEL analysis were used for C & F and SLC. The results were apparent that, there is a lack of knowledge about C&F locally as compared to SLC. High Capex and low production rate for C&F exonerate it as a preferred method while SLC is regarded as the best method because of high level of information repository within DeBeers mines.

The long-term mining method solution for K01 is a sub level cave since this is the only method that can accommodate the high variability of the orebody contacts and the expected waste dilution from the slope failure. The study was pursued to investigate the mining method for K01 kimberlite orebody deposit. The research found that there is significant irregular shape on the east of K01 deposit. The irregular shape coupled

by infilling of other facies may hinder caving to occur. The expected hydraulic radius for caving to happen is 35m with an undercut length of 160m, and this pose a threat on the amount of area required to initiate caving. The other challenge for block caving if applied, is a potential for wedges to be formed on the edge of the kimberlite and this will create fall of ground which is a risk for the underground workings.

5.8 Limitation of The Study

The main limitation for this study is the highly confidentiality of the information and red tape which is applied to company information. The stacking of core and interpretation of the results was a challenge because of skill competence of the assay personnel. Some core samples have been weathered because of the exposure to the sun and this has also influenced the results in a negative way. The study focused on K01 Kimberlite orebody which is not a true representation for K02 kimberlite orebody. The study focused on the geotechnical parameters and leave the financial parameters to determine best mining method. However, if all parameters (financial, K02 deposit) needs to be applied, the study would need to be re-run again. This study is only applicable to Venetia K01 deposit.

5.9 Recommendations

This report was based on a single orebody deposit with only one specific parameter of geotechnical consideration applied to it. The results obtained in this report need to be verified with other geotechnical modelling design, leapfrog and scheduling software (Mine24D) to validate the hypothesis. A more accurate objective function will be realized if detailed modelling applied determine factors discussed in 5.4 and 5.5. The recommended mining method should be implemented, and other supporting infrastructure should be modelled whether they support such method i.e. ventilation, support requirements, access to rim tunnel, and equipment and production target as envisaged by the mine.

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7 APPENDICES

7.1 Appendix A. Gems Modelling

Table A.1: Average values for each domain used in the geotechnical block model.

Domains	RMR		Q	UCS	M _i	Block Code
	Average	StdDev	Average	Average		
Dolerite Intrusions	57	16	18.1	267	19.9	240
Gneiss Package	65	15	18.4	132	13	210
BS rock type only	65	15	16.3	97	13.7	-
Lezel-Tina Shear Zone	54	17	4.8	122	8 ⁺	250
Marble Package	60	17	10.4	125	17	230
Metasediment Package	57	17	9.8	140	8.5	220
Dolerite Porphyry	69	16	7	267 ⁺	19.9 ⁺	260
K01CK	62	13	25.7	127	7.8	101-105
K01CRB	60	16	13.2	177 ⁺	10.6	113-114
K01DVK_HARD	74	14	30.2	134	14.4	1121, 1151
K01DVK_MID	65	16	5.1	75	12.9	1122, 1152
K01DVK_WEAK	48	21	2.1	47	9.9	1123, 1153
K01MVK	65	15	6.4	68	9.8	106, 107, 109
K01RVK	63	13	3.5	29	9.8 ⁺	110-111
K02CRB	66	13	66.9	177	10.6	124-125
K02MVK	78	11	8.2	46	9.8 ⁺	121
K02NW	61	13	10.3	46 ⁺	9.8 ⁺	123
K02VKB	73	14	6.8	47	9.8 ⁺	122
K02VKBL	73	14	9.5	89	9.8 ⁺	126
K03	71	13	43.1	186	9.8 ⁺	130
K04	64	19	6.3	68 ⁺	9.8 ⁺	140
K05	67	15	6.3 ⁺	68 ⁺	9.9 ⁺	150
K06	54 ⁺	17 ⁺	4.8 ⁺	68 ⁺	9.9 ⁺	160
K08CRB	59	14	17.5	177 ⁺	10.6 ⁺	181
K08MK	63	10	7.9	127 ⁺	9.8 ⁺	182
K08VKB	68	14	26.3	47 ⁺	10.6 ⁺	184
K08VKBB	60	14	17.1	78	10.6 ⁺	185

Table A.2. Laplacian gridding parameters used to create clipping surfaces.

Surface	X Start	Y Start	Row Num	Col Num	Row Size	Col Size	View Plane
SILL1_Base	-1350	-800	64	41	10	10	SILL1
Lezel_East	-2060	-910	89	183	20	20	SZ_East
Tina_West	-2060	-910	89	183	20	20	SZ_East
DOLR1_WEST	-760	-1650	152	120	20	20	DOLR1
DOLR2	-1820	-1500	163	183	20	20	DOLR2
DOLR3	-1460	-1410	144	152	20	20	DOLR3
DOLR1_EAST	-1080	-1160	144	174	20	20	DOLR1
POR	-820	-680	107	154	20	20	POR
D1_DYKE3	-490	-380	74	96	10	10	DYKE3
D1_DYKE2	-410	-375	68	83	10	10	DYKE2
SLIMESSZ	-2020	-900	83	145	20	20	SLIMESSZ
FZ1	-1510	-870	87	152	20	20	FZ1
PROLATEZ	-2020	-900	83	145	20	20	SLIMESSZ

7.2 Appendix B. Undercut Design

The MRMR requires few assumptions to be made for adjustment for the calculations:

- Weathering: The ground is still intact not exposed and a minimal weathering is expected for the kimberlite, value assumed is 90%;
- Orientation: Kimberlite doesn't have jointing which can determine a favorable or unfavorable position of mining, and the orientation considered to be 100%;
- Stress: The UCS for the K01 domains is very high with some domains having a UCS of 177, this present a good ground condition which stable and the stress value assumed is 90%;
- Blasting: Good drilling, charging and blasting practices is assumed for the development of K01 kimberlite and assumption is made for blasting adjustment of 90%.

MRMR = RMR x Weathering adjustment X Blasting adjustments X

Orientation X Stress

1) = K01CK (RMR) X Weathering adjustment X Blasting adjustments X

Orientation X Stress

$$= 62 \times 0.9 \times 0.9 \times 0.9$$

$$= 45$$

2) = K01CRB (RMR) X Weathering adjustment X Blasting adjustments
X Orientation X Stress

$$= 60 \times 0.9 \times 0.9 \times 0.9$$

$$= 44$$

3) = K01DVK_HARD (RMR) X Weathering adjustment X Blasting
adjustments X Orientation X Stress

$$= 74 \times 0.9 \times 0.9 \times 0.9$$

$$= 54$$

4) = K01DVK_MID (RMR) X Weathering adjustment X Blasting
adjustments X Orientation X Stress

$$= 65 \times 0.9 \times 0.9 \times 0.9$$

$$= 47$$

5) = K01DVK_WEAK (RMR) X Weathering adjustment X Blasting
adjustments X Orientation X Stress

$$= 48 \times 0.9 \times 0.9 \times 0.9$$

$$= 35$$

6) = K01MVK (RMR) X Weathering adjustment X Blasting adjustments X Orientation X Stress

$$= 65 \times 0.9 \times 0.9 \times 0.9$$

$$= 47$$

7) = K01RVK (RMR) X Weathering adjustment X Blasting adjustments X Orientation X Stress

$$= 63 \times 0.9 \times 0.9 \times 0.9$$

$$= 46$$

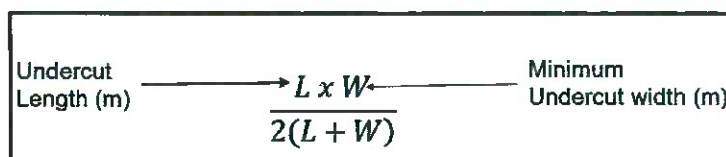
Table B.1. MRMR value for each Kimberlite domain.

Domain (K01)	CK	CRB	DVK_HARD	DVK_MID	DVK_WEAK	MVK	RVK
MRMR	45	44	54	47	35	47	46
Hydraulic radius	28	25	35	32	20	32	30

Calculating the minimum undercut for 46 and 54 level respectively

$$\text{Hydraulic radius} = \frac{\text{Area}}{\text{Perimeter}}$$

$$= \frac{L \times W}{2(L+W)}$$



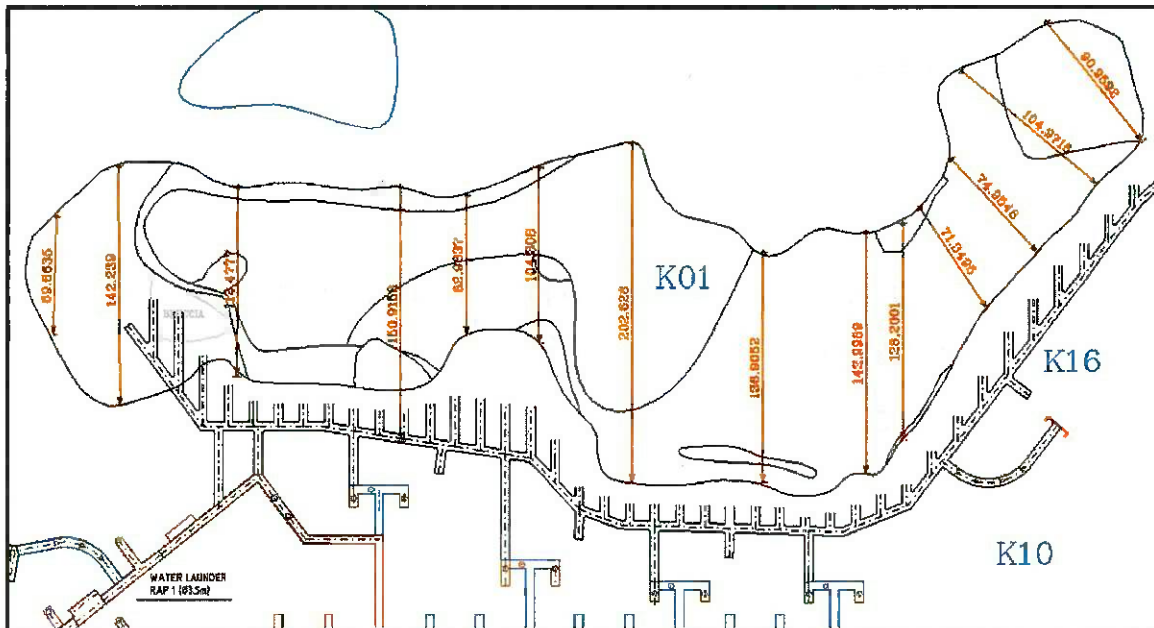


Figure B.1: Level 46 undercut design.

Calculating minimum undercut width for 46 level;

$$\text{Hydraulic radius} = \frac{\text{Area}}{\text{Perimeter}}$$

$$= \frac{L \times W}{2(L+W)}$$

L is the undercut distance = 69.86

Hydraulic radius = 20

$$\text{Width} = \frac{2LHR}{L-2HR}$$

$$= \frac{2 \times 69.88 \times 20}{69.88 - 2 \times 20}$$

$$= \frac{2795.2}{29.86}$$

Width= 93.6 m

Using same equation for Kimberlite domain in 46 level;

L= 69.88

Domain	MRMR	Hydraulic radius (HR)	Undercut length (L)	Minimum undercut width (W) m
KO1CK	45	28	69.88	281.94
K01CRB	44	25	69.88	175.75
K01DVK_HARD	54	35	69.88	-40763.33
K01DVK_MID	47	32	69.88	760.60
K01DVK_WEAK	35	20	69.88	93.55
K01MVK	47	32	69.88	760.60
K01RVK	46	30	69.88	424.37

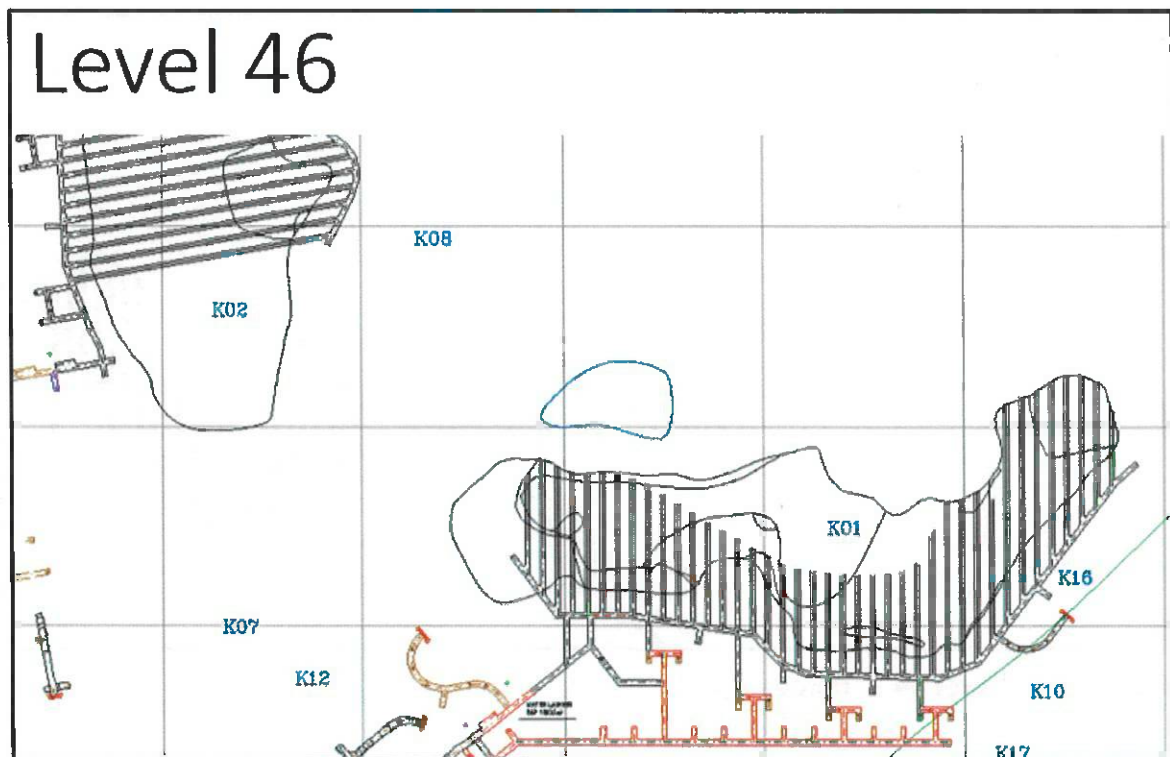


Figure B.2. Level 46 undercut layout.

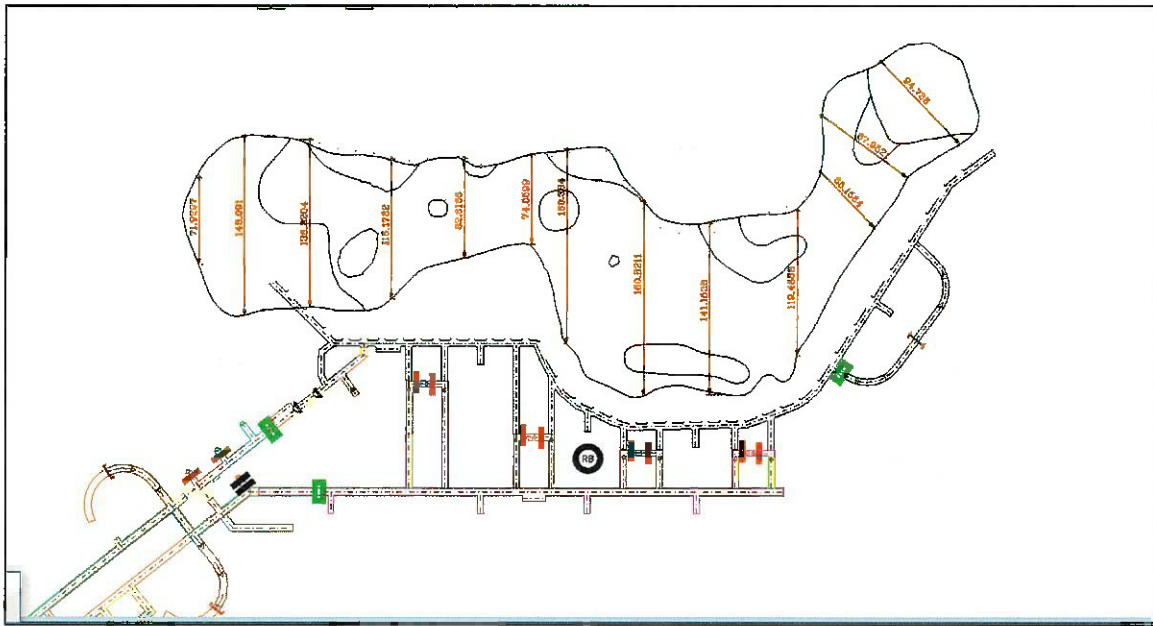


Figure B.3. Level 54 undercut design measurement.

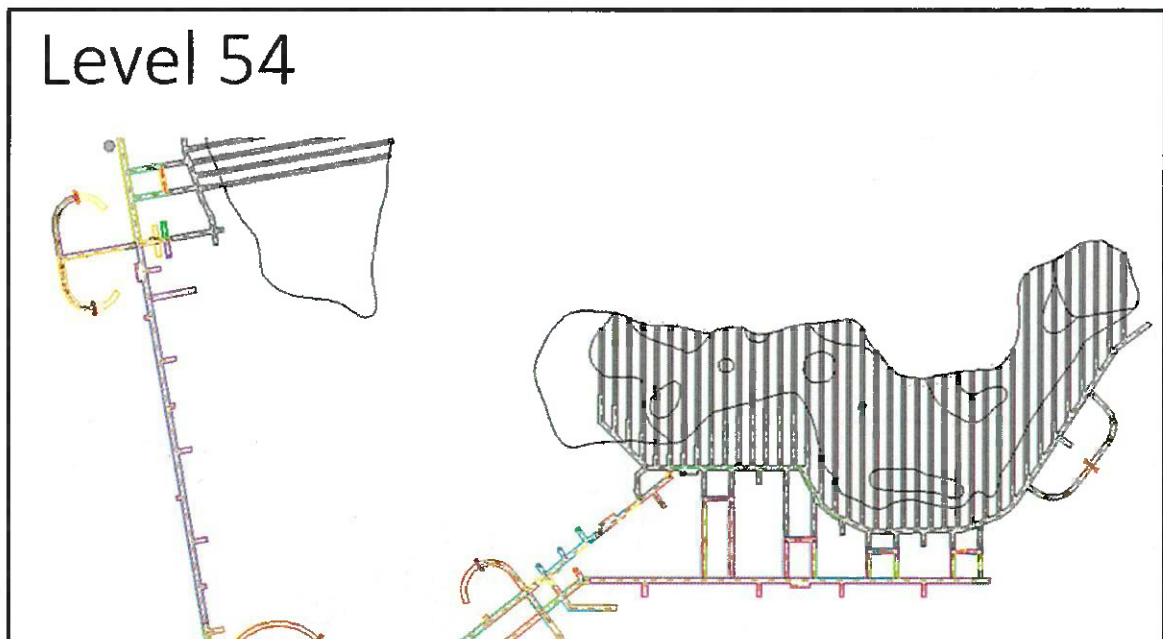


Figure B.4. Level 54 undercut layout.

Table B.2. Level_46 Kimberlite facies calculation for undercut minimum width.

	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	69.88	28	39.13.28	13.83	281.94	5.00	15.00	14		K01OK	45	142.24	28	79.53.44	85.24	82.50	5.00	15.00	5
	K01CRB	44	69.88	25	34.94	19.88	175.75	5.00	15.00	9		K01CRB	44	142.24	25	71.12	82.24	77.10	5.00	15.00	4
	K01DVK_HARD	54	69.88	35	48.16	-0.12	-407.63.30	5.00	15.00	-2038		K01DVK_HARD	54	142.24	35	89.68	72.24	157.83	5.00	15.00	7
	K01DVK_MID	47	69.88	32	44.72.32	5.88	760.60	5.00	15.00	38		K01DVK_MID	47	142.24	32	81.03.30	78.24	116.35	5.00	15.00	6
	K01DVK_WEAK	35	69.88	20	27.93.2	29.88	88.55	5.00	15.00	5		K01DVK_WEAK	35	142.24	20	56.89.6	102.24	55.85	5.00	15.00	3
	K01MVK	47	69.88	32	44.72.32	5.88	760.60	5.00	15.00	38		K01MVK	47	142.24	32	81.03.36	78.24	116.35	5.00	15.00	6
	K01RVK	45	69.88	30	42.92.8	9.88	424.37	5.00	15.00	21		K01RVK	45	142.24	30	83.41.4	82.24	103.77	5.00	15.00	5
	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	112.48	28	62.88.88	56.48	111.52	5.00	15.00	6		K01OK	45	150.92	28	84.51.52	94.92	89.04	5.00	15.00	4
	K01CRB	44	112.48	25	56.24	62.48	90.01	5.00	15.00	5		K01CRB	44	150.92	25	75.46	100.92	74.77	5.00	15.00	4
	K01DVK_HARD	54	112.48	35	78.73.6	42.48	149.35	5.00	15.00	9		K01DVK_HARD	54	150.92	35	105.64.4	80.92	130.55	5.00	15.00	7
	K01DVK_MID	47	112.48	32	71.98.72	48.48	148.49	5.00	15.00	7		K01DVK_MID	47	150.92	32	95.88.88	85.92	111.12	5.00	15.00	6
	K01DVK_WEAK	35	112.48	20	44.99.2	72.48	62.08	5.00	15.00	3		K01DVK_WEAK	35	150.92	20	60.35.8	110.92	54.42	5.00	15.00	3
	K01MVK	47	112.48	32	71.98.72	48.48	148.49	5.00	15.00	7		K01MVK	47	150.92	32	95.88.88	85.92	111.12	5.00	15.00	6
	K01RVK	45	112.48	30	67.88.8	52.48	128.00	5.00	15.00	6		K01RVK	45	150.92	30	90.59.2	90.92	99.60	5.00	15.00	5
	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	82.1	28	45.7.6	26.1	175.15326	5.00	15.00	9		K01OK	45	104.31	28	58.11.36	48.31	120.9141	5.00	15.00	6
	K01CRB	44	82.1	25	41.05	32.1	127.88162	5.00	15.00	6		K01CRB	44	104.31	25	52.15.5	54.31	88.03204	5.00	15.00	5
	K01DVK_HARD	54	82.1	35	57.47	12.1	474.25868	5.00	15.00	24		K01DVK_HARD	54	104.31	35	73.01.7	34.31	212.8155	5.00	15.00	11
	K01DVK_MID	47	82.1	32	52.54.4	18.1	280.29634	5.00	15.00	15		K01DVK_MID	47	104.31	32	66.75.84	40.31	165.6125	5.00	15.00	8
	K01DVK_WEAK	35	82.1	20	32.84	42.1	78.004758	5.00	15.00	4		K01DVK_WEAK	35	104.31	20	41.72.4	64.31	64.87949	5.00	15.00	3
	K01MVK	47	82.1	32	52.54.4	18.1	280.29634	5.00	15.00	15		K01MVK	47	104.31	32	66.75.84	40.31	165.6125	5.00	15.00	8
	K01RVK	45	82.1	30	49.26	22.1	222.85567	5.00	15.00	11		K01RVK	45	104.31	30	62.58.6	44.31	141.2458	5.00	15.00	7
	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	202.63	28	113.47.28	146.63	77.387165	5.00	15.00	4		K01OK	45	138.91	28	77.78.96	82.91	93.82415	5.00	15.00	5
	K01CRB	44	202.63	25	103.15.5	152.63	66.37948	5.00	15.00	3		K01CRB	44	138.91	25	69.40.5	88.91	78.11832	5.00	15.00	4
	K01DVK_HARD	54	202.63	35	141.84.1	152.63	106.94486	5.00	15.00	5		K01DVK_HARD	54	138.91	35	97.23.7	68.91	141.1072	5.00	15.00	7
	K01DVK_MID	47	202.63	32	12.868.32	138.63	93.546278	5.00	15.00	5		K01DVK_MID	47	138.91	32	86.90.24	74.91	118.6789	5.00	15.00	6
	K01DVK_WEAK	35	202.63	20	81.05.2	162.63	49.838283	5.00	15.00	2		K01DVK_WEAK	35	138.91	20	35.56.4	98.91	56.17632	5.00	15.00	3
	K01MVK	47	202.63	32	12.868.32	138.63	93.546278	5.00	15.00	5		K01MVK	47	138.91	32	86.90.24	74.91	118.6789	5.00	15.00	6
	K01RVK	45	202.63	30	12.157.8	142.63	85.240132	5.00	15.00	4		K01RVK	45	138.91	30	83.34.6	78.91	103.6216	5.00	15.00	5
	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	142.1	28	79.57.6	84.1	92.422764	5.00	15.00	6		K01OK	45	128.2	28	71.79.2	72.2	99.4345	5.00	15.00	5
	K01CRB	44	142.1	25	71.05	92.1	77.144408	5.00	15.00	4		K01CRB	44	128.2	25	64.10	78.2	81.96931	5.00	15.00	4
	K01DVK_HARD	54	142.1	35	99.47	72.1	137.96117	5.00	15.00	7		K01DVK_HARD	54	128.2	35	88.74	58.2	154.1924	5.00	15.00	8
	K01DVK_MID	47	142.1	32	90.94.4	78.1	116.44538	5.00	15.00	6		K01DVK_MID	47	128.2	32	82.04.8	64.2	127.8005	5.00	15.00	6
	K01DVK_WEAK	35	142.1	20	56.84	102.1	55.670911	5.00	15.00	3		K01DVK_WEAK	35	128.2	20	31.28	88.2	54.14039	5.00	15.00	3
	K01MVK	47	142.1	32	90.94.4	78.1	116.44538	5.00	15.00	6		K01MVK	47	128.2	32	82.04.8	64.2	127.8005	5.00	15.00	6
	K01RVK	45	142.1	30	85.26	82.1	105.84895	5.00	15.00	5		K01RVK	45	128.2	30	76.92	68.2	112.7838	5.00	15.00	5
	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	74.95	28	41.97.2	18.95	221.48813	5.00	15.00	11		K01OK	45	104.97	28	58.78.32	48.97	120.0392	5.00	15.00	6
	K01CRB	44	74.95	25	37.47.5	24.95	150.2004	5.00	15.00	8		K01CRB	44	104.97	25	52.48.5	54.97	95.47935	5.00	15.00	5
	K01DVK_HARD	54	74.95	35	52.65.5	4.95	105.9.899	5.00	15.00	23		K01DVK_HARD	54	104.97	35	73.47.9	34.97	210.1201	5.00	15.00	11
	K01DVK_MID	47	74.95	32	47.96.8	10.95	438.06393	5.00	15.00	22		K01DVK_MID	47	104.97	32	67.18.08	40.97	163.9756	5.00	15.00	8
	K01DVK_WEAK	35	74.95	20	29.98	34.95	83.779685	5.00	15.00	4		K01DVK_WEAK	35	104.97	20	41.98.8	64.97	64.62675	5.00	15.00	3
	K01MVK	47	74.95	32	47.96.8	10.95	438.06393	5.00	15.00	22		K01MVK	47	104.97	32	67.18.08	40.97	163.9756	5.00	15.00	8
	K01RVK	45	74.95	30	44.97	14.95	300.80288	5.00	15.00	15		K01RVK	45	104.97	30	62.98.2	48.97	140.0534	5.00	15.00	7
	Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving		Domain	MRMR	Undercut length (L)	Hydraulic radius (HF)	2-LR	L-2-LR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
	K01OK	45	90.95	28	50.32.2	34.95	145.72818	5.00	15.00	7		K01OK	45	90.95	28	50.32.2	34.95	145.72818	5.00	15.00	7
	K01CRB	44	90.95	25	45.47.5	40.95	111.05005	5.00	15.00	6		K01CRB	44	90.95	25	45.47.5	40.95	111.05005	5.00	15.00	6
	K01DVK_HARD	54	90.95	35	63.65.5	20.95	308.88021	5.00	15.00	15		K01DVK_HARD	54	90.95	35	63.65.5	20.95	308.88021	5.00	15.00	15
	K01DVK_MID	47	90.95	32	58.01.8	26.95	215.08516	5.00	15.00	11		K01DVK_MID	47	90.95	32	58.01.8	26.95	215.08516	5.00	15.00	11
	K01DVK_WEAK	35	90.95	20	34.38	50.95	71.403337	5.00	15.00	4		K01DVK_WEAK	35	90.95	20	34.38	50.95	71.403337	5.00	15.00	4
	K01MVK	47	90.95	32	58.01.8	26.95	215.08516	5.00	15.00	11		K01MVK	47	90.95	32	58.01.8	26.95	215.08516	5.00	15.00	11
	K01RVK	45	90.95	30	54.57	30.95	176.31654	5.00	15.00	9		K01RVK	45	90.95	30	54.57	30.95	176.31654	5.00	15.00	9

Table B.3. Level_54 kimberlite facies calculation for minimum undercut width.

Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving	Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
K01CK	45	71.93	28	4028.08	15.93	252.86	5.00	15.00	13	K01CK	45	148.09	28	8293.04	92.09	90.05	5.00	15.00	5
K01CRB	44	71.93	25	3596.5	21.93	164.00	5.00	15.00	8	K01CRB	44	148.09	25	7404.5	95.09	75.49	5.00	15.00	4
K01DVK_HARD	54	71.93	35	5036.1	1.93	2608.86	5.00	15.00	130	K01DVK_HARD	54	148.09	35	10366.3	78.09	132.75	5.00	15.00	7
K01DVK_MID	47	71.93	32	4603.52	7.93	580.52	5.00	15.00	29	K01DVK_MID	47	148.09	32	8477.76	84.09	112.71	5.00	15.00	6
K01DVK_WEAK	35	71.93	20	2877.2	31.93	90.11	5.00	15.00	5	K01DVK_WEAK	35	148.09	20	5623.8	108.09	54.80	5.00	15.00	3
K01MVK	47	71.93	32	4903.52	7.93	590.52	5.00	15.00	29	K01MVK	47	148.09	32	9477.76	84.09	112.71	5.00	15.00	8
K01RVK	46	71.93	30	4315.9	11.93	391.76	5.00	15.00	18	K01RVK	46	148.09	30	8895.4	88.09	100.87	5.00	15.00	5
Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving	Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
K01CK	45	138.32	28	7745.92	82.32	94.10	5.00	15.00	5	K01CK	45	116.18	28	8506.08	80.18	108.11	5.00	15.00	5
K01CRB	44	138.32	25	6916	88.32	78.31	5.00	15.00	4	K01CRB	44	116.18	25	5809	68.18	87.78	5.00	15.00	4
K01DVK_HARD	54	138.32	35	9882.4	88.32	141.72	5.00	15.00	7	K01DVK_HARD	54	116.18	35	8132.6	48.18	178.11	5.00	15.00	9
K01DVK_MID	47	138.32	32	8952.48	74.32	119.11	5.00	15.00	6	K01DVK_MID	47	116.18	32	7435.52	52.18	142.50	5.00	15.00	7
K01DVK_WEAK	35	138.32	20	5532.8	98.32	66.27	5.00	15.00	3	K01DVK_WEAK	35	116.18	20	4647.2	78.18	81.00	5.00	15.00	3
K01MVK	47	138.32	32	8952.48	74.32	119.11	5.00	15.00	6	K01MVK	47	116.18	32	7435.52	52.18	142.50	5.00	15.00	7
K01RVK	46	138.32	30	8299.2	78.32	105.97	5.00	15.00	5	K01RVK	46	116.18	30	6970.8	56.18	124.08	5.00	15.00	5
Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving	Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
K01CK	45	82.62	28	4626.72	26.62	173.81	5.00	15.00	9	K01CK	45	74.06	28	4147.36	18.06	229.64	5.00	15.00	11
K01CRB	44	82.62	25	4131	32.62	126.64	5.00	15.00	6	K01CRB	44	74.06	25	3703	24.06	153.91	5.00	15.00	8
K01DVK_HARD	54	82.62	35	5783.4	12.62	458.27	5.00	15.00	23	K01DVK_HARD	54	74.06	35	5184.2	4.06	1276.95	5.00	15.00	64
K01DVK_MID	47	82.62	32	5287.68	18.62	283.98	5.00	15.00	14	K01DVK_MID	47	74.06	32	4739.84	10.06	471.16	5.00	15.00	24
K01DVK_WEAK	35	82.62	20	3304.8	42.62	77.54	5.00	15.00	4	K01DVK_WEAK	35	74.06	20	2862.4	34.06	88.98	5.00	15.00	4
K01MVK	47	82.62	32	5287.68	18.62	283.98	5.00	15.00	14	K01MVK	47	74.06	32	4739.84	10.06	471.16	5.00	15.00	24
K01RVK	46	82.62	30	4657.2	22.62	219.15	5.00	15.00	11	K01RVK	46	74.06	30	4443.8	14.06	316.05	5.00	15.00	15
Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving	Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
K01CK	45	160.38	28	8981.28	104.38	86.04	5.00	15.00	4	K01CK	45	141.16	28	7804.96	85.16	92.82	5.00	15.00	5
K01CRB	44	160.38	25	8019	110.38	72.65	5.00	15.00	4	K01CRB	44	141.16	25	7059	91.16	77.42	5.00	15.00	4
K01DVK_HARD	54	160.38	35	11226.6	90.38	124.22	5.00	15.00	6	K01DVK_HARD	54	141.16	35	8881.2	71.16	138.88	5.00	15.00	7
K01DVK_MID	47	160.38	32	10264.32	98.38	106.50	5.00	15.00	5	K01DVK_MID	47	141.16	32	9034.24	77.16	117.08	5.00	15.00	6
K01DVK_WEAK	35	160.38	20	8415.2	120.38	53.29	5.00	15.00	3	K01DVK_WEAK	35	141.16	20	5848.4	101.16	55.82	5.00	15.00	3
K01MVK	47	160.38	32	10264.32	98.38	106.50	5.00	15.00	5	K01MVK	47	141.16	32	9034.24	77.16	117.08	5.00	15.00	6
K01RVK	46	160.38	30	9622.8	100.38	95.85	5.00	15.00	5	K01RVK	46	141.16	30	8489.6	81.16	104.36	5.00	15.00	5
Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving	Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
K01CK	45	119.46	28	6889.76	63.46	105.42	5.00	15.00	5	K01CK	45	65.16	28	3648.96	9.16	398.36	5.00	15.00	20
K01CRB	44	119.46	25	5973	69.46	85.99	5.00	15.00	4	K01CRB	44	65.16	25	3258	15.16	214.91	5.00	15.00	11
K01DVK_HARD	54	119.46	35	8362.2	49.46	169.07	5.00	15.00	8	K01DVK_HARD	54	65.16	35	4581.2	-4.84	942.40	5.00	15.00	-47
K01DVK_MID	47	119.46	32	7645.44	55.46	137.86	5.00	15.00	7	K01DVK_MID	47	65.16	32	4170.24	1.16	3595.03	5.00	15.00	180
K01DVK_WEAK	35	119.46	20	4778.4	79.46	60.14	5.00	15.00	3	K01DVK_WEAK	35	65.16	20	2608.4	25.16	103.59	5.00	15.00	5
K01MVK	47	119.46	32	7645.44	55.46	137.86	5.00	15.00	7	K01MVK	47	65.16	32	4170.24	1.16	3595.03	5.00	15.00	180
K01RVK	46	119.46	30	7167.6	59.46	120.54	5.00	15.00	6	K01RVK	46	65.16	30	3909.8	5.16	757.67	5.00	15.00	38
Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving	Domain	MRMR	Undercut length (L)	Hydraulic radius (HR)	2LHR	L-2xHR	Min. undercut Width (W)	Tunnel width	Pillar width	No. tunnels to open for Caving
K01CK	45	87.95	28	4925.2	31.95	154.15	5.00	15.00	8	K01CK	45	94.8	28	5308.8	38.8	136.82	5.00	15.00	7
K01CRB	44	87.95	25	4397.5	37.95	115.88	5.00	15.00	6	K01CRB	44	94.8	25	4740	44.8	105.80	5.00	15.00	5
K01DVK_HARD	54	87.95	35	6156.5	17.95	342.98	5.00	15.00	17	K01DVK_HARD	54	94.8	35	6636	24.8	287.58	5.00	15.00	13
K01DVK_MID	47	87.95	32	5828.8	23.95	235.02	5.00	15.00	12	K01DVK_MID	47	94.8	32	6097.2	30.8	198.99	5.00	15.00	10
K01DVK_WEAK	35	87.95	20	3518	47.95	73.37	5.00	15.00	4	K01DVK_WEAK	35	94.8	20	3792	54.8	69.20	5.00	15.00	3
K01MVK	47	87.95	32	5828.8	23.95	235.02	5.00	15.00	12	K01MVK	47	94.8	32	6097.2	30.8	198.99	5.00	15.00	10
K01RVK	46	87.95	30	5277	27.95	188.80	5.00	15.00	9	K01RVK	46	94.8	30	5688	34.8	163.45	5.00	15.00	8

Table B.4. Numerical data for level 46.

Sum of No: tunnels to open for Caving		Column Labels																									
		K01DKV_K_HARD		K01DKV_K_HARD Total		K01DMVK		K01DMVK Total		K01DKV_K_MID		K01DKV_K_MID Total		K01DKRV		K01DKRV Total		K01DKCB		K01DKCB Total		K01DKV_K_WEAK		K01DKV_K_WEAK Total		Grand Total	
Row Labels		35		22		32		30		30		30		30		25		20		20		20		20		20	
202.63		5.347244213		5.347244213	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	4.677313713	
150.92		6.527681661		6.527681661	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	5.5561896	
142.74		6.891472868		6.891472868	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	5.817586912	
138.51		6.898058252		6.898058252	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	5.822729129	
124.51		7.055362466		7.055362466	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	5.933947404	
128.7		7.706259129		7.706259129	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	6.390031153	
112.48		9.267419362		9.267419362	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	7.744224442	
104.57		10.50600515		10.50600515	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	8.198779595	
104.31		10.64077518		10.64077518	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	8.280625155	
90.25		15.19451074		15.19451074	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	10.79925788	
82.91		23.74979338		23.74979338	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	14.51491713	
74.95		52.99494949		52.99494949	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	21.90319635	
69.88		38.02991397		38.02991397	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	18.02899197	
Grand Total		162.7810356		162.7810356	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	143.3484784	

Table B.6. Graphical representation of domains versus undercut length/Hydraulic radius.

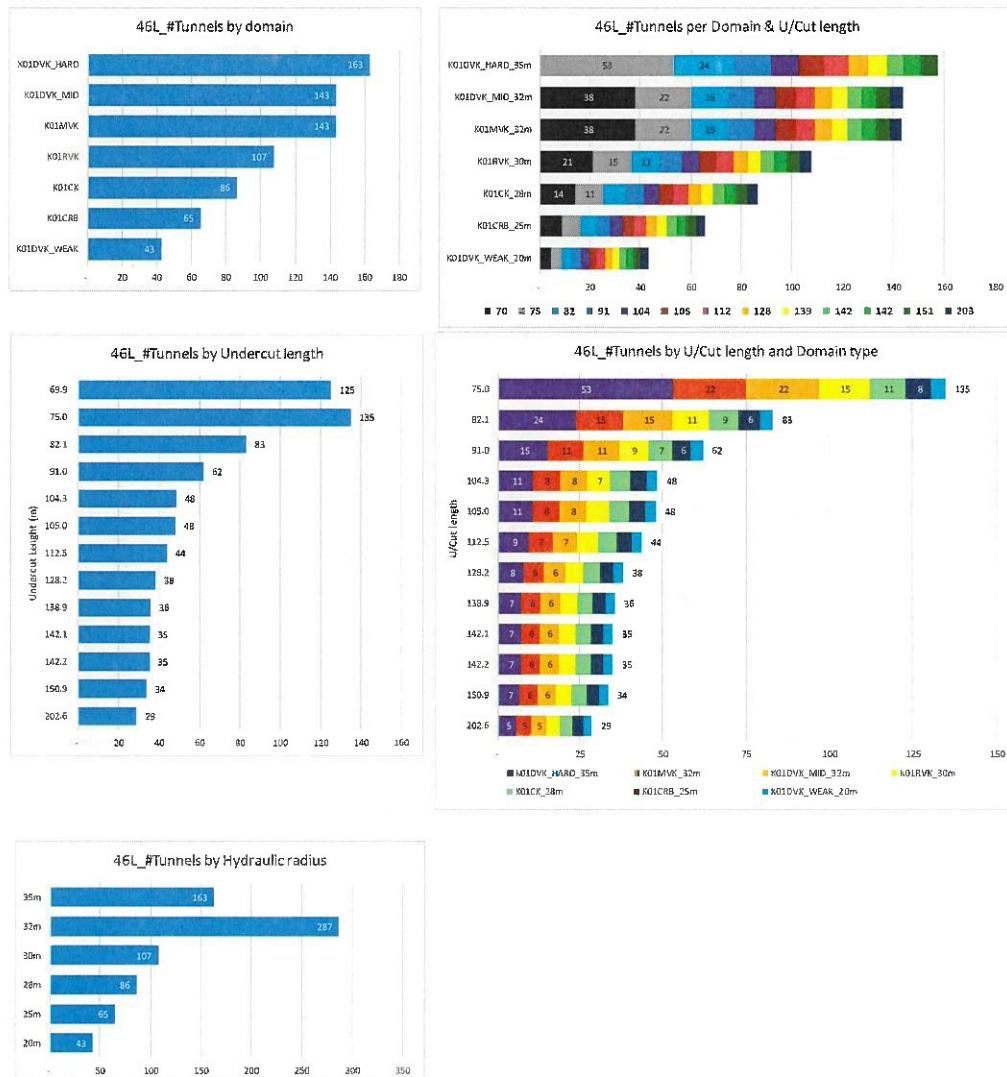
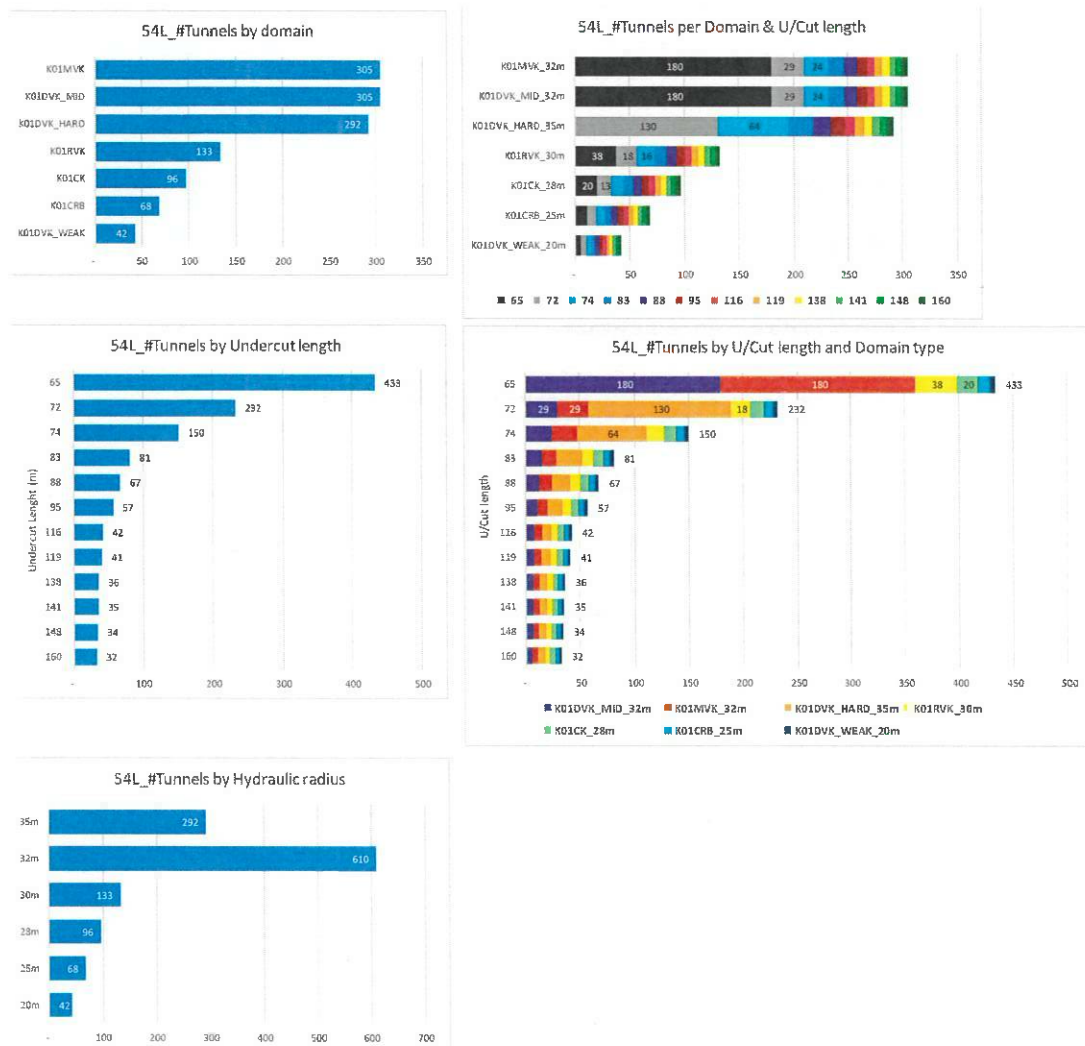


Table B.7. Numerical data representation for 54 level.

Sum of No: tunnels to open for Caving	Column Labels	65.16	71.93	74.05	82.62	87.96	94.8	116.18	119.45	138.32	141.16	148.09	160.38	Grand Total
KO1DVK_WEAK		5.179650238	4.505480739	4.348796242	3.877053027	3.668404588	3.459854015	3.050144385	3.006795872	2.81366965	2.790826414	2.740123971	2.66456222	42.10536137
20		5.179650238	4.505480739	4.348796242	3.877053027	3.668404588	3.459854015	3.050144385	3.006795872	2.81366965	2.790826414	2.740123971	2.66456222	42.10536137
KO1CRB		10.74538259	8.1999544	7.695344971	6.332004905	5.793807642	5.290178571	4.388788154	4.29959689	3.915307971	3.871215445	3.774339892	3.632451531	67.93837296
25		10.74538259	8.1999544	7.695344971	6.332004905	5.793807642	5.290178571	4.388788154	4.29959689	3.915307971	3.871215445	3.774339892	3.632451531	67.93837296
KO1CK		19.91790393	12.6430634	11.48217054	8.690308039	7.707668232	6.841237113	5.405516783	5.270847778	4.704761905	4.641240019	4.502682159	4.302203487	96.10960339
28		19.91790393	12.6430634	11.48217054	8.690308039	7.707668232	6.841237113	5.405516783	5.270847778	4.704761905	4.641240019	4.502682159	4.302203487	96.10960339
KO1RVK		37.88372093	18.08801341	15.80227596	10.95755968	9.440071556	8.172413793	6.203987184	6.027245207	5.298263534	5.217841301	5.043364741	4.793185894	132.9279432
30		37.88372093	18.08801341	15.80227596	10.95755968	9.440071556	8.172413793	6.203987184	6.027245207	5.298263534	5.217841301	5.043364741	4.793185894	132.9279432
KO1DVK_HARD		130.4430052	63.84482759	22.91362916	17.14902507	13.37903226	8.805326981	8.453497776	7.086065574	6.942945475	6.637405558	6.210776721	5.8655373	291.8655373
35		130.4430052	63.84482759	22.91362916	17.14902507	13.37903226	8.805326981	8.453497776	7.086065574	6.942945475	6.637405558	6.210776721	5.8655373	291.8655373
KO1DVK_MID		179.7517241	29.0259773	23.55785288	14.19892589	11.75114823	9.849350649	7.124875431	6.892751533	5.855651238	5.854224987	5.635485789	5.324922183	304.9228902
32		179.7517241	29.0259773	23.55785288	14.19892589	11.75114823	9.849350649	7.124875431	6.892751533	5.855651238	5.854224987	5.635485789	5.324922183	304.9228902
KO1AMVK		179.7517241	29.0259773	23.55785288	14.19892589	11.75114823	9.849350649	7.124875431	6.892751533	5.855651238	5.854224987	5.635485789	5.324922183	304.9228902
32		179.7517241	29.0259773	23.55785288	14.19892589	11.75114823	9.849350649	7.124875431	6.892751533	5.855651238	5.854224987	5.635485789	5.324922183	304.9228902
Grand Total		433.230106	231.9914717	150.2891211	81.16840638	67.26127354	56.84141705	42.109351436	40.94348659	35.72937111	35.17251863	33.96888879	32.25302422	1240.792599

Table B.8. Graphical representation of domains versus undercut length/ Hydraulic radius.



7.3 Permission to conduct research study

DE BEERS GROUP

20 June 2019

Mr T Mudeu
P.O.Box 617
Dzanani
0955

RE: PERMISSION TO CONDUCT RESEARCH STUDY

Reference is made to your letter dated 21 April 2019, requesting permission to conduct research for your master's report in which the report will be focusing on the De Beers operation and primarily on the Venetia Underground Project.

We are delighted to inform you that your request has been successful and we are looking forward to assisting you in your research.



Richard Greig
Senior Site Manager - Venetia Underground Project
Tel: 015 575 2642

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