Investigation of Vertical Cutter Mining for Increased Primary Resource Recovery and Decreased Environmental Impact

A VCM Study for De Beers, Victor Mine, Canada

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Challenge the future

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A VCM Study for De Beers, Victor Mine, Canada

by

M. A. Groenewegen

in partial fulfillment of the requirements for the degree of

Master of Science
in Applied Earth Sciences

at the Delft University of Technology,
to be defended publicly on Friday September 8, 2017 at 10:30 AM.

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Project duration:	February 8, 2017 – September 8, 2017	
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Preface

The mining industry is generally perceived as a rigid industry that has been hesitant to reform. Today's societal development requires constant change and an increased awareness on the impact of any industrial activity. Regardless of whether the hesitation to reform was justified in the past, currently, the mining industry is greatly updating its known and applied methods. Learning from technologies and experiences in other industries can be an extremely valuable method of getting ahead more quickly.

During the winter of 2016, I was introduced to the trench cutting technique during an internship at Bauer Maschinen GmbH. Bauer, having developed the trench cutting technology for the civil industry, suggested a cross over to the mining industry. With some experience in diamond mining, they were eager to learn what likely mineral deposit types could be suitable for extraction using their trench cutters. Completion of this study was followed with the suggestion to expand on kimberlite extraction in more detail. I identified diamond bearing kimberlites as the deposit type with the greatest potential for success and suggested performing a case study to prove so. I am very happy and grateful to have taken part in the follow-up and the ability to aid the development of new and alternative mining technology.

M. A. Groenewegen

Delft, August 2017

Abstract

The purpose of this thesis is to aid in the development of vertical cutting mining as a mining method. In order to do so, the technical feasibility and viability, and environmental benefit of vertical cutting as a complementary mining method were investigated. The investigation was performed for the case of the Victor Diamond mine in northern Ontario, Canada, where open pit mining ends by the end of 2018 or beginning of 2019.

Vertical cutting has been used for several decades for the construction of water retention walls in the civil engineering industry. By placing the vertical cutter system directly on top of an ore target and cutting straight, vertical trenches up to a maximum depth of 150 m, it is intended to cross over to the mining industry. Extraction with vertical cutting can occur according four extraction scenarios. Three of the scenarios are land-based, the fourth assumes flooding of the mine, and has not been considered for the Victor project.

Checkerboard mining is the base case extraction scenario with an extraction rate of approximately 30%. The long trenching scenario would increase the recovery with an additional 15% but induces a high risk of instability in the existing pit walls and the kimberlite in between the trenches. Application of backfill is the third scenario and achieves a recovery of 98%. Backfilling of the trenches requires the movement of significant volumes of additional rock as well as induces time delays due to the curing time of the backfill.

Financial evaluation of the vertical cutting scenarios shows a high dependency of the project value on a decreasing cutting performance. Cumulative cash flow analysis and NPV suggest that extending the mine life at the Victor Diamond mine with vertical cutting is favourable. Even in the case of increased rock strengths, as expected in the deeper parts of the Victor pipes, vertical cutting has a positive net present project value. Long trenching, which is considered to be of high risk for pit stability has only marginally greater project value than the base case.

The development of alternative mining solutions also aims to reduce the impact of the mining operations on the surrounding environment. Vertical cutting combines multiple mining processes into one operating piece of equipment. It reduces the GHG emissions, improves the safety of extraction process and is expected to increase the support from stakeholders. Extending operational life using conventional methods would require large expansion of the mine involving the increase of the operational fleet, pumping capacity and land usage. The application of vertical cutting has the ability to prevent the negative impact of enlarged open pit mining while maintaining the benefit of continued production.

Acknowledgements

This thesis project is the last assignment before I complete my Master of Science studies at the Resource Engineering section at Delft University of Technology. I would not have been able to do all this work alone and, therefore, I would like to thank everybody that contributed to this project.

First of all, I would like to thank Stefan Schwank from Bauer Maschinen for providing me with the opportunity to continue my internship research during this thesis, his assistance and pragmatism were of invaluable help during this study. In addition, Stefan enabled a visit to an active cutting site in Paris. Seeing the cutting systems operate proved to be of tremendous help for the optimization of the cutting schedule and my general understanding of the system.

Furthermore, I would like to thank Mike Buxton for supervising my thesis. The discussions we had allowed me to progress through my research the way I did. His comments and suggestions forced me to continuously evaluate my performed work while simultaneously providing the guidance for future tasks. Dominique Ngan-Tillard and Phil Vardon deserve thanks for their input and their assistance during the geotechnical assessment and evaluation, as well as the numerous other TU Delft staff that provided me with their advice.

Special thanks go to De Beers Group Canada and more in particular Stephan Kurszlaukis and Hani Qureshi. They supplied me of their valuable advice and expertise on the Victor mine. I would also like to thank many other De Beers employees providing me with the Victor mine data, suggestions and answers to my many questions.

Last, but not least, I would like to recognize the contribution of my family and friends. To you, no question was too trivial whether it concerned the use of a certain colour scheme or the application of different finite element modelling techniques. Every conversation we had, concerning my research and everything but the thesis, contributed to the results of this project.

Contents

List of Abbreviations

BFA	Bench face angle
BSMT	Basement rock
Can$	Canadian dollar code
CBH	Critical buckling height
cpht	carat per hundred tonnes
DIN	Deutsches Institut für Normung
FN	First Nation
FoS	Factor of safety
GHG	Greenhouse gasses (water vapour, CO_2, CH_4, N_2O, O_3)
HDS	Hose drum system
HK	Hypabyssal kimberlite
ICMM	International Council on Mining & Metals
IRA	Inter ramp angle
IRR	Internal rate of return
Is_{50}	Point load index strength
K	Point load conversion factor
L/h	litres per hour
LDD	Large diameter drillhole
LMST	Limestone rock
MVK	Massive volcanoclastic kimberlite
N.A.	Not available
NAD	North American datum
NPV	Net present value
PK	Pyroclastic kimberlite
RHR	Red Head Rapids (geological rock formation)
RMR^{76}	Rock mass rating (1976)
RQD	Rock quality designation
RS^2	$Phase^2$ 9.0, 2D finite element software created by Rocscience Inc.
RVK	Resedimented volcanoclastic kimberlite
SRF	Strength reduction factor
SRK	Natural Resource Solutions; SRK consulting Ltd.
t/a	tonnes per annum
t/h	tonnes per hour
TL	Trouser leg
UCS	Uniaxial compressive strength
VCM	Vertical Cutter Mining
VK	Volcanoclastic kimberlite
VMD	Victor Main Deep
VNW	Victor North West
VSW	Victor Southwest

Units: Conventional SI units and prefixes used throughout: {k, kilo, 1000} {M, mega, 1,000,000} {G, giga, 10^9} {kg, kilogram, unit mass} {ct, carat, 200 mg} {t, metric tonne, 1000 kg} {m, metre, unit length} {Pa, pascal, unit stress and pressure (N/m^2)}

1
Introduction

1.1 Background

In 2016, *Bauer Maschinen GmbH* (Bauer) proposed the crossover of trench cutting technology from the civil engineering industry to the mining industry. An initial study aimed to indicate the business potential of that technology regarding the mining of specific deposit types [1]. Through a comprehensive database including different commodities, mineralization types and deposit models, a relative ranking system was designed to indicate the mineral deposit types that upheld the greatest potential for further research regarding suitability of vertical cutting as an extraction method.

The conclusion of the research stressed the potential of vertical cutting in the mining industry. The result showed that deposit types of greatest interest had in common that they are hosted in vertical structures, are spatially situated near surface and are associated with weaker rock types. Small and steep dipping deposits receive preference as they conform to the size of the cutter installation, and make optimal use of its versatility. Vertical cutting enables recovery of reserves in already existing open pit mines without the need of push-backs. It is therefore ideal for salvage-type operations and operations that seek to prolong their operational lifetime. Kimberlite rock, which ranked highest for virtually all parameters, was identified as the primary target for continued research.

The follow-up step after the initial research is to design a detailed work plan for the extraction of ore from a specified deposit.

1.2 Motivation

The mining industry is an industry with great impact on its surroundings and stakeholders. Reducing the impact of the industry or the duration of that impact has grown to be a key focus for the majority of the publicly listed mining companies. Efficient mining methods are sought after in order to decrease energy or fuel dependency, decrease the long-term spatial and environmental footprint of the operation and maintain or increase the material throughput.

Vertical cutting could prove to be an efficient mining method that allows the extraction of resources in small deposits or previously abandoned mines. It reduces the amount of overburden that needs to be removed in order to reach the target mineralization. The spatial impact of the operation would drastically decrease to virtually only the area above the mineralization itself and a processing facility. In addition, mechanical cutting has been proven to be an efficient and safe mining method, replacing multiple stages of the extraction process into one piece of equipment and avoiding the need of blasting.

1.3 Purpose

The purpose of this thesis is to provide a report that contributes to the initial stages of developing vertical cutting as an accepted mining method. In order to do so, a case study on which the technology could be proven and tested is required. The Victor Diamond Mine of *De Beers Group Canada* (De Beers) is considered to be an ideal case as operations are scheduled to end by early 2019 due to inability to expand the pit further. Economic limitations and availability of current mining technology instigate development of alternative mining solutions.

2
Objectives

The objective of this thesis is to investigate the technical feasibility and viability, and environmental benefit of using vertical cutting as a complementary mining method when open pit mining ends in the Victor diamond mine by 2019.

A total of three research questions will be dealt with in order to come to a conclusion:

1. What is a suitable mine design, extraction sequence and task schedule of the extraction processes for vertical cutter mining during resource salvage at a late project stage?
2. What is the added value of vertical cutter mining in terms of additional resources recovered, and how does it compare to conventional open pit or underground mining?
3. Which global standards related to environmental and social impact apply to vertical cutting and how do indicators compare to conventional mining methods?

2.1 Scope of work

The scope of this thesis is to determine on a prefeasibility level the potential of using vertical cutting as a suitable extraction technique for the Victor Diamond Mine. The thesis is limited to the determination of the technical feasibility and a comparative analysis for alternative methods. It is intended to highlight risks, opportunities and dependencies for further research. Calculating true cash-flows or potential profit is not an objective of this thesis.

Activities within the scope of the thesis consist of:

- Review on project details and geotechnical properties of the Victor kimberlite pipes;
- Development of a general mine design for vertical cutter mining;
- Development of potential mining scenarios related to extraction sequence and interaction with auxiliary processes (rock stability, climate conditions, space management, etc.);
- Assessment of recoverable reserves;
- Development of preliminary project plan;
- Evaluation of project time lines and preliminary cost scenarios, and comparison to conventional mining;
- Sensitivity analysis to identify further areas of investigation in more detail; and
- Preliminary assessment of environmental and social indicators, and comparison to conventional mining.

2.2 Exclusions

The following subjects will not be considered for the completion of this investigation:

- Alternative extraction methods, other than vertical cutting, e.g. extraction of resources using a clamshell set-up;
- Diamond grade analysis and distribution inside the kimberlite pipe, including spatial estimation of diamond grades within the deposit;
- Diamond damaging or crushing due to grinding by the cutter wheels;
- Processing efficiency of the Bauer desanding units;
- Diamond market indicators and price estimation; and
- Profitability analysis of the overall project.

2.3 Thesis structure

As defined, the purpose of this thesis is to contribute to the development of vertical cutting as a mining method. In order to do so the method is assessed in light of the Victor Diamond Mine case in northern Ontario. This thesis will address kimberlite geology and conventional mining methods based on literature and established mining principles, first in Chapter 3 and 4. The conventional mining method analysis demonstrates the need for alternative methods for the Victor mine case.

Chapter 5 of the thesis introduces vertical cutting. The method, used for the construction of in-situ water retention walls around underground excavations has been in use for decades in the civil engineering industry. In this chapter the basic layout of the vertical cutter system is described, as well as the cutting process and cutting performance. The performance of cutting equipment is largely dependent on the intact rock strength: the weaker the rock, the greater the production rate. Lastly, four extraction scenarios for vertical cutter mining are introduced.

Chapters 6 and 7 address the Victor Diamond mine and the applicability of the vertical cutting extraction scenarios on the extraction of its residual resources. Based on the evaluation of the extraction scenarios and the sequencing of production cuts, extraction rates of vertical cutter mining are estimated. The spatial availability inside the mine as well as the stability of the rock surrounding the trenches are crucial in the scenario evaluation.

The project time line is relevant in the development of vertical cutting, especially for the Victor mine. The mine's remote location and its dependency on a winter road for the transportation of heavy loads require careful planning of all processes before production can start. The financial evaluation follows the project time line, presenting an initial cumulative cost and cash flow model. The primary aim of the financial evaluation is to establish the influence of project parameters on the net present value of the project, as well as the financial comparison of the extraction scenarios.

Chapter 10 discusses the environmental and social indicators that are expected to be relevant or under influence of vertical cutter mining. Indicators are compared with conventional mining, like in the case of the Victor mine. However, it has been aimed to address indicators such that the analysis may be applied on different cases as well.

3

Kimberlites: Primary Source of Diamonds

Diamonds have captivated humanity's interest for many centuries. The diamond reflects a material that is pure, brilliant, hard and of extreme value. It was first found and sought after on a large scale in the river sands of India between 800 and 600 B.C. [2]. For centuries, the diamond has represented wealth and love and purity.

India has been the only source of diamonds for most of humanity's history. The river sands they were found in, however, are in terms of geological origin a secondary source. Due to their hardness, diamonds are virtually unsusceptible to weathering by natural processes, as opposed to the rock it is usually found in. The true source rock of diamonds is called kimberlite, an intrusive igneous rock in which diamonds may or may not be located. Weathering of the kimberlite frees the diamonds and subsequent transportation by for instance water may lead to them being placed in secondary deposits like river sands. [3]

This chapter will introduce kimberlite as the primary source rock of diamonds. An overview of the general geology of kimberlites and emplacement styles will be discussed. The Victor kimberlite deposit serves as the case study for this thesis. It is located in the James Bay lowlands of northern Ontario, Canada, and part of the Attawapiskat kimberlite cluster.

3.1 General Geology

At present day, the most common source of mined diamonds originates from kimberlites and lamproites (for simplicity also referred to as kimberlites). Kimberlite is an igneous rock associated with volcanic rock structures and may or may not contain diamonds. Aside from being the source rock for extraction on earth's surface, it is commonly accepted that the true source of diamonds lay not within the rock itself but in Earth's lithospheric mantle. This theory was substantiated by the discovery that the diamonds are older than their host rock [4]. Diamond formation is associated with high pressures and moderate temperatures [5]. When these magmas rise to surface and form a pipe-like structure they are referred to as kimberlites.

Diamonds are brought to surface by a series of volcanic eruptions. In the magmas, diamonds can be present as "strange minerals" or xenocrysts, and require rapid rise of the magma in order to not be graphitized or resorbed by the magma. When these mantle magmas, containing the diamond xenocrysts, reach the upper lithosphere or Earth's surface they form primary diamond deposits in subvolcanic pipes, sills and dykes. Besides diamonds, the magmas have incorporated many other xenocrysts and rock fragments, or xenoliths, during their ascent to Earth's surface. Hence, kimberlites are referred to as highly variable in their composition and appearance. [3]

Until the discovery of kimberlite in 1870 in South Africa, primary diamond deposits rarely served as source of diamonds. Due to weathering of the diamondiferous rock, secondary deposits, like alluvial sources, were much more common. Diamonds, due to their hardness are not very susceptible to supergene alteration as opposed to their kimberlite host. Therefore, after transport, diamonds were often found in placer deposits associated with rivers, sea beds or other weathered material like the ones in India [5].

In 1869, Boer prospectors discovered kimberlite pipes on the farms of Bultfontein and Dorstfontein in South Africa [6]. Prospectors moved away from the Vaal River, where diggings for alluvial diamonds had grown crowded. By 1870, a newly founded mining town emerged in Kimberley, South Africa, marking the turning point for the extraction of non-alluvial diamonds. A few months later, two other great kimberlite pipes were discovered on an adjacent farm and mining quickly followed at the De Beers and Kimberly mines [7]. The discovery of the diamond holding volcanic pipes at Kimberley later gave name to the igneous rock inside the pipes: kimberlite. [6]

Following the discovery of diamonds in Kimberley, exploration towards kimberlites quickly spread. Kimberlites, then referred to as any igneous rock containing diamonds, were found throughout what is currently known as South Africa [6]. Due to the large amount being discovered there, most current kimberlite definitions are based on the kimberlites found in southern Africa. Skinner and Clement [8] defined kimberlite rock as an igneous rock which occurs in small volcanic pipes, dykes and sills. It is comprised of a fine grained matrix with a distinctive presence of large minerals inside the matrix as well as potentially other rocks, xenocrysts and xenoliths. In contrast with the first description of kimberlite, the rock may or may not contain diamonds in a low quantity [8]. In fact, a majority of known kimberlites does not show any presence of diamonds. If diamonds do occur, their quantities are reported in carats per tonne of kimberlite, or 200 mg per tonne.

The definition of Skinner and Clement [8] describes a highly variable rock both internally, as on a macro scale, showing considerable potential alterations between two different locations on earth. However, the classic emplacement model for kimberlite, is quite consistent. The carrot-shaped kimberlite magmatic system, consisting of a root zone, a steep-sided diatreme, and a relatively shallow crater zone appears to be especially consistent for South African kimberlites [9, 10]. Figure 3.1 displays the classic kimberlite emplacement model for South African kimberlites. The model shows a simplified lithological sequence adapted from Hawthorne [10] and Clement [11]. The crater zone is infilled with the rocks that comprised the volcanic fountain during the eruption: resedimented volcanoclastic kimberlites (RVK) and pyroclastic kimberlites (PK). The main part of the pipe, the diatreme zone, is occupied by tuffistic kimberlite breccia (TKB) which is typically fairly uniform in texture and composition [12]. Hypabyssal kimberlite (HK) which has crystallized subsurface and not experienced degassing occupies the root zone of the classic South African kimberlite pipe [13, 14].

Discoveries of new kimberlites around the world have forced a review of the widely accepted South African emplacement model. A total of three emplacement styles are distinguished [15]:

1. Deep steep-sided pipes conforming the South African model;
2. Broad, shallow craters occupied primarily by pyroclastic kimberlite;
3. Small, steep-sided pipes dominated by resedimented volcanoclastic kimberlite.

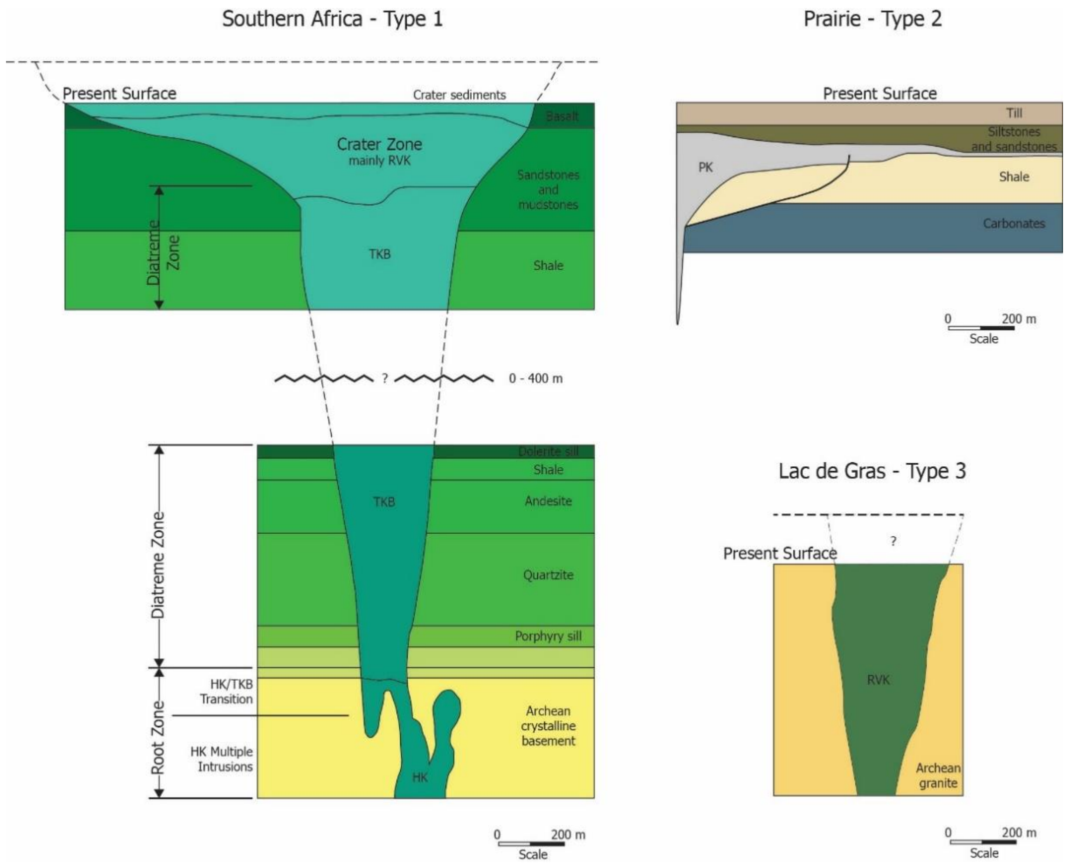


Figure 3.1: Emplacement models of kimberlites illustrating three emplacement styles adapted from Gurney et al [15], Hawthorne [10] and Clement [11]. All diagrams are at the same vertical and horizontal scale. Type 1: model of deep, steep-sided Southern African kimberlite volcano consisting of crater, diatreme and root zone. Country rock are simplified and illustrate a typical sequence in southern Africa. Type 2: Shallow, crater-shaped kimberlites of the Canadian Prairies. Type 3: small, steepsided pipes characterized by the kimberlite volcanoes of the Lac de Gras field.

Many known kimberlites conform the steep-sided emplacement model. However, the continuing discovery of kimberlite pipes proves that the geology and emplacement mechanisms are highly variable worldwide, as was expected according Clement and Skinner's definition. Especially kimberlites found in Canada display a wide range of variations compared to the classic South African model. The broad and shallow model (2) characterizes the kimberlites found in the Canadian prairies where the pipes are thought to be the result of explosive formation processes in sedimentary rocks like marine sediments or porous sandstone [15]. The kimberlites in the Canadian Slave region (Northern Territories) are characterized by small, steepsided kimberlites (3) and are often referred to as "Lac de Gras" style, after the Lac de Gras pipe. The pipes of this type are characterized by massive resedimented volcanoclastic kimberlite (RVK) or tuffistic kimberlite breccia (TKB). The new De Beers mine in Canada's Northwest Territories, Gahcho Kué, conform the small, steep-sided model. The regional geologic setting in the Canadian Slave region, consisting of competent crystalline rocks, has induced the difference in the emplacement mechanism [15].

3.2 Attawapiskat kimberlites

The ca. 170 Ma Victor kimberlite pipe is part of the Attawapiskat kimberlite cluster, consisting of a total of 19 known kimberlite pipes [16]. The cluster forms a north-northwest trending band through the country rock geology and is located on the Canadian Shield in northern Ontario. The highest spatial density of the kimberlite pipes is located in the north of the cluster, close to the Attawapiskat River, where also Victor is located. The geophysical anomaly at the Victor location was considered to be of prime interest for further exploration. The Victor kimberlite pipe is both the largest of the Attawapiskat kimberlites as well as yielded the best results of a mini-bulk sampling program [17].

The cluster was discovered in 1988 by traditional glacial sediment sampling and airborne magnetic surveys [18]. After confirmation that the geophysical anomalies were kimberlites, further work was postponed because the kimberlites were considered to be only of moderate interest. However, diamond recoveries of the Attawapiskat kimberlites now suggest that approximately 93.8% of the pipes are diamondiferous [19].

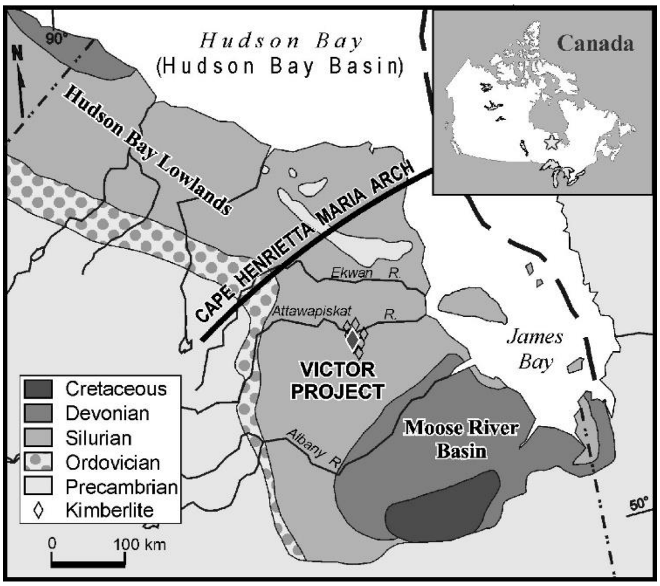


Figure 3.2: Regional geological setting and location of the Attawapiskat kimberlites and Victor Project (after Kong et al. [18] and Norris [20])

4

Conventional Mining Methods

Mining is commonly simply referred to as the extraction of the valuable materials from either Earth's subsurface, or human waste heaps ("urban mining"). The extraction of material of value, ore, is as old as civilization itself and provides the source of virtually all resources used in humanities personal and industrial activities. At present day, a range of mining methods is differentiated that enable the mining company the most efficient way of extracting Earth's resources.

The mining industry is a high impact industry on multiple levels. On an economic level mining activities contribute to the regional and or national economy by employment, outsourcing of contracts, taxes and royalties paid to appropriate institutions, etc. [21] Also, within the company the extraction and processing of ore is associated with high investments, and commonly offset by high revenues originating from the sales of concentrate. On a societal level, economic benefits are combined with additionally environmental and social impacts on the surrounding nature and communities. The usage of large tracts of land, the application of chemicals in complex industrial processes and the emission of noise, vibrations and greenhouse gasses are major concerns that influence people and the local ecology surrounding the mine site.

Technology can be the driver of beneficial impacts as well as the way of mitigating negative impacts of the mining industry. Currently, the Victor mine employs conventional open pit mining. This chapter explores conventional mining methods, a selected amount of trends within these methods and some of the consequences that they may have on prolonging the mine life of the Victor Diamond Mine.

4.1 Mining method selection

Before extraction of a mineral deposit can take place, a potential ore body has to be extensively investigated and mapped. Mineral exploration as such converts a mineral occurrence into a body that can be of value by delineating the extents of the body and estimating the grades inside the deposit. Subsequent, and more detailed studies may follow once the mineral occurrence has been determined to be of value. Later studies, in turn, take more characteristics of the ore body into account in order to design a suitable mining method, constrained by the size of the body, technical properties such as rock strength, value and environmental and social limits.

In 1993, Nicholas [22] presented the selection procedure for the determination of a suitable mining method for an ore deposit. His classification is predominantly based on 3 considerations: geometry, grade distribution and rock mechanics characteristics. Based on a relative rating system, the characteristics of an ore body including the rock quality of the hanging wall, ore zone and footwall, are rated in order to determine potentially suitable mining methods. Although not necessarily giving a definitive answer, the system already narrows the search for the most efficient method extensively. Mining method selection, in fact, is dependent on several more characteristics than the three used in Nicholas' classification.

Nicholas' numerical selection procedure accounts for a total of ten mining methods. If applied correctly, only a limited amount of those should appear to be applicable in theory for the extraction of an ore deposit like a kimberlite pipe. As discussed in Chapter 3, kimberlite pipes are generally vertical structures consisting of a steep-sided cone that decreases in size with increasing depth. The strength of the kimberlite rock, which will be assessed in more detail in Section 6.5 of this thesis, is variable but for simplicity it is assumed to be 'moderate'. In this case, the procedure is applied on the kimberlite that is expected to be left behind after excavation in the ultimate pit has stopped. The kimberlite specifically consists of the lower, smaller part of the pipe, directly beneath the excavation. The following three mining methods are ranked highest:

1. Surface mining: open pit (24);
2. Cut and fill stoping (21); and
3. Sublevel stoping (18).

Open pit mining is a surface method and currently applied at the Victor mine, sublevel stoping and cut and fill stoping will be considered together as underground conventional mining methods. It must be noted that for the numerical mining method selection presented above no joint spacing of conditions have been considered.

4.2 Surface mining

Surface mining, being the method that is currently applied at Victor, is considered to be the most relevant conventional mining method that can be investigated for mine life extension. By expanding the pit in a lateral direction with push-backs the mining operations are able to expand deeper into the subsurface. In theory, current operational procedures can continue unchanged in a greater pit, maintaining the production capacities.

As mentioned by Nicholas [22] the slope angle and stripping ratio are critical parameters for an open pit mine. When expanding the open pit mine into greater depths while maintaining the same slope angles, the total volume of the cone that represents the pit expands. The material within the slopes which are generally made in waste material, holds no or little value. Therefore, expanding the pit downward to recover more ore also increases the amount of waste material that needs to be moved from within the pit slopes. The stripping ratio reflects the ratio of volume or tonnage of ore over the volume or tonnage of waste that is moved in order to access the ore. As such, expanding the pit increases the stripping ratio. In practice, the operation will require capacity changes to accommodate for the increase of the amount of material that is moved in order to reach the ore. Changing the capacity of the operation can be translated in additional project costs.

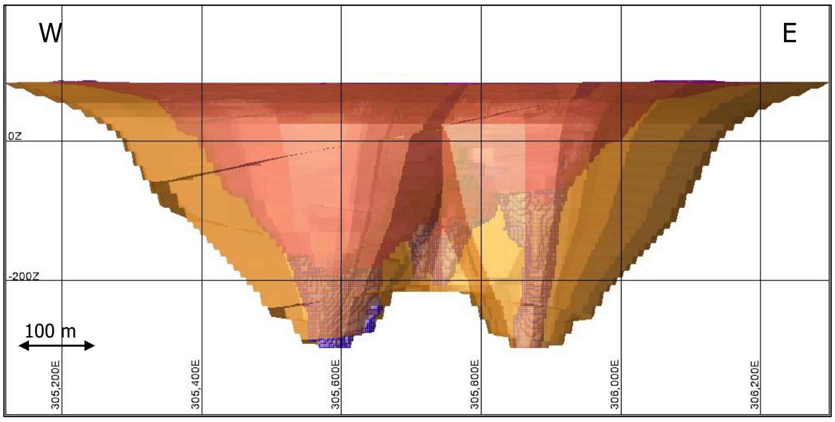
Although surface mining is associated with the lowest relative operating cost of the mining methods [23], the extraction of material that has no value generates no revenue. The stripping ratio contributes to the determination of the design of the ultimate pit. When the stripping ratio increases such that the cost of extracting and moving the units of waste material is no longer offset by the revenue generated by one unit of ore the pit limit is reached. By result, in open pit mining it is aimed to minimize the stripping ratio and maximize the slope angles while maintaining safe conditions.

After production at the Victor mine commenced in 2008 with 'cut 1', the pit was expanded in 2011 with so-called 'cut 2'. The cut 2 is currently still in operation and is prospected to be the final pit at the Victor mine, after which operations are likely to stop. The objective of this chapter is to explore the theoretical options of prolonging operations in the mine with conventional methods. A theoretical 'cut 3', consisting of a new expansion, would follow the cut 2 in order to extract the residual material that is located in the bottom parts of the Victor kimberlite pipes. This conceptual cut 3 would consist of an extrapolation of the cut 2 mine.

The slope design of the Victor pit is according to the geotechnical feasibility report compiled in 2003 by SRK [24]. Controlled blasting is used to excavate 10 m and 20 m benches with a bench face angle (BFA) of 79° in the bedrock. The design inter-ramp angle (IRA) slopes vary between 45 to 55 degrees for the weathered limestone, limestone and kimberlite units based on the feasibility level engineering geology assessments. As some rock units display very competent behaviour, the option of double benching is considered following the updated pit design considerations by Golder Associates [25]. It is likely to assume that mining would continue according to the designed criteria by Golder and SRK in cut 3.

Figure 4.1 displays a conceptual pit design for a theoretical cut 3 based on down-ward extrapolation of the existing design parameters. The pit in Figure 4.1 assumes complete extraction of all residual kimberlite in both the Victor Main and Victor South pipes. The stripping ratio for cut 3 becomes 55:1 by volume, influenced by the waste rock around the kimberlite. The total volume of rock extracted for cut 3 approximates 66.5 million m^3.

(a)



(b)

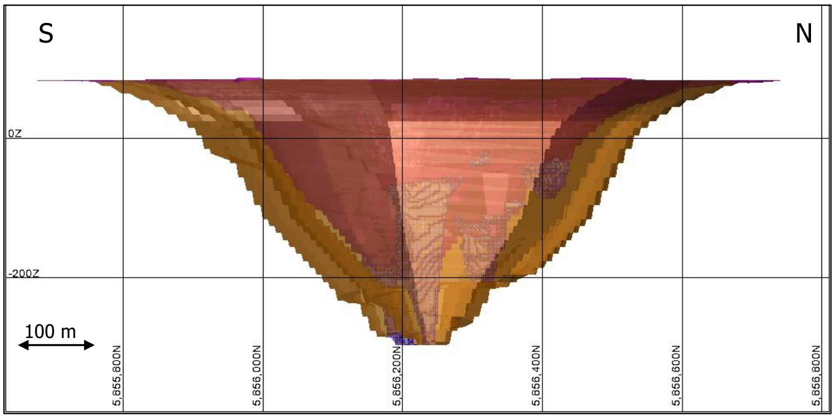


Figure 4.1: Conceptual design of cut 3 (gold), displayed with cut 2 (purple) and kimberlite block model of trouser legs for comparison. (a) Section W-E looking north, (b) section S-N looking west

Stripping ratios of this order of magnitude are unlikely to lead to a sustainable mining operation. Handling of large quantities of waste rock has a significant impact on the financial evaluation of the operation as will be illustrated in Section 9.4. Actual open pit mining is more likely to occur at much smaller stripping ratios, in the order or 3:1. Without forcing the Victor operation to extract all kimberlite resources, it is likely that an optimal pit design may be found with a stripping ratio lower than 55:1. However, the alternative mining methods being sought after is intended to extract all residual resources and conventional pit expansion is therefore compared as such.

4.3 Underground mining

Mining operations that continue operations after open pit mining by transitioning to underground methods are not an uncommon. Also in diamond mining, transition from open pit mining to underground mining have occurred multiple times. One of the most relevant examples is the Diavik Mine in Canada's Northwest Territories. The transition to underground mining started in 2009 and took 3 years and a cost of CAD$ 800 million. After already one year of underground development, Diavik commenced underground mining from the A154N pipe in February 2010. The underground mining methods applied in Diavik were identified during the 2005-2007 feasibility studies to be blast hole stoping with cemented backfill for the stronger kimberlites (>30 MPa) and underhand cut and fill for the kimberlites weaker than 30 – 50 MPa. The underhand cut and fill mining method was replaced by sub-level retreat mining after technical studies performed in 2010. [26]

At this stage, making the transition to underground mining, sublevel stoping or otherwise, is not desirable for the Victor mine. The relatively limited amount of the kimberlite remaining beneath cut 2 is considered insufficient to justify the capital investment of the necessary underground infrastructure, and change in equipment. For comparison, Table 4.1 summarizes the volume of the proven and probable ore reserves at the Diavik mine, as of December 2015, after 5 years underground production. The estimated volume of the total amount of kimberlite material below cut 2 in Victor (not considering resource/reserve classifications) is approximately a tenth of the total Diavik reserves as of December 2015.

Table 4.1: Proven and probable kimberlite ore reserves at the Diavik mine, as of December 2015 (adapted from Diavik diamond mine: 2015 sustainable development report [27])

Pipe	Tonnes (millions)
A154N	8.8
A154S (underground)	1.5
A418 (underground)	4.6
A21 (future open pit)	3.7
Ore stockpile	0.1
Totals	18.7

4.4 Mining technology innovation

Although investigating the options of conventional mining at the Victor mine in great detail is not within the scope of this research, comparison of the stripping ratios and the possible reserves for a future underground mine suggest that conventional mining will not provide the desired solution towards mine life extension. Extending operational time with these methods is expected to be associated with very high costs and unacceptable impact on the mine's surroundings.

Professional services company, Deloitte, publishes annually a report tracking the trends that the mining industry is facing. Aside from Deloitte, other consultancy firms, as well as the International council on mining and metals (ICMM) extensively report on challenges and opportunities of innovation within the mining industry. Returning items like stakeholder engagement, reduction of fossil fuel produced energy and the ability to mine lower grade ores have dominated the outlooks for years [28, 29, 30]. Mining companies continuously face these and other operational challenges or constraints. While the demand for raw materials grows ever greater, the amount of "easy" mineral deposits grows smaller. Valuable resources are located deeper, are less accessible, exhibit lower grades and are located in close proximity to communities or natural habitats. The need for innovation to handle these challenges is great.

In their 'Tracking the trends 2016' report, Deloitte highlighted the strive for operational excellence. Finding new and innovative solutions for increased material output and automation was indicated as the number one trend for the mining industry in 2016. Learning from other industries for best practices has been reported to be one of the most efficient methods to reach greater productivity, better safety and environmental standards, and improve the overall innovation system within the industry [28].

Mechanical rock cutting is a prime example of innovation in the mining industry. Although as a method being in use for a longer amount of time, it is not widely applied as an extraction method. Rock blasting, especially in surface mines is the dominant method of fracturing rock [31]. In the civil engineering industry rock cutting has been used efficiently for decades not only as a tunnel boring technique but also to make vertical cuts from surface. The remainder of this thesis will investigate the potential of the mechanical cutting of vertical cuts from surface, "vertical cutting", as an extraction technique. In order to do so, vertical cutting technology needs to cross over from the civil engineering industry to the mining industry. Extraction scenarios need to be designed and operational constraints will change. An important difference between applications in civil engineering and mining is the volume of material being handled. As an extraction technique, the total amount of rock handled by cutting equipment need to be far greater. Increasing the volume of material extracted can be both an objective of mining and a common method applied to decrease the operational cost per tonne ore. For this study it is assumed that a single vertical cutting system is used, as opposed to multiple pieces of equipment. Increasing the material throughput by implementing multiple cutting systems may be possible, but is not investigated.

5

Vertical Cutter Mining

During the 1980's, the civil engineering industry underwent an important change regarding the development of underground water retention walls, or diaphragm walls. The development of the trench cutting technique led to the ability to increase the wall thickness and reach greater depths. The development of steering plates increased the accuracy with which the cutter could advance vertically. By the 1990's, rocks with increasing compressive strength could also be cut. At present day Bauer is able to deliver this trench cutter technology to its customers customized to their demand.

For over 30 years, the trench cutter technology has been developed for the civil engineering industry. Its main application is the in-situ construction of retention walls around underground construction sites and areas with a high ground water level. The trench cutter is specifically designed to cut rock vertically and is different than most known excavation or rock cutting tools used in the mining industry. This chapter describes the proposed cutter and base machine, theoretical cutting performance and cutting sequence. The application of vertical cutting in mining considers a multitude of extraction scenarios based on the cutting sequences. The scenario design is independent from the location or deposit vertical cutting is used in and is addressed at the end of this chapter.

5.1 Cutting equipment overview

A vertical cutter differs from the widely applied mining methods in the way that rock breaking and excavation is not done by the use of blasting or ripping. Mechanical cutting of rock has experienced a recent surge in development in underground mining, attempting to improve the continuity of the ore production cycle. Roadheaders and continuous miners are now widely developed and applied in underground mines with coal and soft rock, like potash and salt. Among improving the speed of the operation mechanical rock cutting has three other advantages in underground hard rock mining (adapted from Vogt [31]):

- Reduction of the vibrations and noise that propagate through the rock, reducing both the impact on neighbouring communities and the need for support;
- High precision and accuracy of cutting along the designed tunnel or stope outline, increasing the selectivity and potentially reducing the volume of the open spaces; and
- Reduction of the amount of people exposed to dangerous working environments associated with explosives.

Although mechanical cutting can be used effectively in soft rock and coal, hard rock conditions still pose a challenge for the development of cutting tools. Tunnel boring techniques, for instance used for shaft sinking in underground mines, can cut hard rock, but the operation remains inflexible and the cost of the boring machine makes it unattractive as an extraction technique. Tunnel-boring equipment can operate both horizontally and vertically, but due to their low mobility, they are not able to follow sharply changing or dipping ore bodies [32].

Vertical cutting, produced by Bauer, will be applied from the surface in a (sub-)vertical direction and is highly mobile. In underground mining the direction of advancement is commonly horizontal instead of vertical, yet, the same advantages are applicable to both types of mechanical cutting. One important difference between the known underground cutting techniques and vertical cutting is the ability to utilize gravity, and therefore the weight of the cutter, when cutting from surface. Whereas the horizontal thrust capacity is often a limiting factor for roadheaders, trench cutters can use their weight to exert the desired force on the rock [33].

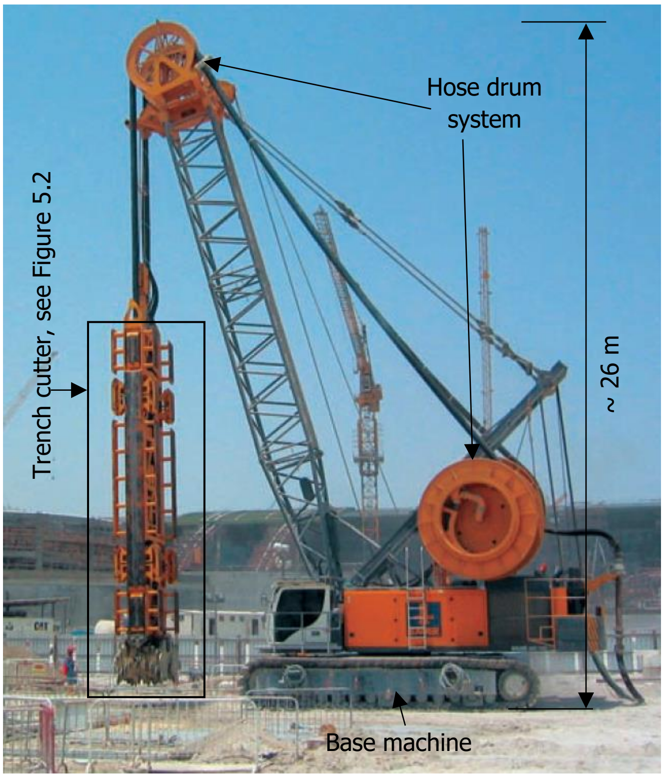


Figure 5.1: Side view of trench cutter system mounted on a MC 64 crawler crane with hose drum system (HDS) and height of approximately 26 m [34]. Figure 5.2 provides a detailed overview of the trench cutter components.

A trench cutter, referring to the cutter head (Figure 5.1), mechanically cuts the rock in a vertical direction. Starting from the surface the cutter advances through the rock downward, creating a trench the size of the cutter frame and of predefined depth. In general, a trench cutter consists of the following parts (indicated in Figure 5.2 on the next page):

- Cutter frame containing a mud pump and mud hose;
- 4 cutter wheels;
- Steering plates;
- 2 gear boxes; and
- Hydraulic hoses.

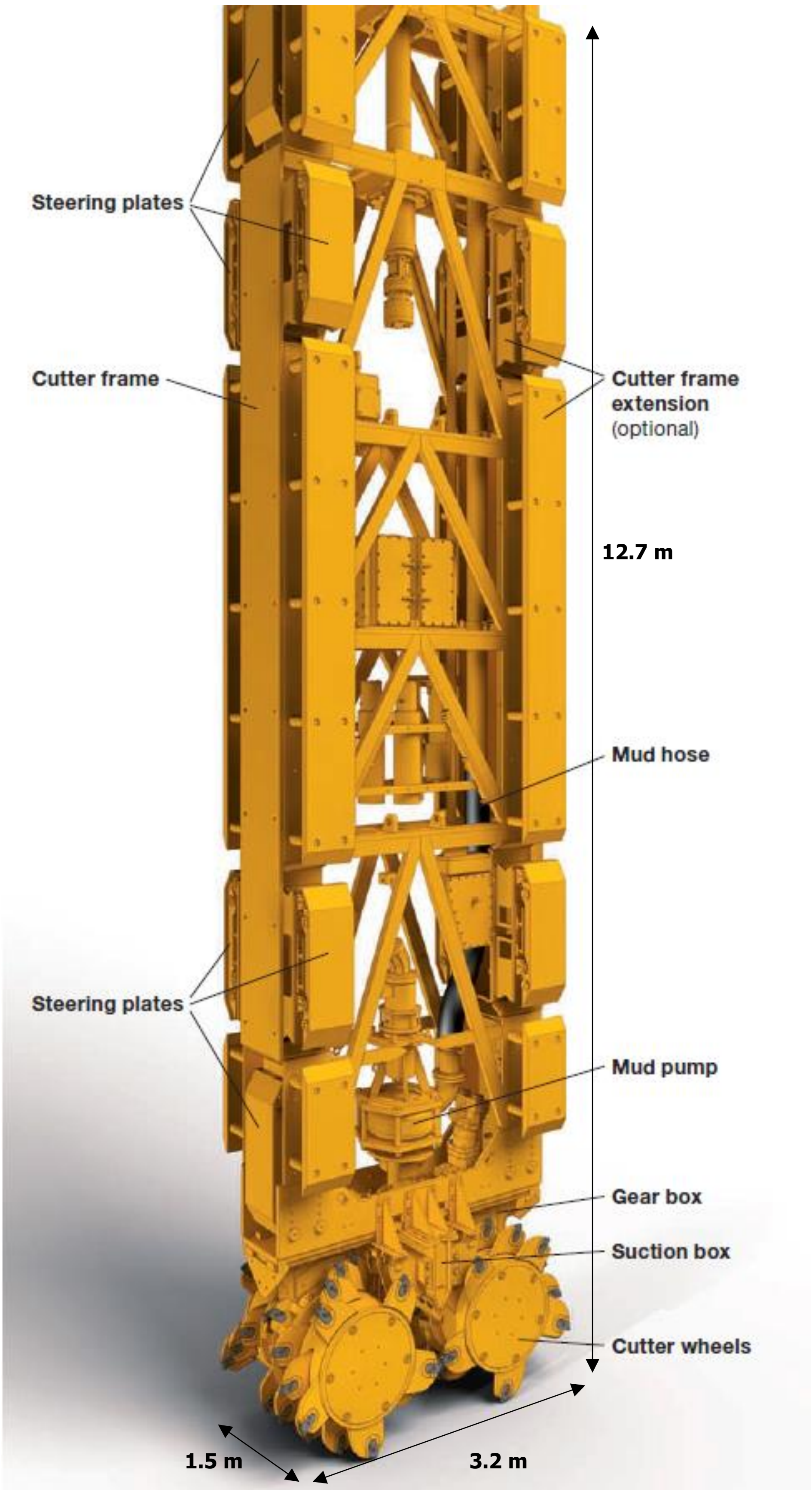


Figure 5.2: Main components of Bauer Trench cutter, dimensions given for a BC 50 cutter [34]

Bauer Maschinen GmbH produces four types of standard cutters, each of them can be configured differently and applied with different cutter teeth set-ups depending on the rock conditions. For the extraction of kimberlite at the Victor mine the BC 50 trench cutter is proposed. The BC 50 is the trench cutter that can exert the highest torque of the four cutter types and has the greatest weight, resulting in the greatest load applied by the cutter. The properties of the BC 50 trench cutter are summarized in Table 5.1, adjusted for the mining application at the Victor mine. Increasing the weight of the cutter head from 50 tonnes, the standard BC 50 weight, to 70 tonnes is an important adjustment. Specifically designed to increase the vertical thrust capacity of the cutter and therefore reach greater production rates than those applied in the civil engineering industry.

Table 5.1: Properties of BC50 trench cutter [34]

	BC 50
Torque max. (kMm)	2 x 120
Cutter width[1] (mm)	1,500
Cutter length[1] (mm)	3,200
Overall height (m)	12.7
Delivery pipe (mm)	Ø 152
Weight[1] (t)	70

1 Customized for the Victor kimberlite deposit

A trench cutter comprises the operating, or cutting, part of the cutting system. The trench cutter or cutter head, is mounted to a base machine providing the hydraulic power to the cutter. A variety of base carriers is available for the cutter to be mounted upon. The selection of the base carrier is constrained by the following technical considerations [34]:

- Clear operating height;
- Surface area available for the cutter system; and
- Prospected or required cutting depth.

In addition to variations in the base carrier, the cutter can be suspended from that base carrier in multiple ways. The difference between the ways of suspension is related to the hose guide system. The mud hose and hydraulic hoses follow the cutter into the trench and must be kept at constant tension. A duty-cycle crane MC 128 is proposed as the carrier for the BC 50 trench cutter with the hose drum system (HDS), to accommodate for the desired cutting power and reach the desired cutting depth of 150 m. For depths such as the ones anticipated for mining purposes, the hoses are coiled on drums at the back of the base machine (Figure 5.1). The other guide systems (hose tensioning and hose synchronization) are limited to cutting depths that equal approximately 1.5 times the boom length. Table 5.2 summarizes the properties of the MC 128 Duty-cycle crane.

Besides the fact that the trench cutter system can be mounted on a crane it can also be installed on a barge. When on a barge, there are little to no changes to the actual capabilities of the cutter. However, putting the installation on a floating platform enables the cutter to not be restricted to dry deposits. Also deposits that occur on- or near-shore become of possible interest as well as deposits beneath a lake or open pit that is filled with water.

Table 5.2: Properties of MC 128 Duty-cycle crane [35]

	MC 128
Engine	CAT C 27
Engine Power (kW)	709
Main winches (kN)	2 x 350
Boom length (m)	54.4
Lifting capacity (max.) (t)	200
Operating weight (app.) (t)	170

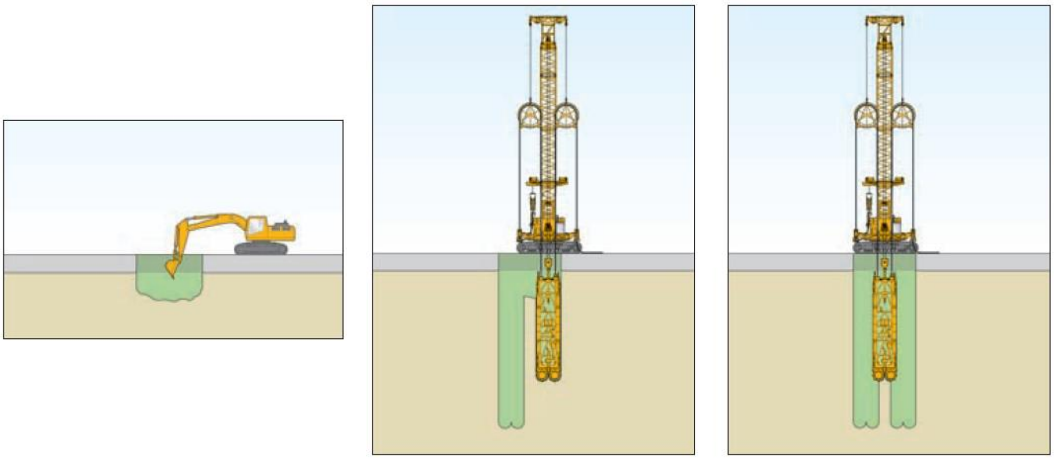
Trenches are filled with mud or slurry. Bentonite slurry, which is used as mud in civil engineering, has a higher density than water due to the colloidal mixture of bentonite clay and water. The mud provides stability to the trench walls as well as is the medium that transports the material that has been cut to surface. Without the presence of the slurry trench walls would collapse due to the hydraulic pressure in the surrounding soil. Bentonite slurry inside the trench counteracts the hydraulic pressure, keeping the trench walls stable. In general, the complete trench, from top to bottom is filled with the mud. For the mining application it is anticipated to use water instead of bentonite, as fluid inside the trenches. The water, naturally present beneath the open pit due to the lowered water table seeps into the trenches from the surrounding rock. When cutting starts, rock particles are suspended in the water, creating mud. The influence on the stability of the walls will be addressed during the stability assessment in the extraction scenario evaluation, Section 7.3.

Auxiliary equipment like a separation plant is required to account for the initial processing step and cleaning of the cutting mud. Desanding plants are used to remove rock or soil particles from the slurry. The separation plant is connected to the cutter via a series of hoses that directly transport the slurry from the working face to the separation unit. Standard desanding units provided by Bauer are relatively compact (container size) and mobile. It is expected that the desanding unit can be placed near the cutter inside, or next to, the pit. With an increasing number of cutters, combined desanding units are required to clean the slurry, and larger tanks will need to be installed outside the pit. This increases the complexity of the hose-pump system.

5.2 Standard cutting sequence

A trench cutter system is designed to operate as a continuous operation for the construction of diaphragm walls. The operating sequence of the cutter involves alternation between primary and secondary panels. A primary panel has the size of the cutter frame. A secondary panel, located in between two primary panels typically has half the size of a primary panel. The sequence, as it is described in Figure 5.3, is of great importance for the advance of the cutter. By leaving an intact rock pillar in place between two primary panels (Figure 5.3b), it is ensured that the cutter will follow a straight vertical path in those panels. In case this pillar would not be present, the cutter would divert into the first panel choosing the path of least resistance. After cutting two primary panels, the pillar in between can be excavated with a secondary cut. When cutting the secondary panel, the cutter is equally unconfined on both open sides, continuing to ensure vertical, straight advancement of the cutter.

Before cutting starts, pre-excavation and the construction of a guide wall need to take place (Figure 5.3a). The pre-excavated cut, or superficial cut, is commonly created in the first meters of unconsolidated soil and excavated with a backhoe excavator. A guide wall inside the superficial cut in turn ensures stability of the side walls when the soil is loose. The combination of the two is essential in order to prevent the cutter from swerving across the surface, but even more so to start the operation of the mud pump inside the cutter frame when the superficial cut is filled with water. In hard rock, like kimberlite, a guide wall is not required. The rock surrounding the superficial cut is strong enough to remain stable and not to cave into the excavation. Loosening of the rock before it can be excavated with a backhoe is anticipated to be executed by trench blasting. Narrow trenches with a width slightly bigger than the cutter are blasted in the kimberlite. Trench blasting, or ditch blasting is a blasting technique commonly used with the construction of pipelines [36]. Ditches for pipelines are created using blasting in a number of settings ranging from farming to urban locations. [37]



(a) (b) (c)

Figure 5.3: Cutting sequence of vertical cutter equipped with hose synchronization system [34].
(a) Initial process of trench construction: digging of superficial cut with back hoe excavator.
(b) Cutting of two adjacent primary panels with trench cutter (front view of cutter and panels).
(c) Cutting of secondary trench connecting the previously cut primary trenches.

In civil engineering, after the primary panel is constructed a reinforcement cage could be placed followed by concreting. These last two steps have no place in a mining sequence and will be disregarded.

5.3 Cutting performance

The cutting performance of the trench cutter is dictated by the strength of the rock to be cut. The weaker the rock becomes, the higher the performance of the cutter. Figure 5.4 demonstrates the cutting performances of the BC 50 based on experience from past civil engineering projects executed by Bauer. The cutting performances have been calculated based on an expected advance per hour within kimberlite rock of density 2.4 t/m^3.

The cutting performance as displayed in Figure 5.4, experiences clear influence from an increasing rock strength. Cutting performances in lower strength rock, up to approximately 20 MPa, are high at 75 t/h for the 70 tonne trench cutter. With increasing rock strength, the cutting performance decreases. A rock with intact strength within the range of 30 MPa to 50 MPa, the strength range in which kimberlites are expected to be, the performance varies between about 45 t/h and 30 t/h. Although rocks with strengths up to 200 MPa have been cut by Bauer, it is generally perceived that rocks with a strength over 100 MPa are unfavourable for commercial extraction.

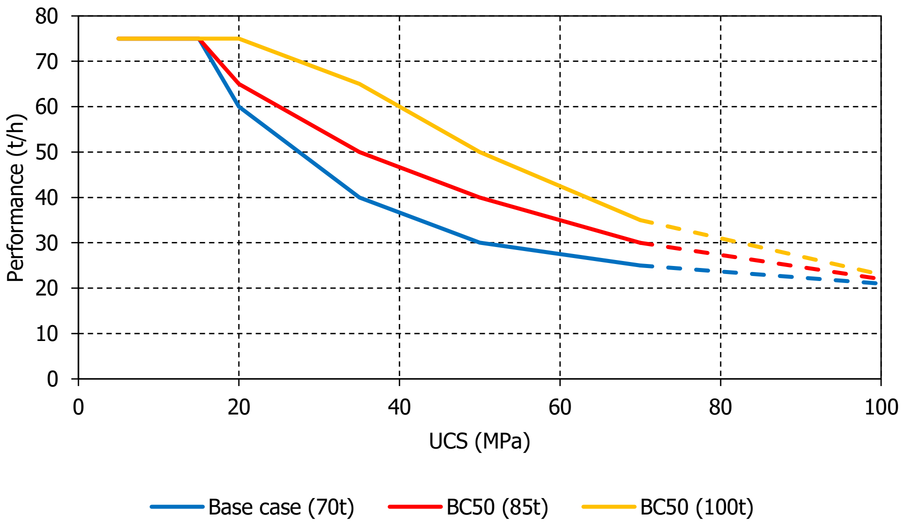


Figure 5.4: Cutting performance of BC 50 trench cutter, based on interpreted performance results from Bauer trench cutter systems from civil engineering projects and estimated performance of potential increase of the cutter weight. The cutting performance for rock stronger than 70 MPa has been extrapolated based on the results from the weaker rock. [Pers. Comm., Schwank, 2017a]

The cutting performance is, besides dependent on the strength of the material to be cut, influenced by the force that can be exerted by the cutter itself and the torque delivered by the cutter wheels. The force applied by the weight of the cutter can be increased by developing a heavier trench cutter. The base case cutter weight assumed for kimberlite extraction at Victor, already has increased from 50 tonnes to 70. As demonstrated in Figure 5.4, it is expected that with these modified cutters the performance will increase even more, especially when cutting rock of intermediate strength. Increasing the cutter weight is currently under review by the Bauer design department. It is expected that an increase of 15 tonnes from the current cutter weight (to 85 tonnes) can be achieved realistically. Comparing the performance of the base case cutter weight and potential developments it must be noted that the greatest increase of performance is located in the 25 MPa to 60 MPa rock strength range. Increasing the cutter weight does not significantly impact the cutting performance in rocks that are weaker than 25 MPa or stronger than 60 MPa. Increasing the cutting performance in stronger rock will require more improvements to the cutter besides a heavier cutter head.

5.4 Extraction scenarios

Trench cutting for civil engineering often requires the construction of one long, straight vertical trench for the construction of an underground retention wall. For the extraction of resources, the primary objective is to recover as much ore as possible at the greatest efficiency, while maintaining safe working conditions. In essence, multiple rows of trenches as created in civil engineering can be extracted next to each other in order to maximize recovery. This highlights the difference between the civil engineering industry and the mining industry: the volume of material produced.

By driving the vertical cutter system on its tracks inside the pit, vertical cutting is anticipated to be used as a mining method. The tracked base machines provide a high mobility to the system, allowing it to move around throughout the pit between different areas. It is assumed that the bedrock is levelled appropriately to ensure this mobility. By placing the cutter system directly above the to-be-extracted ore, the rock is extracted by cutting vertical cuts. Then, the ore is transported to surface via the mud hoses. After initial separation of fines in the desanding unit, pre-concentrated material can be hauled to the processing plant, where it is further treated. As a mining method, vertical cutting can operate by applying a total of three land based extraction scenarios, and one assuming cutting from a barge:

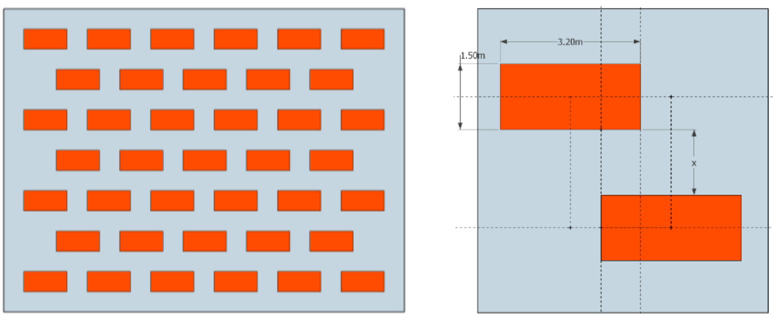
1. Checkerboard trench extraction;
2. Long trenching with remaining crown pillar;
3. Vertical cutting with backfill; and
4. Barge-mounted cutting after pit flooding.

The scenarios are listed according increasing recovery, with checkerboard extraction considered the base case and with the lowest recovery. The three other scenarios are variations of the checkerboard scenario. With the checkerboard scenario the ore deposit is partly extracted by creating a grid of cuts consisting only of primary panels (Figure 5.3a). Long trenching is a variation of checkerboard extraction where in addition to primary cuts also secondary cuts are extracted (Figure 5.3b). Vertical cutting with backfill and barge-mounted cutting achieve complete extraction of the deposit by also extracting the rock that is left behind in between two rows of cuts.

Suggestions have been made to apply a combination of clamshell and cutter mining. A clamshell has similar dimensions as a trench cutter (Figure 5.2). However, instead of being equipped with cutter wheels and mud pumps, it consists of a grab. A clamshell extraction scenarios requires unconsolidated or loosed rock material. The rock is than brought to surface by hoisting the grab up and emptying it in a haul truck. The design and applicability of extraction scenarios with a clamshell have not been investigated during this research.

5.4.1 Checkerboard trench extraction

The checkerboard scenario is considered to be the base case extraction sequence of the vertical cutter. The ore deposit is punctured with free, not-connecting trenches with the size of the cutter head. The checkerboard set-up is considered 'base case' because the other scenarios are extensions of the checkerboard, but also because of Bauer's experience and the greatest expected rock stability.



(a) (b)

Figure 5.5: Schematic overview of checkerboard scenario, (a) top view and (b) close-up top view

With the checkerboard scenario, extraction is limited to the cutting of primary cuts. A secondary cut, as described with the standard cutting sequence in Section 5.2, is not extracted. In addition to leaving a pillar in between two adjacent primary cuts, a crown pillar is left in between two rows of cuts. Both the pillar and crown pillar are anticipated to have a width half to the primary cut width, if stability assessment allows to do so. It is because of the interconnected pillars and crown pillars that the checkerboard scenario has the greatest expected rock stability.

The layout that has been designed for the checkerboard extraction scenario is displayed schematically in Figure 5.5a. A close up of two adjacent trenches is displayed in Figure 5.5b, including symmetry lines.

5.4.2 Long trenching

The second scenario that is considered is the long trenching method. With long trenching a continued trench extraction sequence is applied. The standard cutting sequence, discussed in Section 5.2, ensures continuous extraction of the rock by alternating between primary cuts and secondary cuts. With the long trenching scenario, primary and secondary cuts are connected until the maximum extents of the target are reached. After the extraction of one continuous trench, the cutting system retreats, leaving a single crown pillar, and starts extraction of the subsequent trench.

Figure 5.6a and b display a schematic overview of the top lay-out of the long trench scenario, primary trenches are displayed in orange, secondary trenches in green. Figure 5.6c and Figure 5.6d in turn display the symmetry from the top and the side respectively. The schematic layout for the long trenching scenario has been adapted from current civil engineering practice to create a single long interconnected trench, but place a multitude of such trenches behind one another.

Due to its favourable symmetry, the long trench scenario is considered the base case for stability modelling. Long trenches can be considered infinite in length due to their length to width and depth ratio. The side view, as displayed in Figure 5.6d, is, therefore, used in the stability assessment during the scenario evaluation (Section 7.3). The checkerboard scenario, which is expected to be sufficiently stable due to the rock bridges interconnecting the pillars and crown pillars is less favourable for 2D stability modelling.

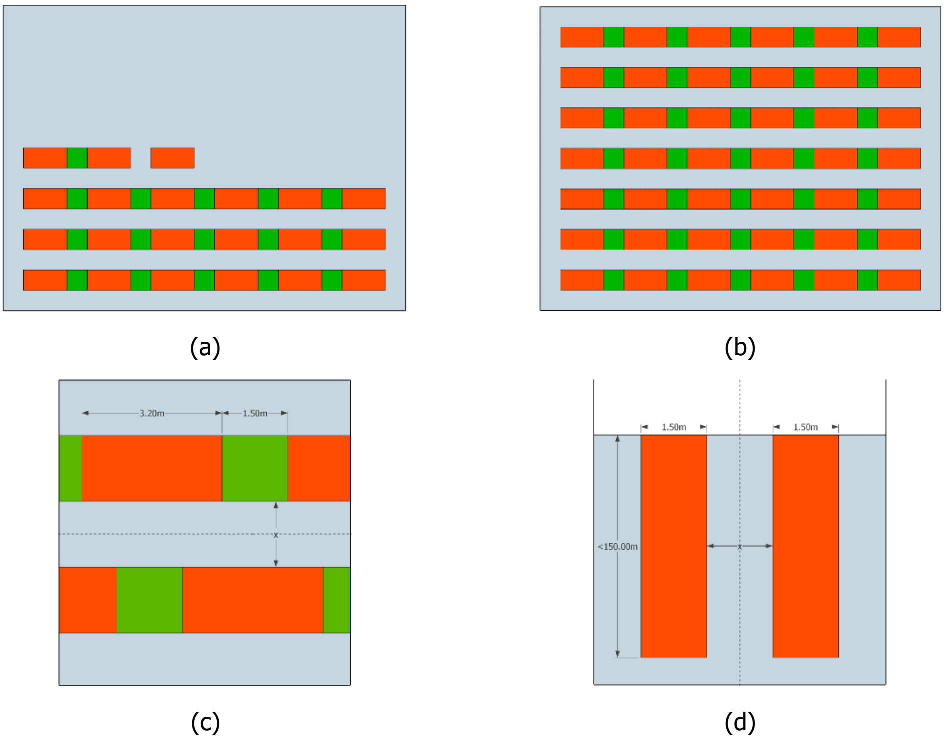


Figure 5.6: Schematic layout of the long-trench method, (a) top view of intermediate step in cutting sequence, (b) top view of complete long trenching layout, (c) close-up of top view, including symmetry line and (d) close-up side view, including symmetry line

5.4.3 Vertical cutting with backfill

Backfilling involves the filling of an already cut panel with waste rock or rock from a secondary source mixed with other components including water, fly ash or cement. With other mining methods backfill is commonly applied as a means of ground control to prevent caving or subsidence of surrounding rock into excavated underground stopes. With vertical cutting backfilling would serve the same purpose and prevent collapse of the surrounding rock into the excavated cuts. In essence, backfilling replaces the function of the bentonite slurry used in civil engineering applications after cutting has been completed.

The backfill scenario employs the same excavation sequence as the long trenching scenario. Initially long, interconnected trenches are created that are separated by a crown pillar. After completion of one trench, backfilling can start, retreating at the same retreat rate as the cutting system. After maximum extraction of the target according to the long trenching scenario the backfilled trenches will serve as a working platform, allowing extraction of the kimberlite left behind in the crown pillars. When applied as such the theoretical extraction rate would be near 100%.

Backfilling can be applied to the other scenarios as well. By filling the panels with backfill the stability of the trench and the surrounding rock can be ensured. If vertical cutting proves to cause instability in the ore zone or the already existing mine, backfill may be required to ensure safe working conditions.

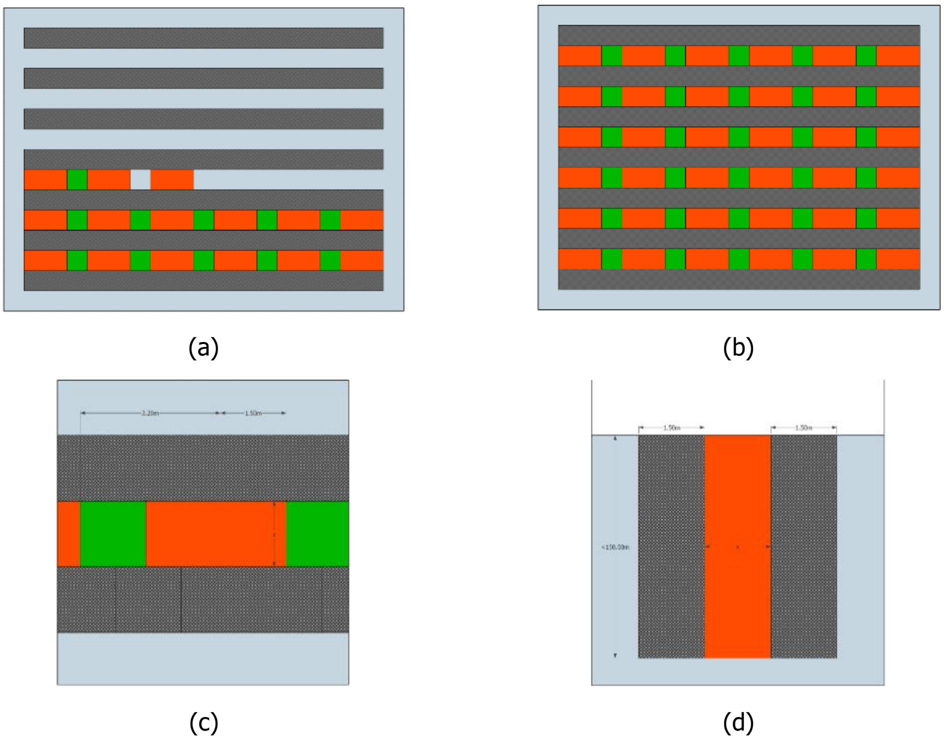


Figure 5.7: Schematic layout of the backfilling scenario, (a) top view of intermediate step in cutting sequence after trenches cut in phase 1 have been backfilled, (b) top view of complete backfill layout (c) close-up of top view and (d) close-up side view

Figure 5.7 display the schematic layout of the backfilling scenario without considering proper short term scheduling. The figure shows the second phase of extraction. The trenches of the first phase are extracted according the long trenching scenario (Figure 5.6) and already backfilled (displayed in grey). For the second phase, primary cuts are displayed in orange and secondary cuts in green. After completion of the extraction of the secondary phase trenches, they may be backfilled as well. Backfilled trenches serve as the working platform when returning for the ore left behind after phase 1.

5.4.4 Barge-mounted cutting after pit flooding

The final scenario includes flooding of the pit and extracting the ore using a vertical cutter mounted on a barge. Pit flooding entails a similar working method as the previous scenarios. Because there is no activity inside the pit, small scale instability is allowed to occur. As a result, the working sequence can be greatly optimized and no predefined retreat direction is required. Cuts will no longer be needed to be backfilled for total extraction.

This scenario would reach a theoretical extraction rate of 100%, similar to backfilling. Stability considerations and spatial availability of the pit that will define the previous scenarios, will become of less importance or change significantly due to water load exerted on the pit slopes. It is important to note that the cutting depth will be reduced by the depth of the water. Loss of material due to the rise of the water table is considered a disadvantage for deposits that continue into greater depths.

6

The Victor Kimberlite Deposit

The following chapter introduces the Victor Diamond mine. The Victor mine, which is part of the Attawapiskat kimberlite cluster, serves as an ideal case for the investigation of the applicability of vertical cutting. Chapter 4 has clearly expressed the need for alternative mining of its residual resources when open pit mining ends by the end of 2018, or early 2019. Particular attention will be paid to the geotechnical assessment of the kimberlite deposit for the estimation of the intact rock strengths.

6.1 General description and context

The Victor Diamond Mine (Victor mine) of De Beers Group Canada (De Beers) commenced mining operation in 2008. Construction at the mine site started in February 2006 and finished 6 months ahead of schedule by the end of 2007. In the first 3 years of production, mining focused on the high grade ore bodies in the main kimberlite pipe. The first open pit (cut 1) was planned based on the mineral resource block model in 2007 and had an annual production rate of 780,000 ct from treating 2.7 million tonnes of ore in 2011 [17]. Mine production continued in the Victor Southwest kimberlite pipe with cut 2 after 2011.

Located in northern Ontario, 90 km west of the community of Attawapiskat First Nation, the Victor mine is Ontario's first diamond mine. During its lifetime, the Victor mine has received wide recognition for its operations and diamond quality. Partly due to the excellent colour of the diamonds, the Victor mine has often been referred to as an exceptional mine. The mine has been reported to employ 420 people on site, giving priority to local, Aboriginal employment and development. [38]

However, not all effects of the mine are perceived as beneficial. Especially the isolated Attawapiskat First Nation community is concerned about receiving a fair share of the profit as well as potential environmental impacts of the mine on the surrounding waterways. [39]

According to the 2012 Business Plan, it is predicted that the mineral reserves of the Victor mine will be depleted by the end of 2018 [17]. Expansion of the pit, however, has been considered for a long time in order to reach diamond resources below the current cut (cut 2). Due to the absence of a buy-in from the surrounding Aboriginal communities potential expansion of the pit was forced to be halted. Options to delay pit closure including processing of lower grade ore, exploration of nearby kimberlites and vertical extension of the pit have been reported to be under investigation. [40]

6.2 Location and access

The Victor mine site is situated in the northern Ontario lowlands, west of James Bay. The closest community, Attawapiskat FN, is located approximately 90 km to the east of the Victor mine. The Universal Transverse Mercator coordinates for the centroid of the current Victor open pit (cut 2) are 305700E, 5856200N (NAD 83 Zone 17). De Beers own large areas of land surrounding the Victor mine with potential other diamond bearing kimberlites.

The Victor mine is a remote fly in/fly out location and comprises the only form of development on the north Ontario James Bay Lowlands besides the First Nation communities of Attawapiskat, Kashechewan, Fort Albany and Moosonee. The Lowlands are flat and covered by wetlands and peatlands, and is drained by numerous rivers and streams ending in the James Bay. Water drainage in the area of the Victor mine happens through the Attawapiskat River and its tributaries. [41]

There are no regular, all-weather, roads that connect the Victor mine to other infrastructure or communities. Access to the mine site is via Attawapiskat FN and Timmins, Ontario, on chartered commercial aviation [19]. During winter time, a winter road allows for the mine site to be reached by heavy haul trucks. Generally, the winter roads are open during a 2 month period ranging from late January to March. Loads and equipment are transported by railway to Moosonee, from where they continue over the winter roads to the mine site. [42]

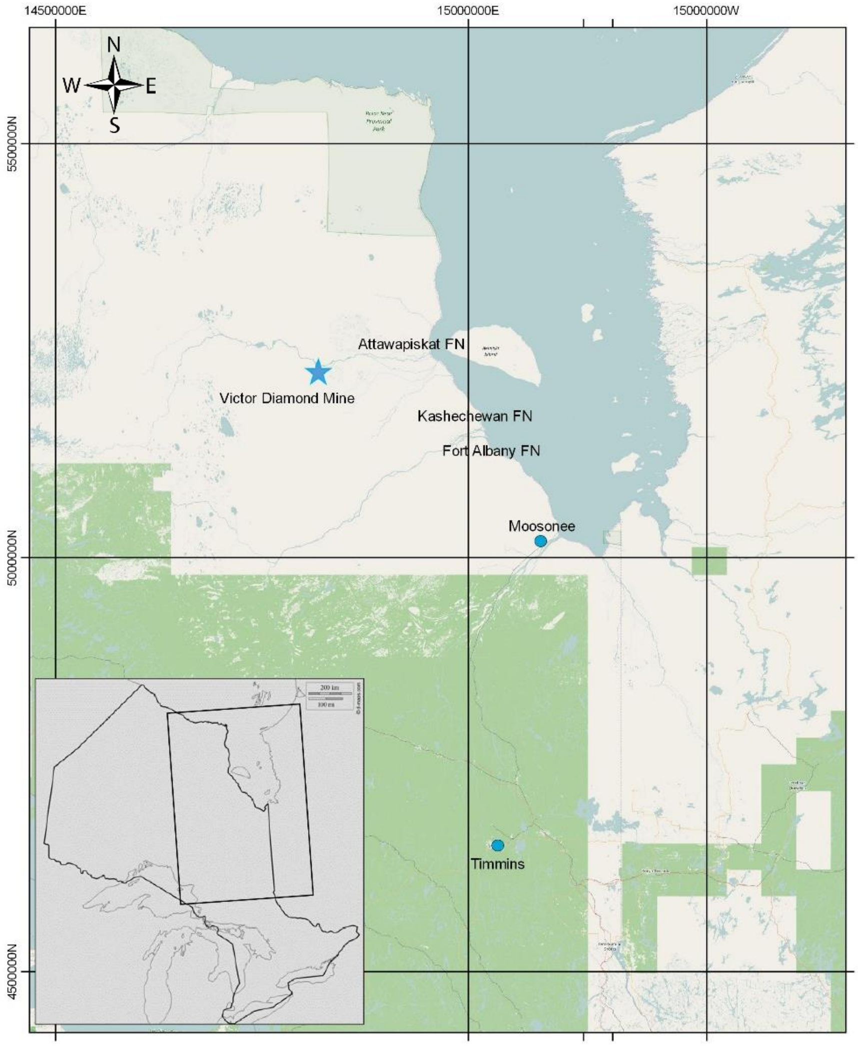


Figure 6.1: Site location, including First Nation reserves

6.3 Climate

The James Bay Lowlands are designated as the ecoregion that ranges from the James Bay shores in the west to the Attawapiskat River in the north. The Victor mine site is fully situated in the northern part of the ecoregion. The region experiences cool, short summers and cold winters. The eco-climate is described as humid high-boreal, influenced by the moderating effects of the Hudson Bay and James Bay during the summer. Winter is dominated by cold, dry air from the arctic as the Hudson Bay and James Bay are (partly) covered by ice. Temperature and precipitation are predominantly dependent on the proximity of a site to the Hudson Bay or James Bay. [43]

Regional climate data are collected in weather stations in Moosonee and Lansdowne House. They are located approximately 225 km southeast and 285 km west of the Victor mine respectively. The weather centres at Moosonee and Lansdowne House are the closest stations that collect long term climate data to the Victor site and are operated by Meteorological Service of Canada, Environment Canada. [43]

Since 2000, a meteorological station is operated at the Victor site. At Victor, wind speed and direction, temperature, relative humidity, net radiation, precipitation and snow depth are measured. Following two upgrades of the station since 2000 four parameters were added: barometric pressure, pan evaporation, solar radiation and heat flux. [43]

Figure 6.2 shows the monthly averages of the temperatures and the precipitation in Moosonee. The monthly averages of the temperatures and precipitation measured in Lansdowne House resemble the monthly averages of Moosonee closely. Lansdowne House measured slightly higher average precipitation during the summer months. It is expected the monthly averages at the Victor site are similar to the ones of Moosonee and Lansdowne House.

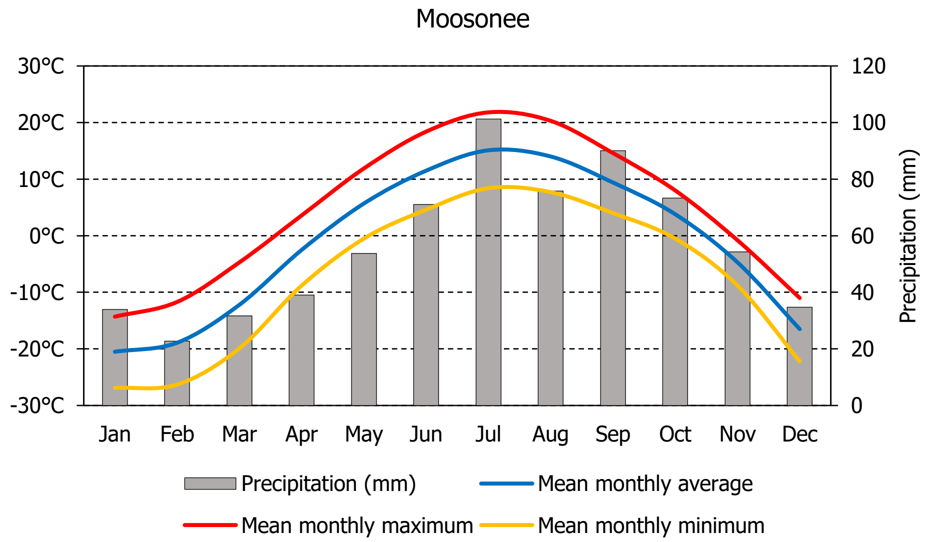


Figure 6.2: Average monthly precipitation, and maximum, minimum and average temperatures measured at Moosonee weather station [44]

The dominant wind direction measured at the Lansdowne House station experiences seasonal changes throughout the year. The wind originates most often from the west. Average wind speeds are reported to be fairly constant around 5 km per hour. For the Moosonee weather station at Moosonee Airport the dominant wind direction is also from the west, however, during spring time (March through July) the wind originates predominantly from the north. [45]

Climate, and more in particular extreme cold and snow, influence the project significantly. Equipment shipping is dependent on the availability of the winter road reaching the Victor mine site. In addition, equipment will need to be winterized and extra measures to provide a safe and comfortable working environment provided. In the James Bay ecoregion discontinuous permafrost is present. However, due to it being located outside the project area, it will not be discussed in this research. [43]

6.4 Victor geology

After discovery of the Victor pipe, it was found that the Victor kimberlite in fact consisted of two separate pipes, Victor North and Victor South. Based on a drilling campaign during 1998/1999 it was found that Victor North in itself also consisted of two different parts, termed Victor Main and Victor Northwest. Figure 6.3 displays the plane view of the Victor kimberlite pipes. The Victor Main pipe is shown in blue, Victor Northwest in yellow and Victor South in red.

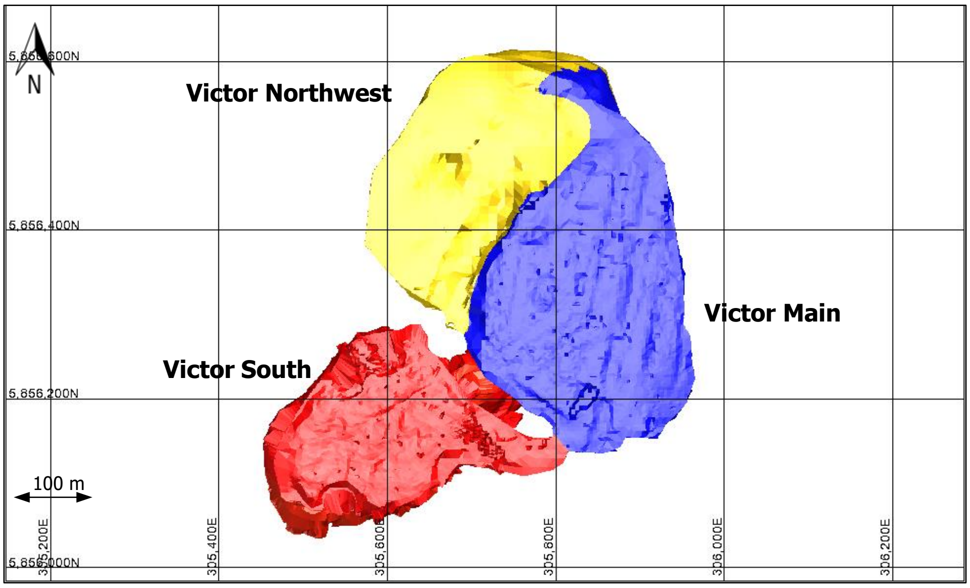


Figure 6.3: Plane view of the total geological model of the Victor Main (blue), Victor Northwest (yellow) and Victor South (red) kimberlite pipes

Although the two pipes at Victor occur as two distinct pipes at present day, their close proximity to each other and the slope of the pipe contacts, have led to believe that they united or cross-cut at higher levels which are no longer present [16]. Significant erosion has occurred over the course of geologic history, decreasing the size of the pipe to its present day shape [18]. Approximately half of the original pipe depth is still present.

Victor North and Victor South are mainly consistent of pyroclastic or volcanoclastic kimberlite and to a lesser extent hypabyssal kimberlites. Especially Victor South comprises pyroclastic kimberlites whereas Victor North, with its more complex interior, consists of both pyroclastic and hypabyssal kimberlite [16]. Tuffistic infill which is commonly associated with the diatremes in South African kimberlite pipes is absent at Victor.

The Victor kimberlite differs significantly from South African kimberlites in shape, infill and emplacement processes. It is suggested by Kong et al. [18] that the Victor pipes, like the other Attawapiskat kimberlites, formed in subaerial conditions by an overall two-stage process:

1. Pipe excavation without the development of a diatreme
2. Subsequent pipe infilling by primary pyroclastic air fall processes

The two-step pipe formation is, according Webb et al. [16], followed by a second phase of excavation and infilling. The initial pipe infill reportedly consisted of lower grade kimberlites. The second phase deposited high grade pyroclastic kimberlites from two adjacent vents in the crater that was excavated within the initial, low grade, diamond bearing kimberlite. The several eruption processes from adjacent, crosscutting and nested craters demonstrates the complex internal geology of the Victor pipes. The geological model shown in Figure 6.4 displays the 'grade' (diamond-bearing) kimberlites that are of interest for this study. The grade pipes are surrounded by additional kimberlite of 'no grade' which are not shown in Figure 6.4. The Victor Northwest pipe that was displayed in yellow in Figure 6.3 has no to a low grade and is not shown in Figure 6.4 as it is not considered a target for this study.

The shape and size of the Victor pipes resemble a Lac de Gras type kimberlite, however, Victor has primary pyroclastic kimberlite infill, whereas the Lac de Gras type kimberlites are generally infilled by resedimented pyroclastic kimberlite. Similarly, the two stage emplacement suggests close relation with kimberlites on the Canadian Prairie. However, the geological setting at Victor differs both from the Canadian Slave region and Prairies [16]. Although only showing selected parts of the complete Victor pipes, it can be seen in Figure 6.4 that remaining Victor pipes are neither broad nor shallow. The steep sided and relatively small size of the Victor pipes is of great interest for the development of vertical cutting as an extraction method. The vertical continuity of the pipes allows the operation to make optimal use of the cutting direction for maximal recovery.

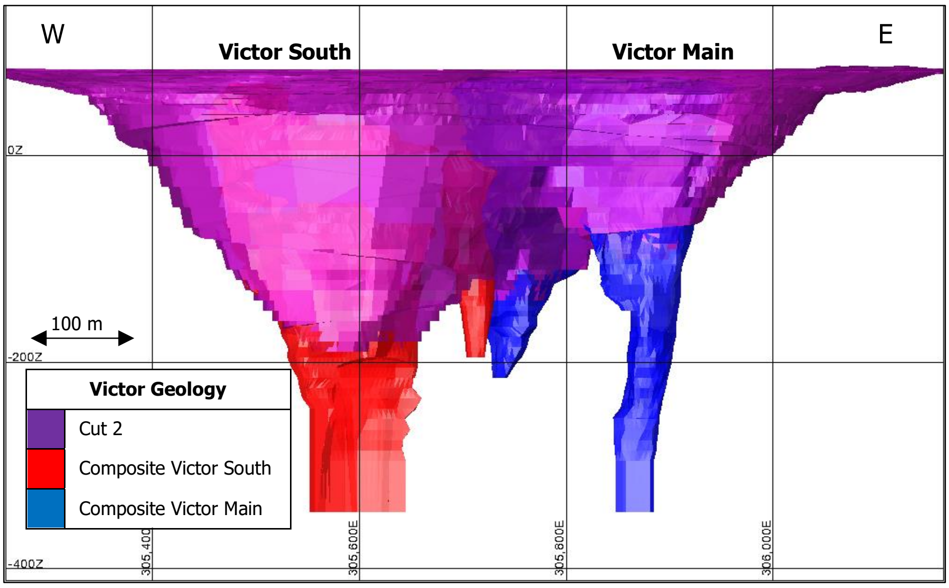


Figure 6.4: Section view of selected parts of the Victor kimberlite pipes, Victor Main and Victor South. Colour codes coincide with the codes used in Figure 6.3. Victor Northwest pipe not shown in the section view.

Within the kimberlite pipes a much more detailed distinction is made describing internal kimberlite facies. Webb et al. [16] defined the kimberlite facies according to the main textural types of the kimberlite rock for both Victor North and South. Based on later geological drilling the kimberlite facies were further defined in 2015 by Mann [46].

For the purpose of this study no distinction will be made between the kimberlite facies, except during spatial interpolation of the geotechnical rock properties in Paragraph 6.5.1 and the identification of the target zones. However, the rock characteristics (strength, grade, etc.) of kimberlite facies may differ between another.

6.5 Geotechnical assessment

The geotechnical properties of the rock units in and around the Victor deposit will have significant influence on the cutting performance of the vertical cutter equipment, as well as the stability of the trenches and the existing pit. Review of the available geotechnical parameters is therefore crucial in order to give a feasibility assessment. The following section will address the data analysis of the geotechnical parameters of the Victor kimberlite pipes.

A total of three datasets regarding the rock strength of the kimberlite at the Victor deposit have been made available. The first of which comprises geological logs that have also been logged for geotechnical data. The drill cores give insight in both the strength of the intact rock as well as core recovery and rock quality. The second set consists of UCS sample measurements during the 2003 geotechnical feasibility study [24] and the 2014 stability assessment [47]. Samples taken during these stability assessments aimed to determine the rock strength of the kimberlite inside cut 1 and cut 2 respectively. The last set contains point load test measurements. After data analysis of the individual data sets, Paragraph 6.5.4 will attempt to combine all data and determine an average intact rock strength. In addition to the strength analysis, potential degradation of the kimberlite due to weathering or exposure to water was assessed.

6.5.1 Geotechnical core logging

De Beers have provided core logging data of a total of 39 drill holes. For the drill holes the geology and geotechnical logging data were reported with an accuracy of 1.5 m intervals (exceptions exist). The geotechnical parameters that are of interest consist of:

- Core recovery parameters (total recovery, solid core recovery and rock quality designation)
- Intact rock strength field identification
- Number of discontinuities
- Number of joints and their conditions
- Alteration level

It must be noted that not for all drill holes these parameters were reported, or reported with the same accuracy during field logging. Only for the boreholes drilled in 2016 the RMR^{76} was calculated based on the mentioned parameters.

Cutting performance of the vertical cutter is reported to be highly dependent on the intact rock strength or UCS. Determining the intact rock strength in the field is a timely and costly effort. During geotechnical drilling rock samples are classified using a strength index, displayed in Table 6.1. [48]

Table 6.1: Field estimation of rock strength (modified from ISRM [48])

Grade	Description	Field Identification	Approx. Range of UCS (MPa)
R0	Extremely weak rock	Indented by thumbnail	0.25 – 1.0
R1	Very weak rock	Crumbles under firm blows with point of geological hammer, can be peeled by a pocket knife	1.0 – 5.0
R2	Weak rock	Can be peeled by a pocket knife with difficulty, shallow indentations made by frim blow with point of geological hammer	5.0 – 25
R3	Medium strong rock	Cannot be scraped or peeled with pocket knife, specimen can be fractured with single blow of geological hammer	25 – 50
R4	Strong rock	Specimen requires more than one blow of geological hammer to fracture it	50 – 100
R5	Very strong rock	Specimen requires many blows of geological hammer to fracture it	100 – 250
R6	Extremely strong rock	Specimen can only be chipped with geological hammer	>250

Core logging provides a limited amount of data to assess the complete strength of the ore body. It is assessed that within one kimberlite facies the strength is fairly consistent within one strength index class. The spatial distribution of the intact rock strength classes can be estimated by using spatial geostatistics on the geotechnical drill hole data. Drill hole data provided by De Beers. Inverse distance interpolation within a kimberlite facies provides a reasonably accurate distribution along the expectations. Spatial interpolation of the core data was performed in Geovia's Surpac. In Surpac, the spatial distribution of the intact rock strength was interpolated by creating a block model spanning the complete Victor geology. The block user block size was chosen to be 3x3x3 m, in order to create a reasonably detailed model while compositing two subsequent intact rock strength measurements. Figure 6.5 displays the spatial distribution of the kimberlite intact rock strength according inverse distance interpolation with power 2. Increasing the interpolation power had very limited effect on the distribution. Nearest neighbour interpolation displays sharp boundaries of intact rock strength classes within one facies. Due to limited knowledge about the geological consistency of the intact rock strength advanced spatial interpolation models have not been considered.

Rock strength indices were interpolated within kimberlite facies, if the data density allowed to do so. No data from geotechnical drilling was available for the OPK4 kimberlite pipe in the centre of the Victor mine model, displayed in light pink.

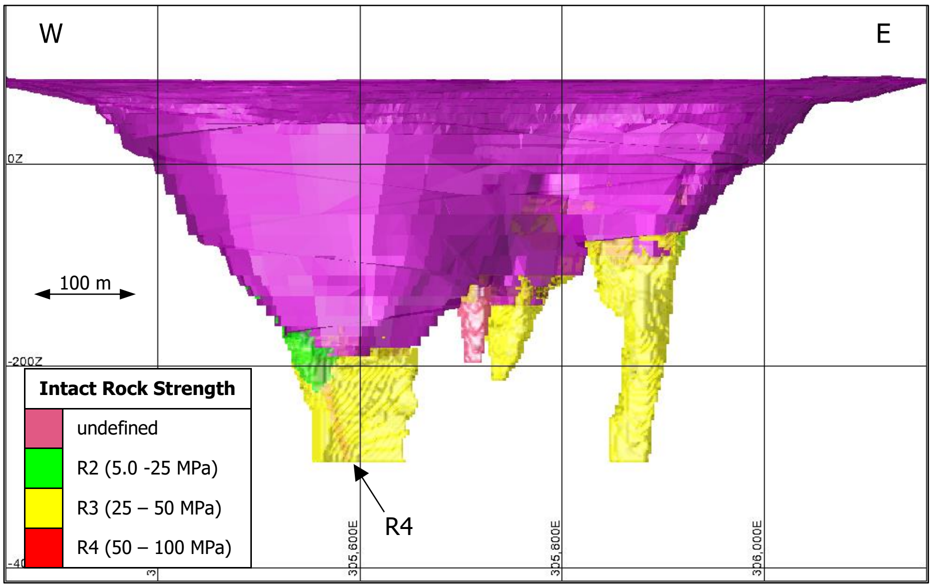


Figure 6.5: Spatial distribution of field strengths for all target zones, section W-E looking north. Distribution created with inverse distance (power 2) interpolation

Figure 6.5 indicates that the large majority of the kimberlites are classified by the geotechnical drilling as medium strong rock with intact rock strength index of R3. This implies that most kimberlites are expected to have a rock strength between 25 MPa and 50 MPa. This observation can be substantiated by data analysis of the core log data and comparison to other kimberlite deposits [26, 49]. Pockets of kimberlite with strength R2 and R4 are encountered sporadically throughout the remaining kimberlite. As is lead to believe by the spatial interpolation of the strength data within a kimberlite facies, to specific facies has a characterising strength different than R3. Kimberlites with the greatest strength is located on the northern side of the Victor South pipe. The location of the near-vertical band of R4 kimberlite is indicated by the arrow in Figure 6.5. No kimberlite weaker than grade R2 or stronger than grade R4 have been observed during geotechnical drilling in targeted the kimberlite. Weak kimberlites have been present but were excavated in the previous cuts of the open pit.

Although the cutting performance estimations are predominantly based on the rock strength, it is desired to quantify the general quality of the rock. Based on the logged core samples the rock quality designation can be calculated according [50]:

$$RQD(\%) = \frac{\sum \text{length of core in pieces} > 10 \text{ cm length}}{\text{length of core run}} \times 100 \qquad (6.1)$$

The spatial distribution of the RQD was determined similarly to the spatial distribution of the intact rock strength shown in Figure 6.5. For spatial interpolation the same block model and block dimensions were applied. Also for the inverse distance interpolation with power two of the rock quality it appeared that other spatial interpolation methods did not provide reasonably better distributions within the kimberlite facies.

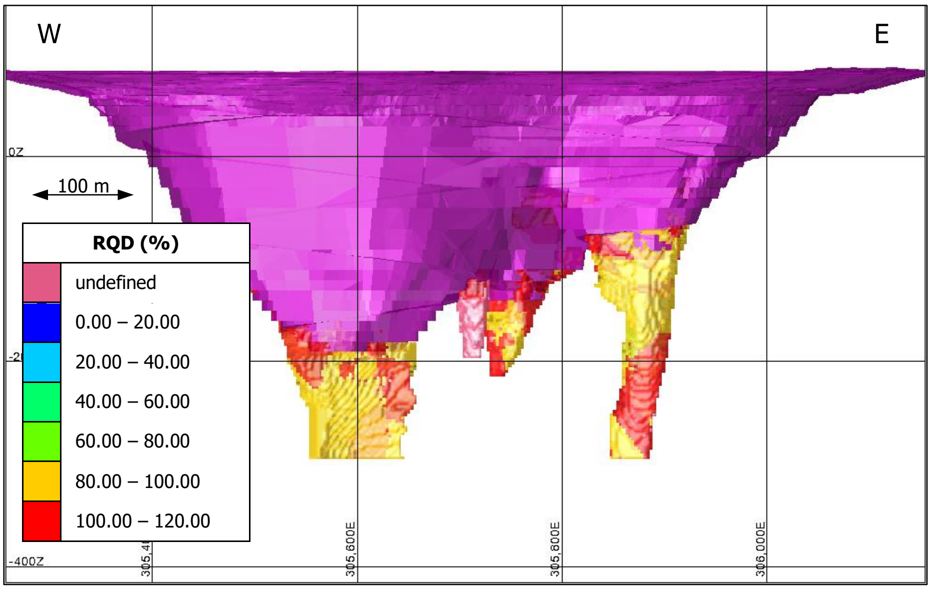


Figure 6.6: Spatial distribution of RQD (%) for all target zones, section W-E looking north. Distribution created with inverse distance (power 2) interpolation

The vast majority of the kimberlites have a high calculated RQD and can be classified as good to excellent rock. No significant jointing is therefore expected in the kimberlite. Virtually all rock has an RQD value higher than 80%. Few pockets of kimberlite with RQD between 60 and 80% have been logged near the contact zone between kimberlite and country rock. The conclusions of the spatial interpolation of the RQD data are substantiated by observations in the kimberlite rock present in the current excavations. Most of the current slopes have been designed inside country rock like limestone. However, in particular areas (Domain VI) slopes are created in kimberlite. Due to excellent rock conditions benches created in the kimberlite can have bench face angles of up to 82° and heights from 10 m to 20 m ("double bench") [25]. It is not expected rock conditions will deteriorate with increasing depth. Risk of failure inside the trench walls and pillars will increase significantly with deterioration of the rock quality.

6.5.2 UCS measurements

Over the course of the past years, 29 UCS tests on kimberlite samples have been collected. The samples originate from different drilling campaigns: one campaign was executed in 2002/2003 as part of the geotechnical (pre)feasibility assessment (7 samples) by SRK [24]. The other samples were tested in 2014 as part of a combined testing program involving both handheld drilling in blasted rock and geotechnical drilling [47]. The handheld drilling was performed using an anchored stand in blast heaps of kimberlite (Domain VI), the geotechnical drilling in the alteration halo zone (Domain IVa). None of the samples are located directly inside the kimberlite that is of interest for vertical cutting (Section 7.2), but expected geological consistency of the deposit may allow for the distribution of the test values to be extrapolated downwards into the target area.

Data analysis of the measured UCS values of the kimberlite samples renders the frequency histogram in Figure 6.7.

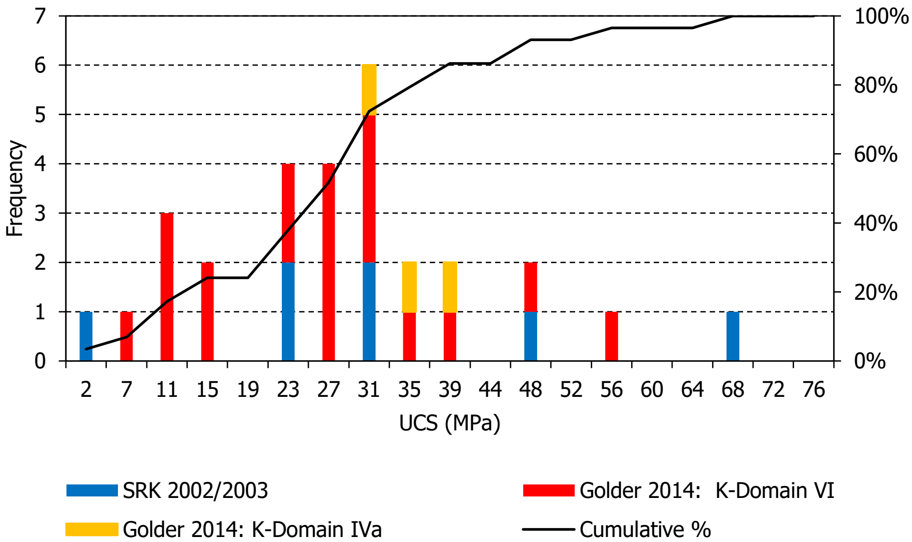


Figure 6.7: Frequency histogram of UCS measurement results

Golder Associates [47] reported that the UCS values of the handheld drilling (K-Domain VI) were notably lower than the UCS values obtained during the geotechnical drilling of 2003 and 2014. Handheld drilling is done in already blasted rock heaps. Micro-fissures or defects may have formed during blasting, causing the rock to have lower strength. Overall it was observed that the UCS results of the samples obtained with handheld drilling were 30% to 50% lower than the samples from the geotechnical coring or the UCS data from earlier investigations. In addition, they reported that the tests in the Alteration Halo Zone were not considered representative of the domain and were only used as an upper-bound strength for the Alteration Halo.

The average UCS values are reported as follows:

- SRK geotechnical (pre)feasibility (2002/2003): 29.4 MPa
- Golder Rock Strength Assessment (2014), Kimberlite: 24.1 MPa
- Golder Rock Strength Assessment (2014), Alteration Halo: 34.3 MPa

Adjusting the data in line with the noted issues by Golder with the 2014 data, the average UCS of the kimberlite is determined by the measurements performed in 2002/2003. Omitting the data from the 2014 geotechnical study suggests an average UCS of 29.4 MPa, based on 7 samples.

6.5.3 Point load test data

A large amount of point load test data was made available. A total of 3122 samples were tested regarding failure load in a point load test. All samples originate from a total of 32 holes, 19 of which contain kimberlite samples. The total number of kimberlite samples is 2161, only these samples are used to determine a strength distribution. For vertical cutting, data of other rock, like the intact rock strength of granites, limestones or mudstones surrounding the kimberlite pipe, is not considered to be of interest.

The UCS can be obtained from the failure load (P) measured during a point load test using:

$$Is_{50} = \frac{P}{De^2} \times F \qquad (6.2)$$

in which the Is_{50} the point load strength index for a core of standard size, P the failure load, De the equivalent diameter and F the size correction factor. The standard core diameter for a point load test has been determined to be 50 mm. All samples delivered in the mentioned set have a diameter of 63 mm, with minor deviations. In order to calculate the standard point load strength the correction factor F is calculated by:

$$F = \left(\frac{De}{50}\right)^{0.45} \tag{6.3}$$

In order to calculate the UCS based on the point load strength index, Is_{50}, it is multiplied by a conversion factor, K. K is dependent on the rock that is being investigated and highly variable. For this study it is assumed that K varies between 14 and 24 [51]. The exact value of K is not known for the Victor kimberlites. A conservative estimate of the strength regarding cutting performance, assumes K to be maximum and therefore 24:

$$UCS = Is_{50} \times K \tag{6.4}$$

The delivered PLT dataset is compiled from four different smaller datasets (EX2001-DLN, EX2002-GTECH, EX2002-DLN and EX2003-GTECH), collected from different years. All data reported consists of the following information:

- Dataset name
- Hole ID
- Test No
- Rock unit (lithology)
- Sample depth
- Sample dimensions (width and diameter)
- Failure load (kN)

In addition to the information provided in the list provided above, the date and person who logged and loaded the data was included in some cases.

The reported data is not consistent for all of the collected datasets. In particular dataset EX2001-DLN appears to be different from the other sets. For EX2001-DLN, the failure load has not been reported in kilonewton (kN). It is assumed that the unit is Newton. Basic data analysis clearly demonstrates the difference between dataset EX2001-DLN and the other sets (Figure 6.8 and Figure 6.9).

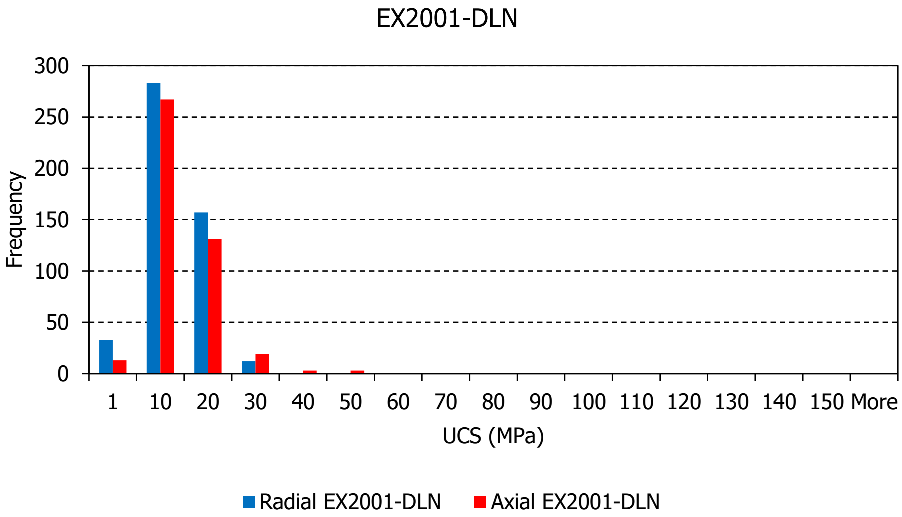


Figure 6.8: Frequency histogram of UCS values based on PLT measurements for dataset EX2001-DLN, K = 24

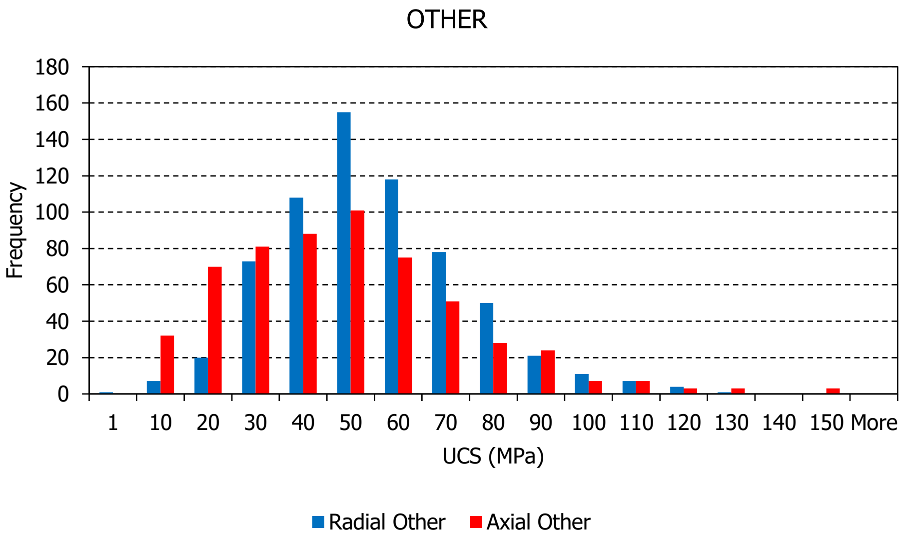


Figure 6.9: Frequency histogram of UCS values based on PLT measurements for datasets EX2002-GTECH, EX2002-DLN and EX2003-GTECH, K = 24

It has not been determined why there is a big difference between EX2001-DLN and the other sets. However, based on experience in the Victor mine, it is assumed that the point load tests for EX2001-DLN have been performed immediately after core recovery, on top of the core box. There may have been procedural issues that cause the difference in measured strength values.

For the other sets (EX2002-GTECH, EX2002-DLN and EX2003-GTECH) a number of very high strengths have been calculated, potentially indicating that the conversion factor K = 24 may be too high.

Within the Victor kimberlites it has been reported that there can be a high percentage of internal dilution due to the presence of xenoliths. In order to rule out whether the failure load of some point load samples may have been influenced by the presence of xenoliths, the full geology log has been compared to the point load test samples. In some cases, it has been found that the kimberlite shows a sudden increase in strength when in the near vicinity of a limestone xenolith (<0.1 m). However, it could not be determined that there is a consistent trend throughout a number of drillholes that supports this observation.

Strength – depth relation

Based on the point load data it has been attempted to determine whether a trend is present that relates an increase of strength to the depth. In order to do so point load strength index has been plotted against the true depth (Figure 6.10 and Figure 6.11).

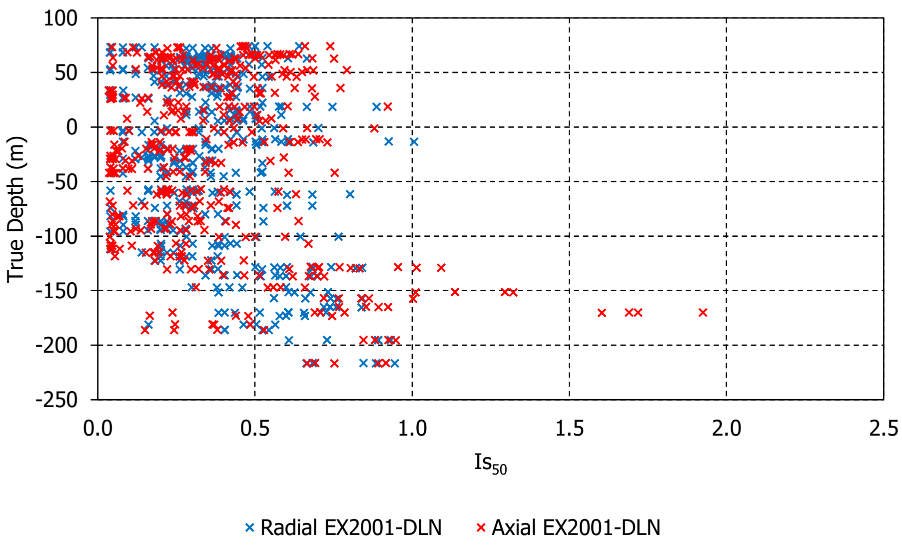


Figure 6.10: Depth vs Is_{50} plot based on radial and axial PLT tests for EX2001-DLN

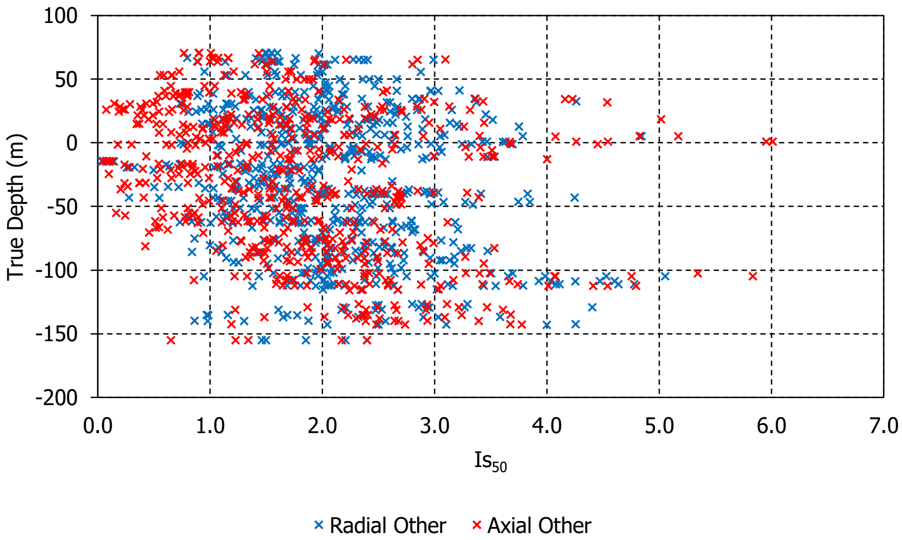


Figure 6.11: Depth vs Is_{50} plot based on radial and axial PLT test for EX2002-GTECH, EX2002-DLN and EX2003-GTECH

Based on the plots displayed in Figure 6.10 and Figure 6.11, there is no specific trend regarding the increase of rock strength related to the depth. However, the rocks with the greatest strength do occur deeper in the hole rather than shallower. In general, one may conclude that there is an increase in strength when the depth becomes greater, but a broad range of strength measurements remains. It appears the lower bound for the point load strength results increases slightly with depth, especially from depths greater than 150m, but the higher bound may take a range of results.

(An-)isotropy of the rock strength

Figure 6.8 and Figure 6.9, as well as Figure 6.10 and Figure 6.11, show the point load test results for both the axial and radial tests separately. Although the two tests show a slightly different distributions, no specific differences can be determined between the axial and radial point load strength for one sample. The rock strength is, therefore, considered isotropic and no further distinction will be made between axially and radially measured strength.

6.5.4 Strength analysis

Internal review of the geotechnical properties of the Victor kimberlite by De Beers has concluded that the intact rock strength of the deep kimberlites at Victor shows a considerable increase in the target zones. The observations done during spring 2017 differ from the strengths suggested by the data presented in the earlier paragraphs and will be discussed separately.

Intact rock strength prior to 2017

In summary, a total of 3 data types have been made available:

1. Geotechnical logging, including core recovery and intact strength classification
2. UCS measurements (29) collected both in situ and from blast heaps
3. Point load data (2161)

The first set is relevant to demonstrate the spatial distribution of the strength zones throughout the model. The classes that have been assigned, however, are considered very generic and to some extent more detail is desired. In general, the strength of the kimberlite is considered fairly consistent throughout the deposit with strengths varying between 25 MPa and 50 MPa with the exception of some kimberlite in the Victor Southwest pipe, which exhibits intact rock strength ranging between 50 MPa and 100 MPa.

The UCS measurements provide the only data of actual performed UCS testing. However, the 2014 data is considered highly inaccurate due to samples being taken from blast heaps and the kimberlite alteration halo being highly degraded and fractured. The UCS results from the 2014 rock strength assessment are therefore disregarded. The UCS results from the 2002/2003 drilling program are considered reliable and indicate an average UCS of 30 MPa.

Because of some discrepancies within the point load sets, and a calculated UCS based on an assumed conversion factor K, the stand-alone accuracy of this set is not considered to be high. The set demonstrates the variability of the rock strength within the kimberlite. When related to more other data, the accuracy of this dataset may increase. With minor adjustments to the assumptions the point load data may provide the most appropriate indication of the intact rock strength distribution at Victor.

A comparison of data sets 2 and 3 results in a better indication of the conversion factor K. Samples from the 2002/2003 geotechnical feasibility study and the point load dataset contain the same drillholes, including 4 samples at a similar hole depth. Table 6.2 shows all 7 UCS results of the 2002/2003 drilling campaign and the calculated UCS from the PLT data for sample of the same borehole and similar depth.

Table 6.2: Comparison of measured UCS and calculated UCS at similar hole depths

	Measured UCS		**UCS from PLT**[1]	
Hole ID	Sample Depth	UCS (MPa)	Sample Depth	UCS (MPa)
V-03-315C	240.2	43.8	N.A.	N.A.
V-03-315C	240.8	64.7	N.A.	N.A.
V-03-318C	166.6	20.4	166.5	67.1
V-03-318C	167.9	20.7	168.3	44.95
V-03-319C	105.7	2.3	N.A.	N.A.
V-03-319C	135	27.1	135.5	50.05[2]
V-03-319C	135.7	27.1	136.2	15.62

1 K ratio of 24 used for all samples (Is_{50} x 24 = UCS)

2 Average calculated UCS of axial and radial test of sample at 135.5 m.

One may expect that kimberlites within a single kimberlite facies and within close proximity to each other have a similar UCS value. The UCS values from the PLT measurements in Paragraph 6.5.3 are calculated by multiplying the point load index strength with a conversion factor K (equation 6.4). Initially the conversion factor was set at its maximum literature value (K = 24), giving the most conservative estimate in regards to production performance. The appropriate value of K is determined based on the comparison of UCS measurements, which are considered the most accurate, and the calculated UCS values from PLT measurements. The comparison in Table 6.2, based on 4 samples, indicates that a conversion factor of 24 is too great, and would be rather half. Assuming no strength anisotropy and the absence of an significant strength to depth relation, the average UCS for the data sets without EX2001-DLN becomes 25 MPa, using a theoretical conversion factor K = 13.

The comparison made above cannot be expected to be representative for the complete pipe, however, it clearly demonstrates that a lower conversion factor may contribute to a good correlation between the measured UCS data and the point load test data. Conversion factor K = 20 has been used by Golder Associates in the 2014 geotechnical assessment of the kimberlites at the nearby Tango pipe [52]. With this conversion factor the average UCS of the "other" datasets (EX2002-GTECH, EX2002-DLN and EX2003-GTECH) becomes 39 MPa (axial and radial samples combined) with a standard deviation 18.5, closely corresponding to the distribution of the intact rock strength from the core logging set. The strength distribution obtained with K = 20 will be used throughout the rest of this research.

Updated intact rock strength analysis

In June 2017 it was suggested during internal review by De Beers that the intact rock strengths at the bottom of cut 2 and below the designed pit are greater than initially anticipated [Pers. Comm., Qureshi, 2017]. The estimations are substantiated by 6 UCS measurements, summarized in Table 6.3.

Table 6.3: Updated intact rock strength results from Victor Deep

Sample	Diameter (mm)	Length (mm)	Mass (g)	Density (g/cm^3)	Peak Load (kN)	Peak Strength (MPa)
V-14-587C_01.249m	63.28	129.66	1034.90	2.54	244.01	77.6
V-14-587C_02.257m	63.27	130.34	991.93	2.42	148.99	47.4
V-14-587C_03.270m	63.31	129.85	1038.74	2.54	338.53	107.5
V-14-587C_04.281m	63.34	129.83	1079.53	2.64	343.05	108.9
V-14-587C_05.291m	63.31	130.02	1034.24	2.53	349.98	111.2
V-14-587C_12.306m	63.33	130.18	1082.58	2.64	434.43	137.9

The recent strength results show that the average intact rock strength of the deep kimberlite is 100 MPa, with 4 samples significantly above the average value. The majority of this research will assume the originally determined average UCS of 39 MPa. In following chapter the influence of an increased intact rock strength of 100 MPa will be assessed if relevant.

6.5.5 Susceptibility to water

Tests have been performed to investigate the susceptibility of especially the softer kimberlites to water. Clayey kimberlites which generally also have lower UCS readings may exhibit rock degradation when in contact with water. Three tests were suggested to be performed to evaluate the decomposition of the kimberlite:

1. Testing according to Standard DIN 4022 Part 1
2. Vicat apparatus, testing according to EN 196-3
3. UCS values of cores prior and after underwater storage

Results of the tests suggest the kimberlite from the Victor pipe does not decay due to exposure to water. Table 6.4 summarizes the test results according Standard DIN 4022 Part 1. Kimberlite samples were mounted in a steel cage and stored under water. Kimberlite samples that decompose due to long term exposure to water fall outside the cage. The samples are weighed after predefined time intervals while submerged: at the beginning of the testing, after one hour and after 24 hours. Before and after testing, the sample is weighed without the influence of buoyancy. None of the pyroclastic kimberlite samples displayed decay after 24 hours.

Table 6.4: Test results on volcanoclastic kimberlite according to Standard DIN 4022 Part 1

Drill core	**Depth from (m)**	**Depth to (m)**	**dry weight in (kg)**	**wet weight at beginning (kg)**	**wet weight after one hour (kg)**	**wet weight after 24 hrs (kg)**	**wet weight out (kg)**
V-16-621	8.9	9.08	*N.A.*	0.7578	0.7641	0.7873	1.3379
V-16-622	29.48	29.67	1.3706	0.8027	0.8117	0.8551	1.4282
V-16-623	48.8	49	1.6537	0.971	1.002	1.012	1.6998
V-16-624	74.55	74.75	1.5053	0.9237	0.9282	0.935	1.53
V-16-625	90.3	90.5	1.656	1.0155	1.0158	1.0166	1.6723
V-16-626	106.57	106.74	1.2652	0.7679	0.7732	0.7882	1.2921
V-16-627	124.48	125	1.6624	1.0564	1.0592	1.0633	1.6735
V-16-628	154.8	155	1.7183	1.0877	1.0876	1.087	1.7267
V-16-629	189.42	189.68	2.0743	1.3754	1.3754	1.3764	2.0795

7

Victor Mine Life Extension

Assessing the viability of extending the mine life of the Victor Diamond Mine with vertical cutting technology requires extensive evaluation of the technical constraints. Available space in the mine, rock stability and appropriate scheduling all affect the efficiency of the cutter system. Maximizing that efficiency while minimizing risk and impact is the ultimate objective.

This chapter addresses the evaluation of the extraction scenarios and a preliminary task schedule for vertical cutter mining. Evaluation will be performed for specific targets within the deposit. The scenario(s) that achieves the most efficient extraction of the kimberlite will be sought after. No evaluation based on costs will be made. All cost considerations that are applicable within the scope of this research will be performed in Chapter 9.

7.1 Expansion and production capacity

Vertical cutting is specifically chosen to extend the life of the current open pit without large scale movement of additional material and overburden. The purpose of a continued production time is to sustain continuous output of the highly valued diamonds originating from the Victor Diamond mine. The estimated costs (financially and otherwise) and effort associated with mine closure and future reopening of the pit surmount the value significantly. It is therefore desired that the kimberlite resources present at the Victor location are fully exploited.

The total quantity of kimberlite ore processed in 2014 was 3.2 million tonnes, at an average mill processing rate of 8,963 tonnes per day [53]. Application of vertical cutting as a mining method, will decrease the total daily production and is more likely to be in the order of magnitude of 600 tonnes per day (16 hours per day).

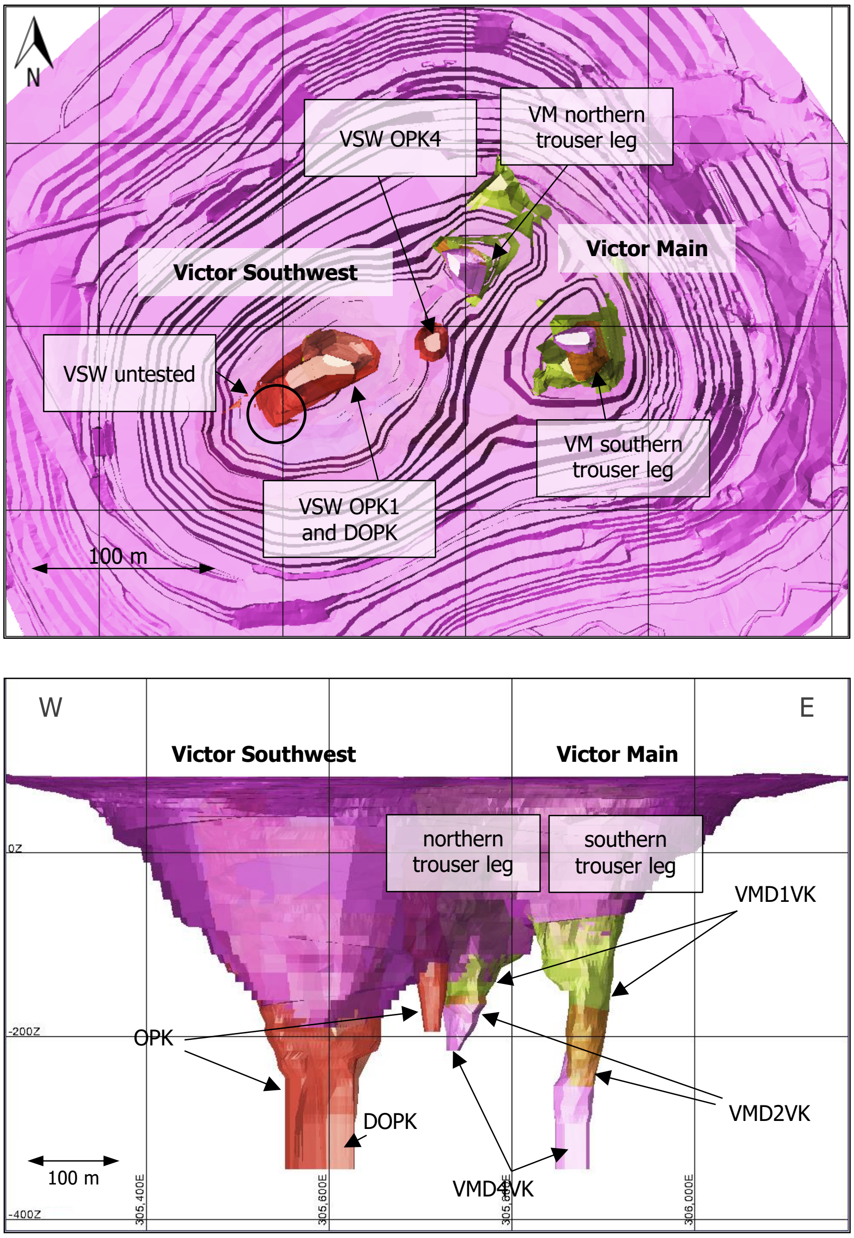
Based on the reported cutting performance graph and the average rock strength of 39 MPa, a cutting speed of 38.5 t/h can be achieved at the Victor mine, applying the BC 50 cutter without modifications. The true cutting performance is expected to divert from the average cutting performance, as the performance is determined by the rock with the greatest intact rock strength being cut, rather than an average strength.

7.2 Target identification

The Victor pit has been in operation since 2008. Following the inability to obtain buy-in from the surrounding communities, pit expansion will not occur in the near future. According to the 2012 Business case, operations in the Victor mine will cease by the end of 2018 or early 2019 [17]. However, below the current pit, cut 2, there are still resources present that may provide to be of interest for extraction.

Vertical cutter mining may provide the solution to extend recovery of diamondiferous kimberlites below the current pit limits. A total of 5 target areas have been identified by De Beers to be of interest for extraction with a vertical cutting system. The targets consist of the northern trouser leg of Victor Main, as well as the southern trouser leg of Victor Main, and three targets in Victor South. The information density in all targets is limited to a handful of delineation holes that cross the target at a limited interval, and a preliminary grade estimation. The location of all targets have been shown in Figure 7.1, including kimberlite facies within the targets.

(a)



(b)

Figure 7.1: Identification of the target zones below cut 2 of the Victor Diamond mine, including rock units, (a) top view, (b) section W-E looking north

For this research the targets are subject to one additional constraint. Kimberlite will only be considered to be of interest for extraction with vertical cutting if the designed or present open pit remains unchanged. Hence, the kimberlite that is located within the slope of the pit will not be considered a target for vertical cutting. Only the kimberlite bodies that are considered targets by De Beers, are not situated inside the pit slope, and are below cut 2 are of interest for future extraction by vertical cutting. Other pipes, in the surrounding area of Victor, may be favourable targets as well but pending De Beers review they are not a primary subject for this research.

A total of five target zones have been identified to be of interest for extraction when the ultimate pit is reached. By not allowing any large scale pit modifications or pushbacks, three targets remain of interest for vertical cutting, the other two ('VSW untested' and 'VSW OPK4') require modifications to the pit slope:

- Victor North West (VNW): the northern trouser leg of the Victor Main pipe
- Victor Main Deep (VMD): the southern trouser leg of the Victor Main pipe
- Victor Southwest (VSW): the OPK 1 and DOPK facies remaining at Victor South

Table 7.1 summarizes the area of the pit bottom above each target, the volume of the kimberlite in the target and the depth of the bottom of the target in respect to pit bottom at that target. The recoverable reserves from these targets will be subject to change based on the scenario evaluation. Because the targets are located directly below the bottom of the final pit (cut 2) no overburden is required to be removed in order to access the kimberlites.

Table 7.1: Surface area, volume of identified targets and depth

Target zone	Pit bottom area (m^2)	Volume (m^3)	Max vertical distance (m)
Victor Main – northern TL	1,824	77,265	75
Victor Main – southern TL	2,290	247,793	150
Victor Southwest	4,106	292,708	105

Diamond grade estimation is not included in the scope of this research. Delineation holes and large diameter drillholes (LDD) that intersect the targets have given an initial diamond grade estimation. To what extent the grades reported by the delineation holes are representative for the entire target will not be assessed for this study. Table 7.2 summarizes the amount of holes and their reported grades for each target. The grades reported in Table 7.2 will be used in Chapter 9 in order to estimate an order of magnitude cash flow.

Table 7.2: Number of delineation holes and LDD per target, and reported grades

Target zone	Number of delineation holes	Number of LDD	Reported grade (cpht)
Victor Main - northern TL	2-3	1	~20-40
Victor Main – southern TL	1	0	~50
Victor Southwest	3	1-2	~20

7.3 Scenario evaluation

The extraction scenarios introduced in Paragraph 5.4 are evaluated in order to maximize the extraction rates while maintaining safe working conditions. Initially, the surface set-up will be used to estimate an initial extraction rate for each scenario. By fitting the layout of the cutter, including the undercarriage, on the available area at pit bottom an initial estimate for the extraction rate can be made on which blocks of the resource model can be extracted. Secondly, each scenario is evaluated regarding its ability to withstand rock failure. The stability assessment may potentially lead to adjusting the extraction rate in order to maintain safe working conditions.

Barge mounted cutting after (partial) pit flooding disrupts in-pit processes by that much that is it is not considered desirable for the Victor mine at this stage. This scenario, in addition, requires different evaluation methods due to the irrelevance of spatial limitations at pit bottom. The extraction rate of the pit flooding scenario (as described in Paragraph 5.4.4) achieves a theoretical extraction rate of 100%. However, at this moment in the development of the technology, the cutting depth from a barge is limited by the water table: the cutting depth is reduced by the water depth from cutter to target. For the scenario evaluation for the Victor mine only the checkerboard, long trenching and backfilling scenario will be considered.

7.3.1 Surface set-up

For the estimation of the extraction rates based on surface set-up the areas of the pit bottom that lie above each target zone are used. By fitting the cutter to the pit bottom area it can be determined which blocks can theoretically be mined by the cutter. Figure 7.2 displays the area on the pit bottoms of cut 2 in gold. The evaluation principle is quite simple and predominantly determined by the size of the base machine. The evaluation criteria are such that the base machine with the cutter must always be able to exit the pit and cannot cross a trench unless it is backfilled, and the machine cannot drive into the pit slopes which are simplified by a rigid vertical wall.

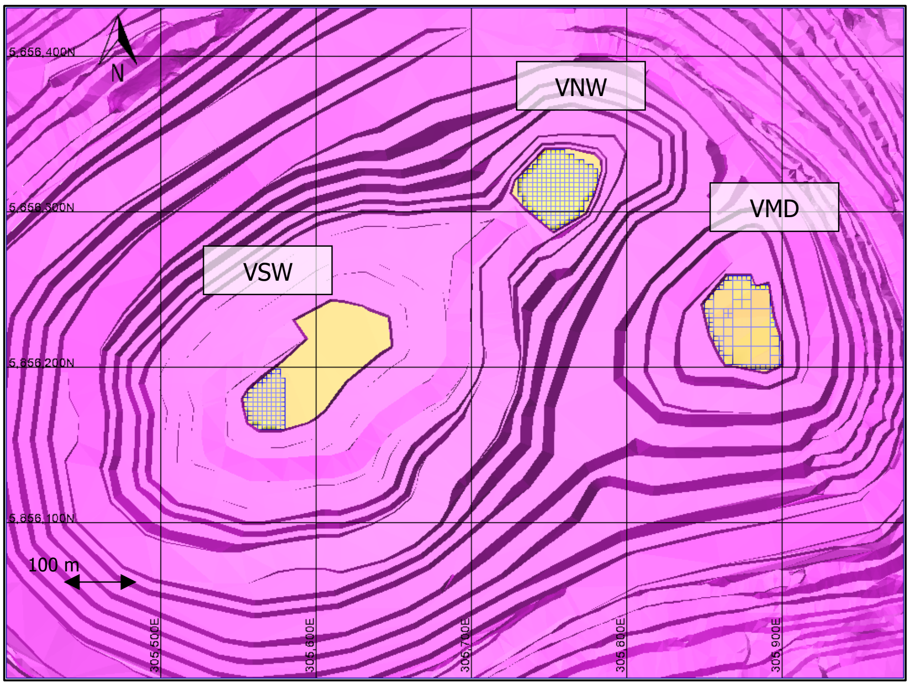


Figure 7.2: Location of the three target zones within cut 2, displaying the surface of the targets with rectangles with dimensions 1.6 x 1.5.

The process will be demonstrated for the southern trouser leg of Victor Main, VMD. The following assumptions have been made:

- The cutter including undercarriage is represented by a rectangle with size 6.5 x 16 m. The cutter itself has the size of one primary cut and is located in the middle of one of the short sides of the rectangle;
- A primary cut has the following dimensions: 3.2 x 1.5 m;
- A secondary cut has dimensions: 1.6 m x 1.5 m;
- Cutting starts at the furthest trench, away from the access ramp, from this first trench the cutter retreats towards the ramp;
- Trenches are cut with their longest edge perpendicular to the axis of the access ramp, this ensures maximal recovery; and
- Spatial requirements of the slurry separation units, slurry tanks or any other auxiliary equipment have been taken into account.

The results of the evaluation of the three extraction scenarios, checkerboard, long trenching and backfilling, are displayed in Figure 7.3. Primary cuts are displayed in orange, secondary cuts in green, backfill in grey and the rectangle that represents the area of the cutter, including its undercarriage is displayed in yellow in the bottom left corner. The fourth scenario, flooding the pit, is not displayed as it is not considered for the mine life extension of Victor. In addition, potential surface limitations at pit bottom would not have been relevant when the cutter is mounted on a barge.

By fitting the cutter including undercarriage inside the pit bottom area it can be seen that the cutting system does not fit in all locations due to its dimensions. Especially towards the sides of the pit bottom a considerable amount of kimberlite is left behind. Choosing the direction of the trenches perpendicular to the retreat direction, the amount of recovered material is maximized. In Figure 7.3 the direction of retreat is equal to the axis of the ramp.

Figure 7.3a, b and c display the first three extraction scenarios, checkerboard, long trenching and backfilling respectively. Especially in (a) and (b), it is assumed that the cutter is unable to return to the starting trench, the trench on the opposite side of the access point, as the trenches remain open. In (c) the backfilling scenario as described in Paragraph 5.4.3 is displayed, extracting the crown pillars in between the previously cut and backfilled trenches. However, in contrast with the other scenarios, in Figure 7.3d it is assumed that the cutter returns a third time across the already backfilled trenches and no longer follows the initially set out trenching or retreat direction. The blue colour represents all the additional material that is extracted, with only 3 blocks of non-extractable material remaining in the corners. Figure 7.3d represents the potential of the backfilling scenario achieving a near 100% extraction rate.

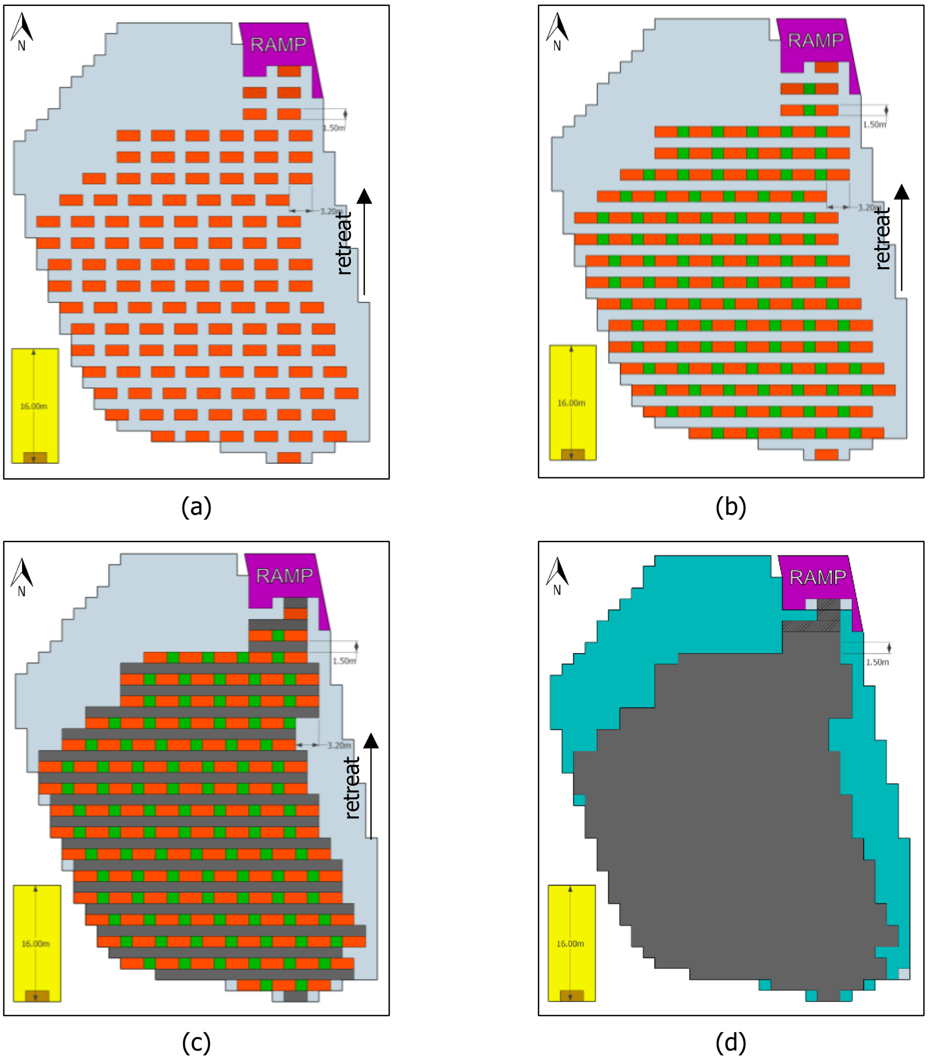


Figure 7.3: Schematic representation of extraction scenarios based on surface set up at VMD target. (a) Checkerboard layout, (b) long trenches, (c) phase 2 trench extraction after backfilling of phase 1, (d) remaining kimberlite to be extracted after backfilling of phase 1 and phase 2 trenches

Based on the surface area required by the cutter, the extraction rate is evaluated as the area extracted versus the area left behind when cutting has stopped. Table 7.3 summarizes the initially estimated extraction rates for the three scenarios and each target. On average, the extraction rates are 28% for the checkerboard scenario, 40% for long trenching and 99% with backfill. The average extraction rates are weighted according the target volume.

Table 7.3: Summary of calculated extraction rates based on surface set-up

Extraction scenario	VNW	VMD	VSW
Checkerboard	28%	25%	30%
Long trenching	40%	35%	46%
Backfill	99%	99%	98%

7.3.2 Trench Stability

The stability of the kimberlite rock mass and the trench plays a crucial role in the determination what scenarios are considered feasible and which ones are not, both economically or safety wise.

Checkerboard trench extraction

Based on Bauer expertise, it is overall accepted that the checkerboard scenario is stable and that collapse of the trench walls is unlikely to occur during the extraction phase. However, with an extraction rate of about 30%, this scenario has a lower extraction rate than the other scenarios.

Improving the extraction rate for the base case checkerboard scenario is beneficial in order to increase the output of the material of value. There are two methods of doing so while maintaining a checkerboard pattern:

1. Increasing the size of a trench by connecting two primary trenches with one secondary trench.
2. Decreasing the crown pillar width, |x|.

Both methods influence the stability of the trench walls and may lead to failure. However, due to the limited extent of the trench expansion or pillar size decrease the probability of for instance toppling and buckling failure as well as planar failures is considered low. Forces around the trench can dissipate around the trench as the pillar remains connected to the rest of the rock mass.

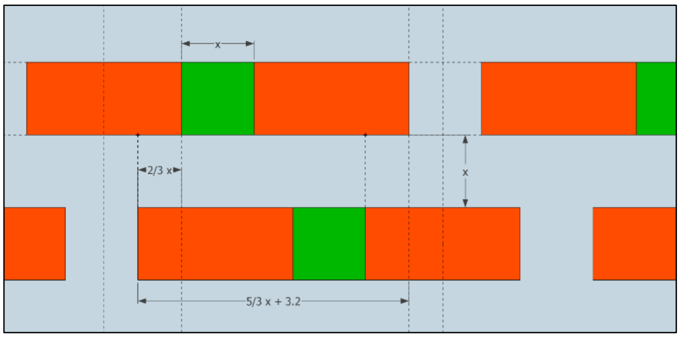


Figure 7.4: Applying secondary trench on two adjacent primary trenches

Figure 7.4 depicts the proposed connection of different primary cuts with a secondary trench. Secondary trenches in green are of half the size of a primary trench. Sequencing of the primary and secondary cuts is done according the standard cutting sequence (Section 5.2). Connecting two primary trenches increases the extraction rate with 5%, yet it is expected that instability of the trench walls will remain limited. The connection of the pillars in all horizontal directions should prevent failure from occurring along wedges or by a toppling pillar and allows stresses to dissipate around the trench.

Long trenching

If the retreat of the cutter is well planned according to the surface layout of the pit-bottom, long trenching can achieve an extraction rate slightly short of 50% of the resources, depending on the target zone. This extraction rate assumes that the pillar width, |x|, is equal to the trench width. Based on the determined extraction rates (Table 7.3), long trenches would appear to be favourable over the base case checkerboard method.

Due to its symmetry, the long trenching extraction scenario will provide the basis for a two-dimensional stability assessment. Broadly speaking, the assessment can be considered to be consistent of two parts: pillar stability (small scale) and pit stability (macro scale). Pillar stability assesses potential rock mechanical failure mechanisms in the pillar wall, whereas for the latter, the scale was enlarged significantly resulting in a much more complex interaction between the rock masses and regional stresses.

Pillar stability: rock mechanical slope failures

Stability of the pillar and trench wall is crucial to ensure the continuous output of material and prevent damaging the cutter equipment. The pillar provides a guidance for the cutter downward. In order to make sure that the cutter continues on a vertical path, the resistance imposed by the walls of the trench should be equal on both sides. If not equal, the cutter would divert towards the open trench, following the path of least resistance (Figure 7.5).

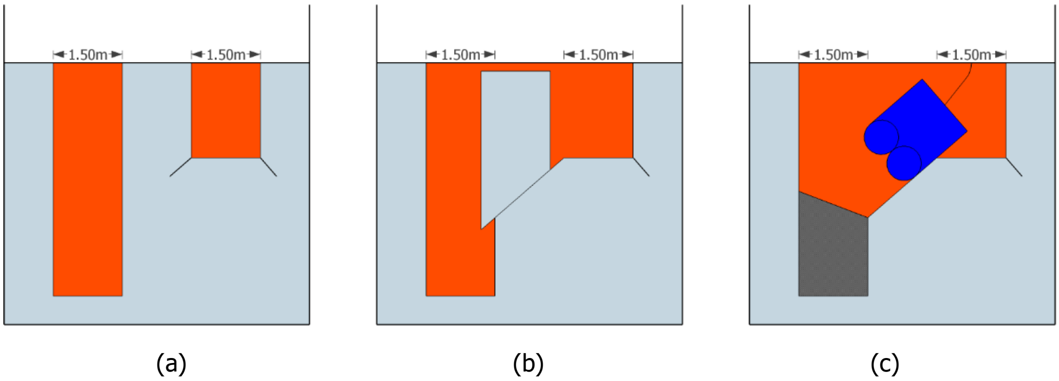


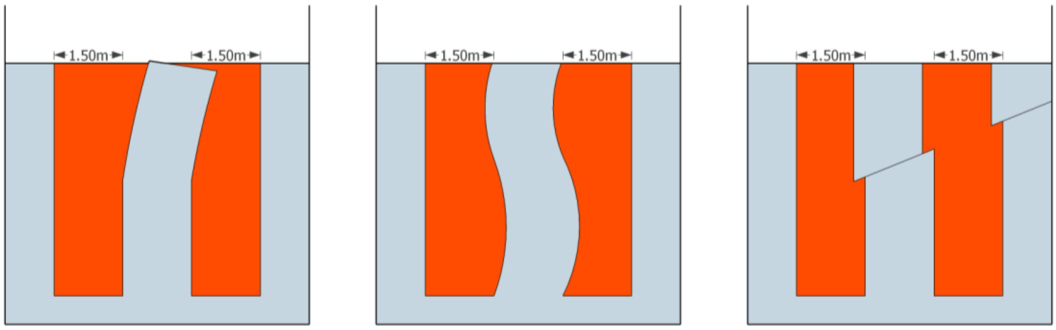
Figure 7.5: Inducing controlled failure of pillar into non-active trench. (a) Original situation: completed trench on the left, cutting occurs in shallower trench on the right, (b) potential planar failure of the pillar in between trenches, (c) cutter in right trench diverting into left trench due to absence of equal confinement

Inducing failure, even when controlled, has a high risk of unintended collapse potentially damaging the cutter or leading to a diverted cutting path. The possibilities of inducing failure of the pillar will therefore not be considered in the rest of this preliminary study. Scenarios to mitigate potential damages and decrease the likelihood of uncontrolled failure can further be investigated to increase the recovery of vertical cutting.

Three types of rock mechanical slope failures are expected to occur on the pillar or trench wall. Failure due to excessive loading at the base of the pillar is unlikely to occur. The load imposed by the overlying rock (average rock unit weight of 25.10 kN/m^3) in the pillar is approximately 3.75 MPa at maximum pillar height of 150 m, and therefore does not nearly exceed the uniaxial compressive strength of the kimberlite.

The three slope failures that may occur are the following:

1. Toppling (Figure 7.6a)
2. Buckling (Figure 7.6b)
3. Planar/wedge failure (Figure 7.6c)



(a) (b) (c)

Figure 7.6: Potential pillar failure mechanisms, (a) toppling failure, (b) buckling failure and (c) planar or wedge failure

Toppling and buckling

Toppling occurs when columns of rock, formed by steeply dipping discontinuities (in the case of trench cutting represented by open cuts), rotate against the slope about an essentially fixed point at or near the base of the slope [54]. If the dimensions of the column are such that the centre of gravity acts outside the base of the column, than there is potential for the slab to topple [55]:

$$\frac{y}{\Delta x} > \cot\alpha \rightarrow toppling\ occurs, \tag{7.1}$$

where y is represents the column height, x the width of the column and α the dip angle of the base of the slab. The relation is solely based on dimensions of the slab and does not take any rock properties into account. As such, assuming perfectly vertical columns consistent of isotropic and homogenous material that remain in between two cuts, there is no reason to assume spontaneous toppling will occur. All forces within the column, are applied in one dimension, keeping the column in a stable upright condition, as it has been already demonstrated that a compressive failure at the base of the column will not occur.

Buckling occurs when a narrow but tall slab starts behaving elastically. Buckling is therefore highly related to toppling, considering that the likelihood of the centre of gravity acting outside the base of the column is increased when buckling occurs. Due to the slenderness of the pillar between two trenches, the possibility of a buckling failure is a stability risk.

Buckling has been studied extensively for various materials including metal and concrete. Self-buckling, where a column starts buckling under its own weight, requires the load that is applied through the axis of the column, to be continuous throughout that column. The Euler's buckling theorem must therefore be adapted to [56]:

$$l_{cr0} = \sqrt[3]{\frac{9B^2}{4}\frac{EI}{\rho g A}}, \tag{7.2}$$

where l_{cr0} is the critical buckling height, or the height of the column at which it will start to buckle, B the first zero of the Bessel function of the first kind of order -1/3, E the modulus of elasticity, I the second moment of inertia of the column cross section A, g the acceleration due to gravity and ρ the density of the rock.

The results of the critical buckling relation are displayed in Figure 7.7. The critical buckling height (CBH), marked by the blue line, depicts the maximum height the pillar can have before buckling occurs. Therefore, the area above the line represents the stable area in regards to buckling failure. Rock properties used to calculate the critical buckling height have been determined based on the data of the geotechnical assessments performed by SRK [24] and Golder [25]. The average measured Young's modulus and average density of the 29 kimberlite samples collected during these studies is used in Equation 7.2.

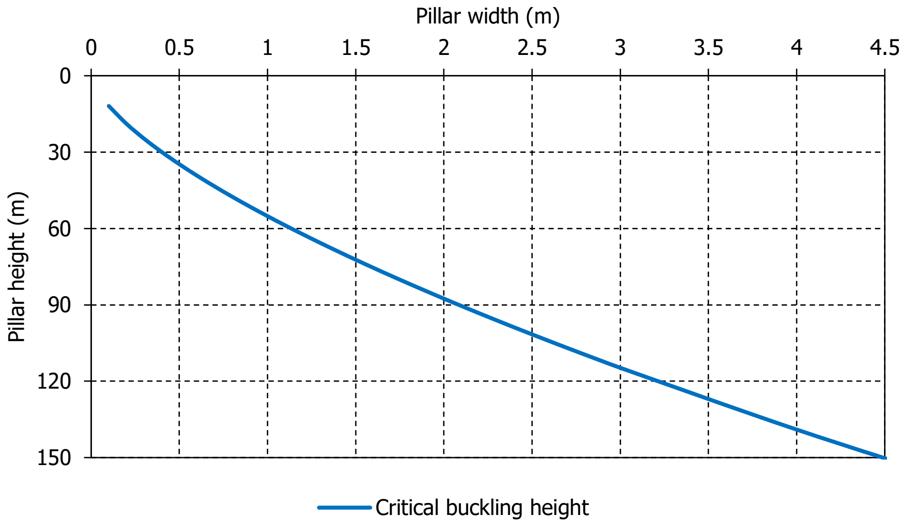


Figure 7.7: relation between pillar width and critical buckling height of the pillar based on self-buckling theorem

Figure 7.7 shows that a buckling failure is likely to occur in the pillar when the pillar widths are minimized and cutting depth is maximized. It is suggested that with a pillar width of 1.5 m the trench depth should remain limited to 72 m in the long trenching scenario. When applying the maximal cutting depth of 150 m, a pillar of about 4.5 m should remain in order to prevent the pillar from buckling under its own weight. The relation between pillar widths and heights has been calculated for an idealized and extreme case. No internal rock variations or failure planes have therefore been accounted for in the relation in Figure 7.7. Accommodating for internal variations may be done by applying a factor of safety to the pillar design, further decreasing the trench depths.

In this pillar failure assessment the water inside the trenches has not been accounted for. Although it is generally assumed that the presence of a fluid inside a trench provides stability to the trench walls, the fluid should not stabilize in the case of a toppling or buckling failure. Because the force exerted by the fluid is equal on both sides of the pillar the net effective force on the pillar is zero.

It can be concluded from the calculations presented above that the stability of the pillar is heavily influence by the geometry of the pillar. When attempting to maximize the extraction rate, the pillar size is consequently minimized. The likelihood of pillar failure when trenches are cut to maximal depth allowed by the cutting equipment is high. Based on the likelihood, it is suggested that the crown pillar width is not reduced below its initially assumed value of 1.5 m. In order to improve stability the pillar width should be increased. When assessing the recoveries of the extraction scenarios, the influence of increasing the pillar width to 2 and 3 m will be addressed. Similarly, reducing the trench depth may provide improved stability conditions in the trench walls. When applying the long trenching scenario, a combination of reducing the trench depth from its maximum value and increasing the pillar width, should decrease the likelihood of failure.

Planar and wedge failure

Planar and wedge failures occur along zones of weakness inside the rock. The failure mechanism is therefore, dependent on the likelihood of the pillar crossing such zones of weakness, e.g. fractures, joints or weak anomalies. In order to assess the likelihood of crossing planes of weakness, the rock quality can be assessed using core log data. As presented in Paragraph 6.5.1 the kimberlite rock quality (RQD) is generally assessed to be good to excellent.

No core data is available from inside the targets and a precise indication of the likelihood of crossing zones of weakness cannot be made. However, assuming the rock strength and rock quality behave fairly uniform throughout the deposit, or increase with depth, planar and wedge failures are unlikely to be a stand-alone failure mechanism. The combination of different failure mechanisms, including a planar failure may, however, be of greater risk.

Pit stability: finite element modelling

In order to simulate the influence of the trenches on the open pit and pit slopes, finite element modelling is used. Finite element analysis allows simulation of the macro scale stability, including the influence of regional stresses and stresses exerted by the pit walls, as well as the opportunity to assess how the pit stability is affected by trench cutting.

Figure 7.8 displays the modelling steps that were applied in order to create a model that reflects the true situation. Validation of this model was done by simulating the current pit design from cut 2. The obtained factor of safety (FoS) was compared with the FoS determined during the pit stability assessments in previous studies [57]. Considering the geotechnical conditions in the Victor mine are currently stable, it was determined that the FoS determined during the pit stability analyses provides the benchmark for the FE models used in this study. Validation of the macro scale stability simulation is further discussed in the Chapter 11, Discussion.

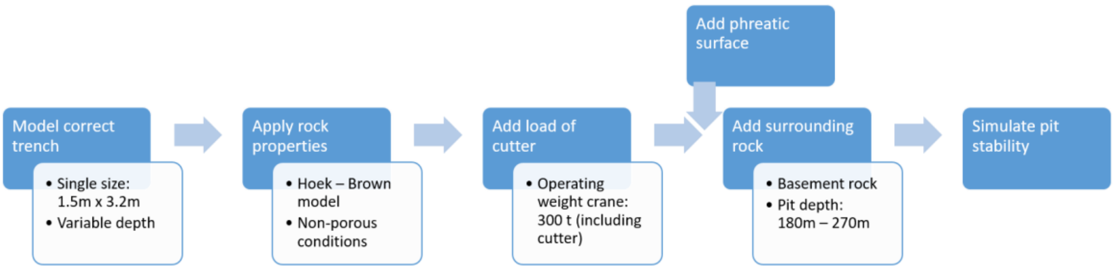


Figure 7.8: Flowchart finite element modelling steps

Finite element analysis gives an optimistic representation of the stresses or displacements within the geological or geotechnical zone. The calculations assume a homogenous material in the model. This means that variation of the compressive strength or elasticity of the rock is not accounted for. Because such variations are very likely to be in place, the validity of the simulation in regards to the real-life situation needs to be questioned. The objective of applying FE modelling software is to highlight potential risks of rock instability. If instability risks are determined based on the simulations with homogenous material, they require attention in order to maintain safe working conditions in real-life.

The finite element analysis performed in the software RS^2 is not suitable to study small scale failure mechanisms like buckling and toppling. Within the pillar, RS^2 will assume two 1D forces that counteract, one represents the weight of the overlying column, the other equal but opposite at the base of the column. No deviational stresses on the rock column (external or internal) can be taken into account by the program, unless specified by the user.

By applying factors of safety or assigning a probability of failure, a relative comparison of the different simulations can be made. In general, the factor of safety or, strength reduction factor (SRF), equals 1 for models that are in equilibrium. For the Victor pit design, a minimum SRF of 1.3 is considered acceptable [58]. If conditions in the slope or model are instable, the SRF takes a value between 0 and 1, where less stable conditions receive a lower SRF. In essence, the SRF represent the factor by which the rock parameters can be divided for the finite element model to be in equilibrium. For instance, when applied to the intact rock strength:

$$UCS_{eq} = UCS_1 \times SRF^{-1}, \quad (7.3)$$

where UCS_{eq} represents the intact rock strength at equilibrium, UCS_1 the intact rock strength as supplied to the model, and SRF the strength reduction factor. An SRF higher than 1 indicates by how much the rock parameters can be reduced until equilibrium. For an SRF between 0 and 1 the rock parameters must be increased in order to reach equilibrium.

The influence of trench cutting on the macro scale stability is investigated along a total of 4 representative cross-sections through the target zones (Figure 7.9). The sections, except section 1-1', are chosen such that they are parallel to the retreat direction at the relevant target, as shown in Figure 7.3. Section 1-1' cross-cuts the pit from NW to SE and is used as a basis for the macro scale stability assessment. The simulation results of the other sections are summarized in Appendix C.

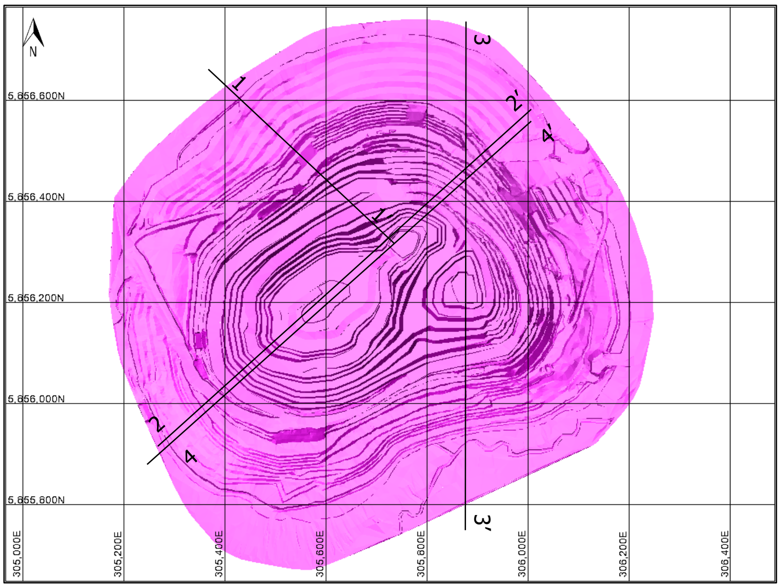


Figure 7.9: Plan view of cut 2 including cross section lines

Using the pit design of cut 2 and the country rock lithologies the model in Figure 7.10 was created for the Victor North West pipe (cross section 1-1'). The trench is indicated with the vertical line at the pit bottom. The depth of the trench covers the full depth of the diamondiferous target kimberlite as discussed in Paragraph Table 7.1. The pit walls, slope angles and water table (indicated by the blue line) roughly conform the pit design parameters of cut 2, some deviations may be present but not significant enough to influence the stability analysis comprehensively. Bench design parameters and geological sequence have been exported from the pit design and geological models of the Victor mine.

The macro scale stability is assessed by evaluating the rock mass strength properties. The rock mass properties have been estimated during geotechnical drilling [58] and adjusted according the results from the intact rock strength assessment in Section 6.5. The generalised Hoek-Brown failure criterion was used in the finite element analysis. The Hoek-Brown criterion is estimated based on four material constants, the UCS, material constant m_i, the geological strength index (GSI), and a disturbance factor. The UCS has been assumed 39 MPa as determines previously, the other material parameters are estimated based on the annually updated stability analyses [57, 58]. The material values are provided in Appendix B. To minimize influence of model boundaries, the full model span is greater than displayed in Figure 7.10. The X and Y spans of all sections are included in Appendix B.

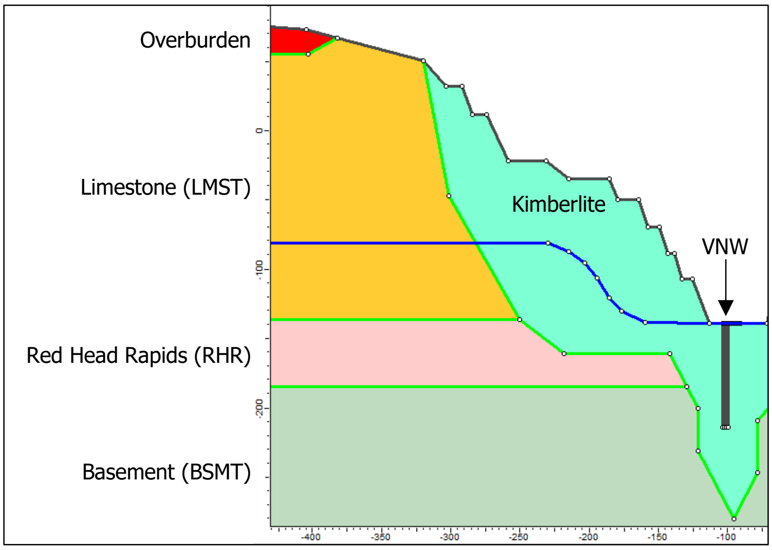
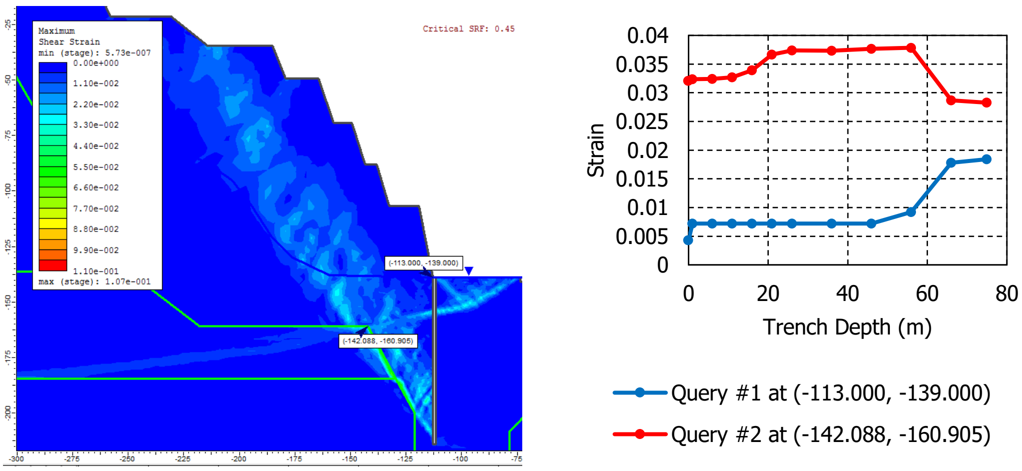


Figure 7.10: Constrained cross section 1-1'. Stratigraphic sequence and pit design overview for critical slope. Phreatic surface indicated by the blue line. Regional stresses simulated according Arjang [59]. Material properties, regional stresses and full model dimensions are included in Appendix B.

In order to maximize the effect of the trench on the slope stability the simulation is performed with the trench at the toe of the pit slope. As can be seen in Figure 7.11, by undercutting the designed slope with a trench, instability may occur in the pit slopes. Without stability measures and with a trench directly at the toe of the slope, the likelihood of failure of the benches above is high.



(a) (b)

Figure 7.11: Maximum plastic strain in pit wall without stability measures in the trench, section 1-1'. (a) Constrained section view of critical pit slope including excavated trench to 75m, (b) change in strain with increasing trench depth for query #1 and #2.

Figure 7.11a displays the pit slope with a trench excavated until 75 m, or until maximum determined depth in the VNW target. Possible failure planes are indicated by the variations in maximum plastic shear strain. Figure 7.11b plots the maximum plastic shear strain versus the trench depth. In the created finite element model, trench cutting was modelled using a total of 12 stages. The first stage represents the undisturbed pit, without a trench. Secondly, the superficial cut is included. The following stages increase the trench depth incrementally with 5 m, 10 m and 15 m towards the later stages. Inside the trench, a distributed load, or ponded water load, is created that represents the water inside the trench. The ponded water load induces stress on the trench walls, providing some additional stability. Query point #1 was chosen in the location where the greatest total displacement takes place according to the finite element model, at the toe of the original undisturbed pit. Query point #2 is located on the edge between kimberlite and the RHR rock formation. As can be seen in Figure 7.11a, the maximum shear strain concentrates along the boundary between the kimberlite and the RHR formation. Query point #2, on the upper boundary of the RHR formation, experiences influence of the trench cutting already with a shallow trench. Figure 7.11b suggests slope failure occurs when the trench depth is approximately 60 m. The SRF for the situation shown in Figure 7.11 is 0.45.

Three methods of improving the overall pit stability have been considered:

1. Moving away from the toe of the slope (SRF = 0.62)
2. Apply a liner or concrete structure to the first meter of the trench (SRF = 0.43)
3. Use a higher density fluid in the trench during cutting (SRF = 1.39)

Installation of a liner has no influence on the slope stability and is not further considered. Moving away from the toe of the slope has some positive effect but decreases the extraction rate by so much it is not considered an attractive stability measure at this time. The heavy density fluid does effect the stability significantly, however, costs and environmental impact are not considered at this time. Simulation results are provided in Appendix C.

Vertical cutting with backfill

The application of backfill into the non-active trenches should ensure appropriate stability if the backfill properties are well chosen. Minimum requirements of the backfill are that it has a density equal to the density of the kimberlite and that it is cemented in order to provide a stable working platform.

In addition, to ensure stable rock conditions, cutting long-trenches must be limited. By cutting multiple shorter trenches (consisting of primary cuts and secondary cuts) stresses imposed by the surrounding rock mass dissipate around the trench while it is being cut and backfilled. The backfill scenario is therefore highly dependent on an appropriate work schedule (Section 7.4).

7.3.3 Extraction rates

A total of three different extraction scenarios have been designed and evaluated in order to maximize kimberlite recovery at Victor, while maintaining stable rock conditions. It has been chosen to evaluate the scenarios based on the stability, both on small scale and macro scale, and available surface area. Based on the stability assessment, extraction scenarios can be adjusted in order to decrease the risk of failure.

Small scale stability, the stability of the pillars in between the cuts, assessed the likelihood of failure occurring inside the pillar or trench wall based on three failure mechanisms. Buckling proved to be the most influencing, limiting the trench depth to 72 m when maintaining a pillar width of 1.5 m. Failure mechanisms are not likely to occur by themselves and a combination of toppling, buckling and planar failures would be more probable for the long trenching scenario. When applying the checkerboard scenario or backfill, the risk of failure is expected to be lower due to interconnection of the remaining rock mass and the stabilizing backfill. Macro scale stability assessment, or pit stability, indicates that undercutting the designed slope may cause movement of the rock inside the slope. Application of a heavy density fluid has the largest impact on improving the stability of the slope. However, increasing the density of the fluid inside the trench with for instance bentonite is expected to have a considerable impact on project planning and project financials.

Considering both the limitations and benefits of finite element analyses, it must be concluded that the geotechnical stability associated with the vertical cutting of long, deep trenches at pit bottom is associated with considerable risk. The creation of long trenches should carefully be studied, if this scenario is decided upon as preferential.

Based on all scenarios, extraction rates were calculated, including the variations on each scenario as determined during the previous paragraphs (Table 7.4). The checkerboard scenario is expanded by the option of trench extension with one secondary cut, increasing the base case extraction rate with 5 to 7%. Secondly, the long trenching method, which was used in the stability assessment, has been expanded with three options for improving stability: maximum depth reduction, pillar width increase and application of backfill. The base case long trenching scenario, which is considered unstable, assumes a crown pillar width of 1.5 m and maximum applicable cutting depth. For the reduction of the maximum trench depth in the long trenching scenario, three trench depths corresponding to the critical buckling heights were selected. According to Figure 7.7 the critical buckling height for a pillar width of 1.5 m was 72 m, 88 m for 2 m and 115 m for 3 m. In order to assess the variations in extraction rates in Table 7.4 maximum depth reduction and increase of pillar width are displayed separately: for the maximum depth reduction the base case pillar size of 1.5 m is again assumed, and for the increase in pillar width no depth restrictions are applied. The adjustment of the sizes of the trenches and pillars significantly reduces the extraction rates. For almost all cases the rates become lower than the checkerboard scenario with trench extension or even the base case checkerboard. The application of only one safety measure does not provide the appropriate stability, the reduction of the depth and the increase of the crown pillar width does. Table 7.4 displays the measures separately to address their influences on the total recovery separately. Application of backfill within the long trenching scenario extracts the kimberlite according the base extraction rate of the long trenching scenario but with stable rock conditions. Lastly, the backfilling scenario assumes near total extraction of the targets and its recovery is the same as determined by the surface evaluation (Table 7.3).

Table 7.4: Extraction rates for evaluated scenarios

Extraction scenario		VNW	VMD	VSW
Checkerboard		28%	25%	30%
Trench extension		35%	30%	37%
Long trenching		40%	35%	46%
Reduce max depth	72 m	39%	19%	38%
	88 m	40%	23%	43%
	115 m	40%	29%	46%
Increase pillar width	2 m	34%	30%	37%
	3 m	27%	23%	29%
Apply backfill		40%	35%	46%
Backfill		99%	99%	98%

Following the stability assessment the checkerboard scenario (potentially with limited extension) or backfilling of non-active trenches after cutting will provide the basis for the rest of this research. Both cases should enable the rock stresses to dissipate laterally around the cuts and provide therefore more stable conditions on both the small and macro scale as well as attain relatively high extraction rates compared to the long trenching scenario with stability adjustments. The risk of cutting long trenches is considered to be of high risk for the overall rock stability of the operations.

7.4 Task schedule

Following the designed extraction scenarios, the design of an appropriate task schedule will ensure that the estimated recoveries are achieved by the vertical cutter system. Optimizing the sequencing of the trench cutting and retreat direction may improve the extraction rates by opening up more of the pit bottom area above a target.

As mentioned during scenario evaluation based on surface area, it is considered optimal to cut trenches perpendicular to the axis of the ramp. By placing the trenches as such, the extraction rate is maximized by allowing the cutter to exceed the limits of the pit bottom at the base of the ramp. By cutting the trenches perpendicular to the axis of the ramp the direction of retreat was chosen to be toward the ramp. Especially when assuming no backfilling, no new working platform is created, rendering the area where cutting already has taken place inaccessible for the cutter. In order to prevent the cutter from being unable to exit after extraction the retreat direction should be towards the ramp.

For the base case, the checkerboard scenario, the direction of trenching as initially assumed is displayed in Figure 7.12 for the VMD target. This target can be distinguished from the other two target based on the vertical continuity of the pipe. With an available surface area on the pit bottom of 2,290 m^2, it is the slightly larger than the VNW target area. However, its total volume is about three times as large as the VNW target. The influence of the inability to reach certain areas on pit bottom it therefore also greatest for this particular target. As can been seen in Figure 7.12d considerable amount of kimberlite is left behind towards the northern and eastern sides of the pit bottom.

The sequence displayed in Figure 7.12 can be extrapolated to the other extraction scenarios. They apply in addition to the primary cuts, also to secondary cuts. The secondary cuts have a length half of a primary cut and connect two primary cuts. A secondary cut is excavated when the two adjacent primary cuts have been completed (Chapter 5).

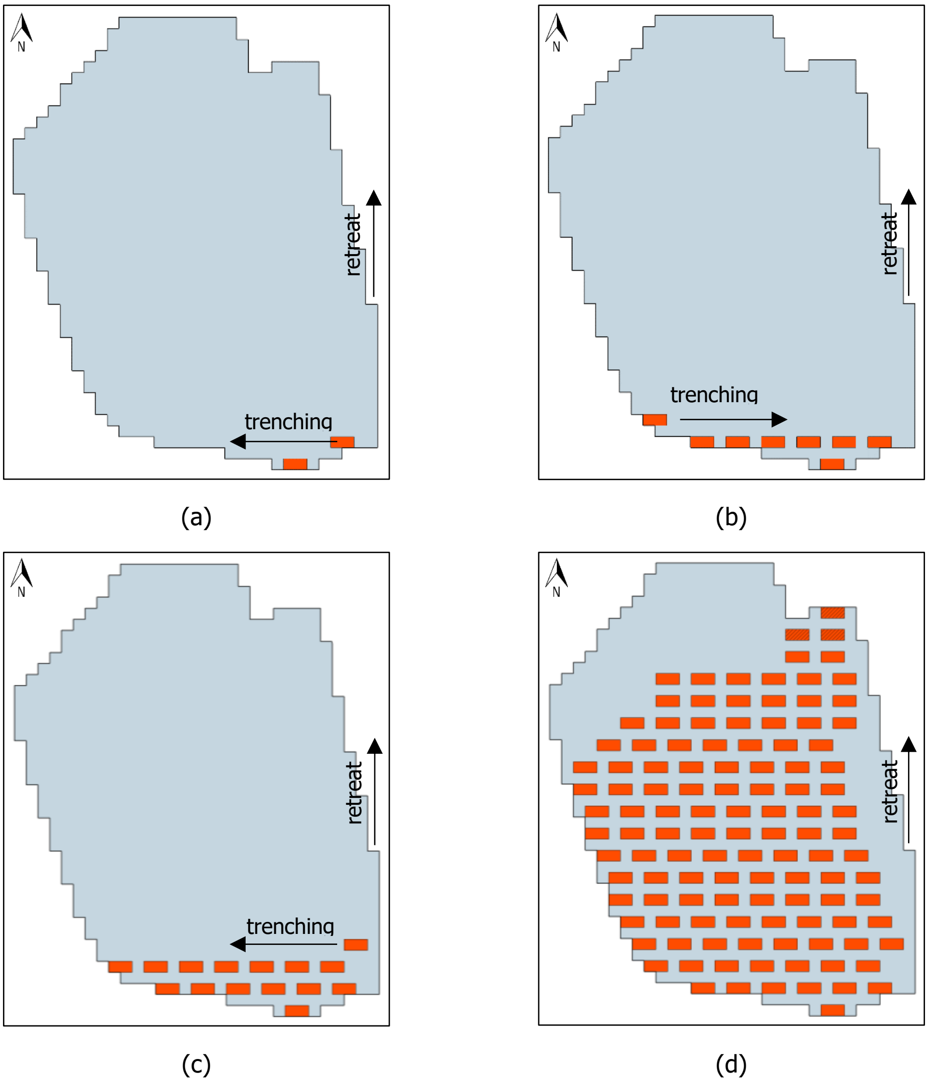


Figure 7.12: Work schedule for the checkerboard scenario at the Victor Main Deep target, indicated the trenching direction and retreat direction. Cutter is located on the rock mass north of the cut trenches. (a) First trench of the second row, (b) and (c) first trenches of the third and fourth row, (d) completed checkerboard lay out

When applying backfill into long, interconnected trenches, the backfill is placed as soon as possible after cutting, but not before cutting inside the long trench that is being backfilled is complete. Putting the backfill in place as soon as possible after cutting ensures the curing time of the backfill is optimally used without disrupting cutting processes. Backfill hardens over time, referred to as the curing time. Only when the backfill is allowed sufficient time to cure it can reach its peak strength. Including the backfilling process the following task schedule is proposed, based on Figure 7.12:

1. Start cutting of regular primary trenches according checkerboard pattern;
2. Start filling of the cut primary trenches with backfill as soon as one trench is completed;
3. Repeat steps 1 and 2 until full kimberlite body extent of target body is mined;
4. Return to pillars that have been left behind, now surrounded by backfill on two sides;

5. Extract and backfill secondary trenches;
6. Return to crown pillars that have been left behind, surrounded by backfill;
7. Repeat steps 1 through 5 for remaining pillars.

Experience from current civil engineering construction sites suggests a different schedule. The schedule, seemingly more complicated regarding sequencing, is currently applied for the construction of diaphragm walls around underground metro stations. It has the additional benefit of creating only shorter interconnected trenches, maintaining stable conditions in the pit. Figure 7.13 displays the suggested schedule according to current practice in civil engineering, applied with only single trenches.

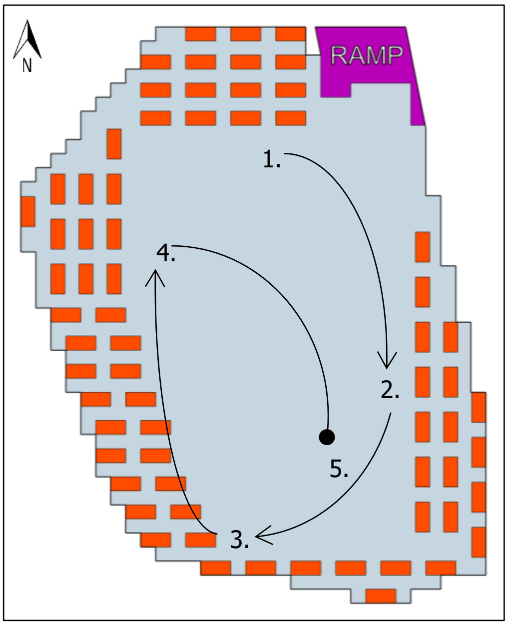


Figure 7.13: Adjusted work schedule based on current civil engineering practice for a checkerboard layout. Cutting location sequencing corresponds to the task sequence provided below.

The adjusted or optimized work schedule makes better use of the full extent of the pit bottom area. With the schedule displayed in Figure 7.12, the kimberlite at the edges of the target are left behind, reducing the extraction rate. By extracting the kimberlite along the edges first, while leaving sufficient room for the cutter to retreat according the established work schedule, the total amount of material that can be extracted is optimized. The sequencing as displayed in Figure 7.13 assumes the following task schedule, where the numbers refer to the process step in the figure:

1. Cutting of single trenches next to the access slope
2. Cutting of single trenches on the next location
3. Cutting of single trenches along the complete boundary of pit bottom, as far as the spatial requirements of the cutter installation allow.
4. Cut remaining trenches that are left behind in between 1. and 3.
5. Return to the middle of the remaining kimberlite and start cutting at the farthest location from the slope. Excavate all remaining kimberlite according checkerboard scenario.

Although the adjusted work schedule does not necessarily require backfilling in order to achieve the proposed extraction rates, the cutter system operates in locations close to the edges of the already cut trenches. Covering the open trenches and surrounding them with fencing or barricade tape in order to ensure no employees can fall into an open trench is recommended. Metal floor grates are currently used on civil engineering projects to cover trenches. Due to the proximity of the operating cutting system to the open trenches, the optimized work schedule may not be applicable for all mining situations. In cases where rock strengths are insufficient for the cutter to be operating close to the edges of non-backfilled trenches it may be suggested to apply the standard cutting sequence. In the latter case all cutting activities remain on top of the more stable rock mass.

Figure 7.14 displays the sequence for the VMD target zone for the complete extraction of the kimberlite assuming a backfilling work schedule. Figure 7.14a and b also display the final layout of the base case checkerboard and long trenching method. The kimberlite that remains after (d) can be excavated by 'cherry-picking' after all backfill is cured. Optimizing the work schedule reduces the kimberlite left behind at the edges of the target for all scenarios.

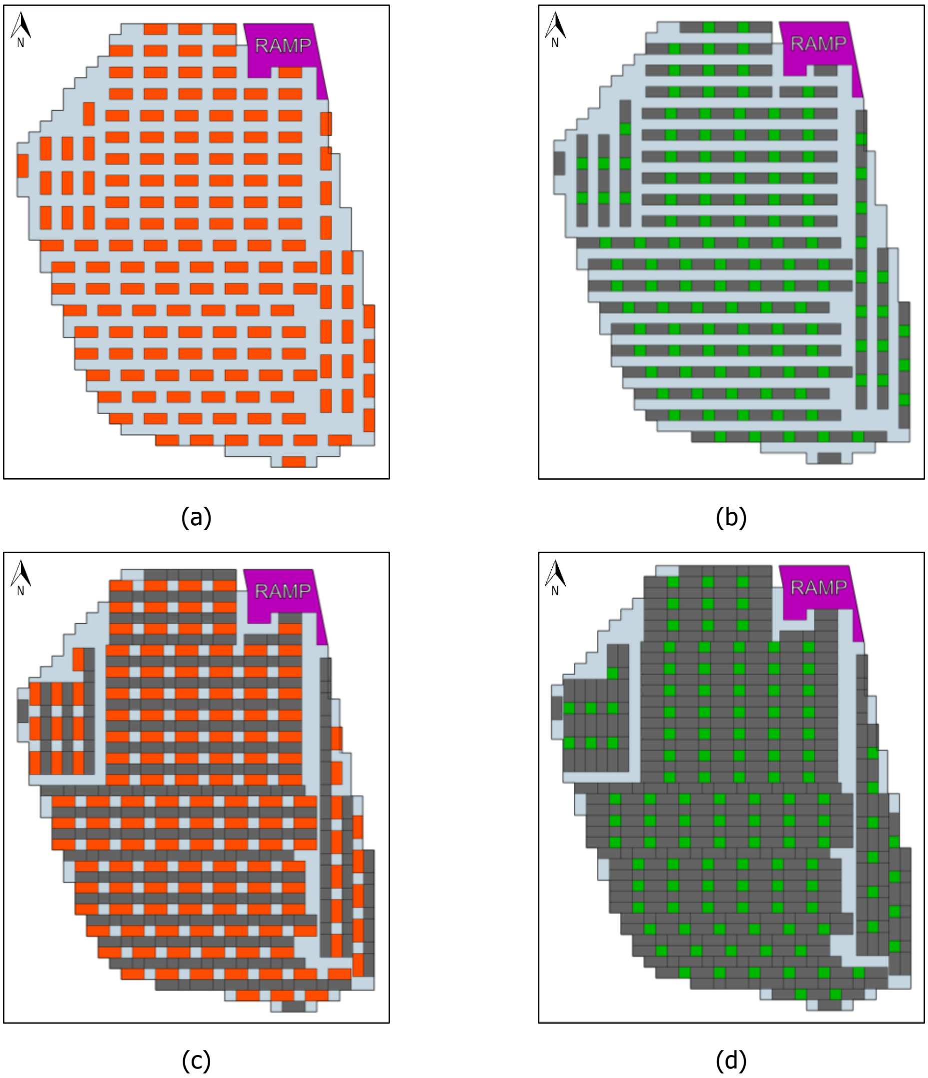


Figure 7.14: Optimized work schedule for Victor main – southern TL (VMD) target zone assuming the application of backfill after each production step. (a) Cutting of phase 1 primary trenches (checkerboard extraction), (b) cutting of secondary trenches in between backfilled primary trenches, (c) cutting of phase 2 primary trenches in between backfilled trenches of phase 1, (d) cutting of phase 2 secondary trenches

7.5 Optimized Recovery

Extraction rates as determined in Paragraph 7.3.3, were based on the initial surface area evaluation of the area needed by the vertical cutting system in the pit bottom. The extraction rates were based on a linear retreat direction towards the exit ramp at the target location, and a trenching direction perpendicular to the retreat. A significant amount of material was left behind towards the edges of the target area due to the dimensions of the cutter. Following the determination of the initial recoveries based on the process of fitting the cutter inside the pit bottom, each scenario was evaluated based on their associated limited scale and macro scale stability. Especially the long trenching scenario, in which long interconnected trenches are created, imposes significant risk to the operation. Rock failures like toppling and buckling, are likely to occur if deep trenches in between narrow crown pillars are created. In order to improve stability, measures like reducing the trench depth or increasing the pillar width may prove effective. However, increasing the stability also reduces the recovery. Macro scale stability highlighted the risk of slope deformation when creating long trenches at the toe of the pit slope.

Optimizing the work schedules according to current practice in civil engineering applications results in a decreased amount of kimberlite that is left behind due to cutter dimension limitations. On average the extraction rates of the checkerboard and long trenching scenarios can be increased with 8% for the two targets in Victor Main. The work schedule at Victor Southwest has not been further optimized. The initially determined schedule for this target already reflected a similar schedule, due to the dimensions of the target surface area and the location of the access ramp.

Especially the long trenching scenario shows an increase in extraction rates, based on the optimized work schedule. For the VMD target the extraction rate increased with 10%. Table 7.5 summarizes the adjusted extraction rates. The rates as displayed in Table 7.5 will be used for the financial evaluation in Chapter 9, Financial Evaluation. The last column of Table 7.5 contains the weighted averages of the target areas according to the total volume of the target. The VSW target with the greatest volume, influences the weighted average the most.

Table 7.5: Extraction rates according optimized work schedule

Extraction Scenario			VNW	VMD	VSW	AVERAGE
Checkerboard			34%	32%	30%	31%
	Trench extension		42%	40%	37%	38%
Long trenching			48%	45%	46%	46%
	Reduce max depth	72 m	47%	25%	38%	34%
		88 m	48%	30%	43%	38%
		115 m	48%	38%	46%	43%
	Increase pillar width	2m	39%	38%	37%	38%
		3m	30%	30%	29%	30%
	Apply backfill		48%	45%	46%	46%
Backfill			99%	99%	98%	98%

8

Time Line Design and Execution Approach

Open pit mining is scheduled to end by the end of 2018 or early 2019 in the Victor pit. It is considered favourable to be able to transgress into vertical cutter mining from that date. This chapter addresses a hypothetical time line and its constraints that may be expected during the execution of the project, assuming acceptance. The hypothetical time line indicates the expected duration of the project starting completion of this research. The remoteness and the location in the north of Ontario result in the fact that the Victor mine is dependent on the winter roads for the transportation of heavy loads. Winter roads and climate in general have significant influence on the project schedule. The preproduction schedule is the shortest possible time line assuming all activities.

8.1 Strategy

The overriding requirement of the time line is to have the heavy equipment ready for shipping across the winter roads by the time they open. The winter roads provide the only access to the Victor site for heavy loads. To ensure continuation of the extraction of the resources, the cutter system would need to arrive early 2019 at the Victor site. Cutting will subsequently start in the early spring.

A significant amount of work will be required over a short period of time to complete the feasibility assessments and engineering studies, as well as obtaining all necessary approvals, environmental or otherwise, in order to start cutting in 2019.

The primary project phases include:

- Completion of feasibility and engineering studies;
- Application for, and receipt of applicable environmental-related approvals;
- Manufacturing of parts and equipment;
- Movement of materials and equipment to the Victor site; and
- Commissioning and testing.

Cutting is anticipated to commence in 2019, following the completion of mining of the Victor open pit. A transition period is expected to be required during which time it is planned to keep the processing plant running using kimberlite stockpiled from cut 2 to allow time to initiate cutting. By applying the base case checkerboard scenario, it is anticipated that mine life of the Victor pit can be extended by an additional 28 months of production at an average production rate of 38.5 t/h.

8.2 Climate effects

As highlighted by the project strategy in Section 8.1 the climate of the James Bay Lowlands has significant effect on the overall planning of the project. Transportation of heavy loads is dependent on the connection of the mine site to all-season infrastructure south of the Lowlands.

The Victor site is connected to Attawapiskat through the James Bay winter road that also connects Attawapiskat FN to the other three First Nation communities along the James Bay coast line (Figure 6.1). Fort Albany, Kashechewan, Attawapiskat and the Victor Diamond Mine are supplied of fuel, building materials, food and other goods by winter road via Moosonee where the Ontario Northland railway ends. [42]

The James Bay winter road corridor jurisdiction extends as far south as Smokey Falls, Ontario, including part of the railway line as well. With a total length of 498 km it is the second longest winter road corridor in Ontario. The winter road itself, with a length of 312 km, is operated and managed by First Logistics, a division of Kimesskanemenow LP, a limited partnership between the four First Nations of Attawapiskat, Fort Albany, Moose Cree and Kashechewan. Slightly over 8,000 people are served by the James Bay winter road. [60, 61]

The operating statistics of the James Bay winter road are summarized in Table 8.1 for the 2016-2017 season. Opening and closing dates are yearly in a similar order of time. Transportation of heavy loads to the mining site can only occur during a two-month time frame. During the operating season of the James Bay winter road approximately 500 truck transports reach the mine site [53].

Table 8.1: Operating statistics James Bay corridor for 2016-2017 winter road season [59]

	James Bay Corridor (2016-2017)
Construction start date	December 29
Road open to light traffic	January 31
Date of official closure	March 26
Operating season (days)	62

Besides the effects of the climate on the transportation of the equipment, climate may affect the general operation of the cutter once arrived on site. Currently, no winterization process is applied to mining equipment operating at Victor. When equipment arrives, it is assembled and put into operation as quick as proper assembly and commissioning allows [Pers. Comm., Rausch, 2017].

Modifications to the mining trucks and excavators are required to ensure working conditions are adequate. Insulated operator cabins, engine block heaters and tank heaters are added to help both the employees and machinery deal with the cold climate. Elevated energy costs are a result of the cold mitigation.

8.3 Key dates

The following hypothetical dates are considered critical to the successful execution of the works assuming the project is accepted:

- March 2018 - Place order for cutter
- December 2018 - Finish manufacturing of cutting equipment
- February 2019 - Arrival of cutting equipment in Timmins, Canada
- March 2019 - Arrival of cutting equipment at Victor mine

8.4 Project schedule

The project schedule is displayed in Figure 8.1. The schedule, as pointed out in the strategy (Section 8.1) is determined by the opening of the winter road. Time frames of activities both before and after the winter road opening are based on their individual lead times and the month the road opens.

The displayed production schedule is based upon the decision to mine the highest grade kimberlite ores first and the lowest grades last. Chapter 9 will discuss the grades associated to each target in more detail. Determining which target will be mined first has a significant influence on the project value and should not be chosen at random. De Beers have indicated that the VSW would be available first when conventional mining ends in cut 2, however, due to the difference in NPV of the project, the schedule in Figure 8.1 will be applied.

Demobilization after production has not been displayed in Figure 8.1. Removal of the cutting set-up from the Victor site should be negotiated with all involved parties and is not considered at this moment for the design of a feasible mine plan using vertical cutting. However, the demobilization of the equipment may consist of considerable cost at the end of the project. If chosen to utilize the full production time of 28 months cutting would end in summer. The cutting equipment when decided to be removed from site, can only be transported back over the winter road in winter. Additional time and costs due to on-site storage and demobilization would be added to the end of the project. A digital file of the project time line is included with this thesis in Appendix D.

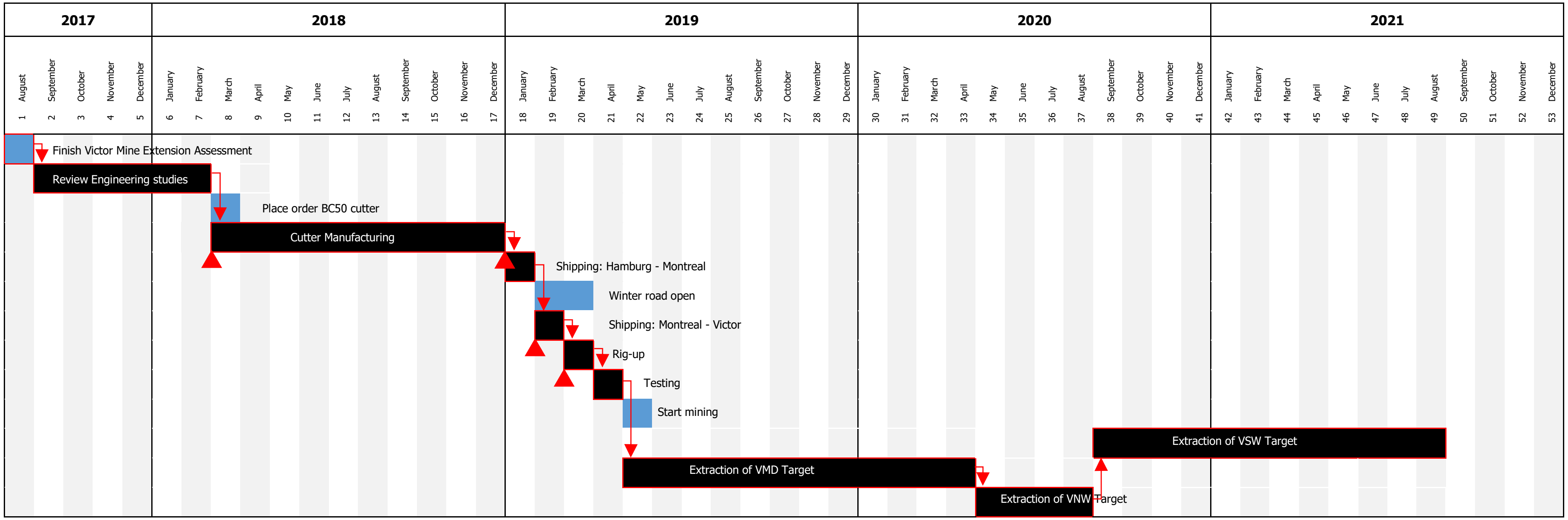


Figure 8.1: Total project schedule for the application of vertical cutter mining at the Victor mine. Critical path and key dates (Section 8.3) marked in red

9 Financial Evaluation

In order to develop vertical cutting as a suitable extraction method of the kimberlite at Victor a number of extraction scenarios have been designed as well as an initial project schedule. The motives of developing alternative mining methods are to exploit all residual kimberlite resources below the ultimate pit. Continuation of operations with conventional methods can at this moment not be justified due to impact on the surrounding environments and due to increasing stripping ratios. Yet, the estimated costs and effort associated with closure and reopening of the mine surmount the estimated value. Consequently, the aim of the alternative mining method should be to be able to justify the continuation of extraction.

The following chapter addresses the costs that are estimated for the extraction of kimberlite at the Victor pipe. Cost estimations have been provided by De Beers Canada and Bauer. The chapter will address the description of the major cost factors, and the evaluation of the project based on different time lines. Estimating expected income and profit of the project is not within the scope of this study. However, in order to deal with the evaluation of extraction scenarios within time, a grade and value per carat are assumed for scenario comparison.

9.1 Project costs

A comprehensive list with cost factors has been compiled with information provided by De Beers and Bauer. Cost items can be broadly divided into six categories:

1. Equipment costs including spare parts and secondary items
2. Production costs
3. Transportation costs
4. Personnel costs including training, camp costs, salaries etc.
5. Additional costs including insurance, contingency, taxes, etc.

Within the categories cost items are further separated in variable costs, costs dependent on time, tonnes or people, and costs occurring once during the project life.

Table 9.1 provides an overview of the single occurring major costs of the project. Equipment costs include factors like fire suppression systems, support and insurance, as well as capital wear parts and spare part packages. These packages are delivered together with the cutter and located on site during the life of the project. Wear parts include additional cutter teeth, mud pumps and suction boxes whereas spare parts can include additional an additional gearbox. The transportation costs include only the transportation to the mine site before production starts. It is estimated that costs of transportation away from the mine site at the end of the project are similar.

Table 9.1: Summary of single occurring costs of the project (modified from [Pers. Comm., Kurszlaukis, 2017]

Cost category	**Cost [Can$]**
Equipment costs, including insurance	$501,960
Training of De Beers personnel	$52,065
Transportation	$2,772,030

It has been proposed that the cutting equipment will be leased by De Beers. Variable costs are summarized in Table 9.2. Distinction is made between standby lease of the cutter, separation plant and mud mixing system and operating lease. The standby lease is applied during months where no production takes place and amounts to 60% of the operational lease.

Table 9.2: Summary of variable costs occurring throughout the project (modified from [Pers. Comm., Kurszlaukis, 2017] and [Pers. Comm. Schwank, 2017b])

Cost category	**Cost [Can$]**	
Equipment operating lease		
cutter incl. base machine	$287,025	per month
separation plant	$70,755	per month
mud mixing system	$9,345	per month
Personnel salaries	$20,000	per day
Wear of consumables	$20	per cubic meter
Power requirements including diesel of cutter (100 L/h)	$157	per hour

In the overview provided by Table 9.2 no processing cost and load and haul costs have been included. However, for the calculation of the total project value and prospected profitability analysis it is suggested to include these operational costs. The processing cost and the load and haul cost can be estimated from the operational data of Victor during previous years and would be in the order of 2 $/tonne and 8 $/tonne for the load and haul, and processing respectively. The application of vertical cutting as an extraction method is likely to cause changes to these costs due to decreased overall tonnages that are being hauled and processed. For this preliminary assessment they have therefore not been included. Personnel salaries are based on a 5 person workforce employed by Bauer but located on site at Victor, it is estimated that an additional 11 employees from De Beers will be required. Production occurs in two 12 hour shifts each day, within this time the cutter is anticipated to cut for 16 hours. The cost of power includes the diesel of the cutter which operates at approximately 100 L/h, as well as a generator for the separation system and the return pump and additional power requirements like lighting.

Due to the arctic weather conditions equipment operating in the Victor mine requires modifications as mentioned in Section 8.2. The costs of for instance an engine block heater and a tank heater have been included in the equipment cost in Table 9.1. In addition to the modifications, diesel operated heaters need to be supplied.

A number of other cost factors has not been included in this investigation. The primary objective of the financial evaluation in this study is the preliminary assessment of costs associated with vertical cutter mining, and the comparison of extraction scenarios. A majority of the costs that have not been included are related to items of factors that apply to all scenarios and to conventional mining and may be reasonably expected to be already present on site. Additional costs that apply to the general operation of the mine and are not directly related to mining should be evaluated separately and have not been mentioned here. Costs that have not been included in the current evaluation, but may be relevant to the overall project of vertical cutting include:

- Taxes, fees, duties, etc.;
- Permitting costs; and
- Water supply for cutter.

Costs of auxiliary processes, like personal protective equipment and welding activities, are considered irrelevant at this stage of the investigation.

9.2 Cash flow model

In order to evaluate the potential application of trench cutting, a basic cash flow will be calculated. In order to do so, assumptions based on income are required. Based on the limited amount of delineation holes and large diameter drillholes a grade estimation has been provided by De Beers for each target. In addition, a value per carat was assumed, based on the 2018 value of the kimberlite ore of the Victor mine. The grades per target and the value per carat have been summarized in Table 9.3.

Table 9.3: Grade estimations and value per carat (2018) (modified from [Pers. Comm., Kurszlaukis, 2017]

Average Grade (Victor Main - northern TL)	0.300 ct/tonne
Average Grade (Victor Main - southern TL)	0.500 ct/tonne
Average Grade (Victor Southwest)	0.200 ct/tonne
Value per Carat (2018)	Can$ 571.45

These assumptions have been chosen arbitrarily, based on experience, and do not necessarily reflect the true situation. Their purpose is to compare scenarios, not to calculate a true revenue.

A dynamic cash flow model which adapts to changes in parameters, e.g. cutting performance, and is also dynamic in time was created. Changes in the project schedule (Chapter 8) are automatically translated into the cash flow model. The cash flow is calculated over the time line presented in Section 8.4 when assuming a cutting performance of 38.5 t/h. When adjusting the cutting performance based on changes in rock strengths the production time can vary from 14 months for very weak rocks (5 – 20 MPa) and 44 for strong rocks (100 MPa). The base case cash flow assumes an average rock strength of 39 MPa.

Table 9.4 summarizes additional parameters for the base case cost model. The base case assumes extraction of the kimberlites at all three targets by applying the checkerboard scenario and without applying backfill. The cutting performance corresponds to the performance estimated based on the cutting performance of the BC 50 cutter (Figure 5.4) for rock with strength 39 MPa and density 2.51 t/m^3.

Table 9.4: Cost model parameters for base case checkerboard extraction

Cutting Performance [t/h]	38.5
Apply Backfilling? (Y/N)	No
Extraction Rate VNW	34%
Extraction Rate VMD	32%
Extraction Rate VSW	30%

Applying the assumed parameters as mentioned in Table 9.4 and the designed project plan presented in Chapter 8 results in a total project duration of 49 months, of which production occurs during 28. Figure 9.1 displays the resulting cumulative cost curve over the full project time.

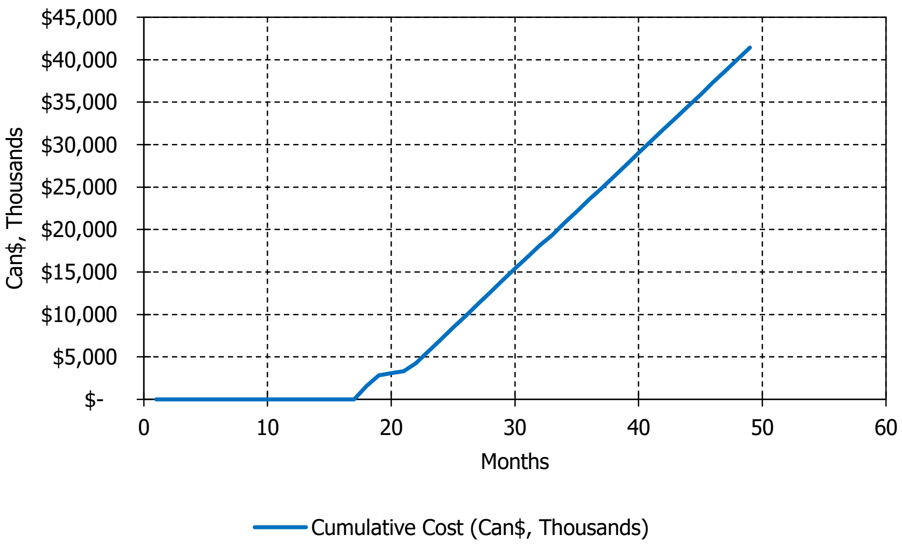


Figure 9.1: Cumulative project cost curve, Can$ (thousands)

During the first 17 months no expenditures are made. All project expenditures are postponed to the end of the manufacturing time of the vertical cutter. Starting from January 2019, when relocation of the cutter system to Victor commences, the cumulative costs sharply increase. Most of the onetime expenditures, summarized in Table 9.1, take place during the start of 2019. From this moment onwards costs accumulate gradually including standby equipment lease cost during transportation and testing months and operating lease from when production starts.

The assumptions of grade and diamond value per carat have been summarized in Table 9.3. Based on these parameters a cumulative revenue and cash flow can be determined, the latter of which is displayed in Figure 9.2. The cost model has the ability to choose what target to extract first based on the grade inside the target. The cumulative cash flow displayed in Figure 9.2 applies this target selection criterion and starts operation in the VMD target zone. The unconstrained grade selection is shown for comparison in red. As can be seen from the cumulative cash flow, not applying the target selection based on the grade postpones revenue from high value targets. Although by the end of the project reaching the same cumulative cash flow, the NPV is significantly lowered by postponing the extraction of the high grade targets.

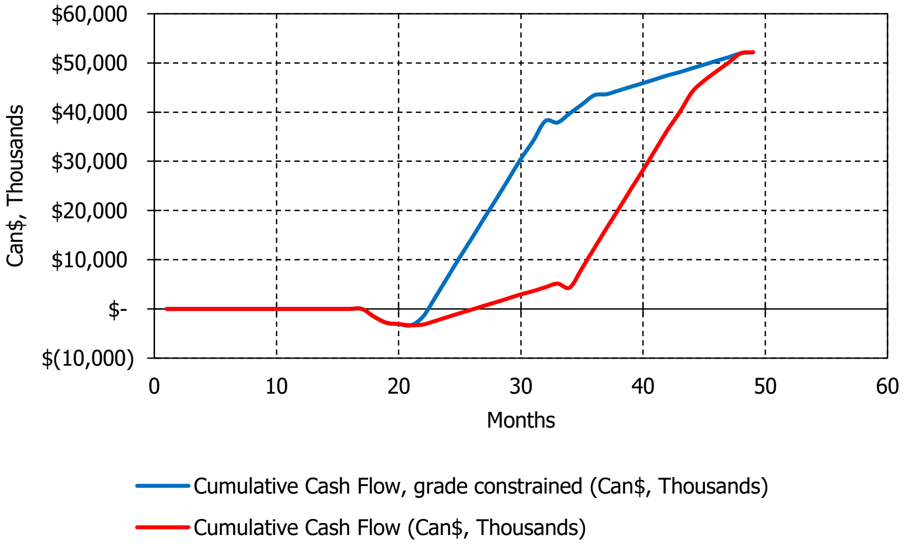


Figure 9.2: Cumulative cash flow curves, Can$ (thousands). The 'grade constrained' curve (blue) applies target selection based on grade inside the target, mining high value ore first.

Figure 9.2 suggests the cumulative monthly revenues based on the assumed parameters eventually surmount the cumulative costs made before and during production. After production has started, the operation requires an estimated 1.5 months for a net zero project cash flow assuming grade based target selection. The cumulative cash flow increases the most during the extraction of the VMD target, and the least during the VSW target which is mined last when selecting the high grade targets to be mined first.

In Appendix D the complete financial model file is included, the file provides all data available for the different cost factors, production rates and extraction rates as well as the cash flow models and sensitivity analyses provided in the report.

In order to account for the time value of money, the NPV has been calculated. Calculating the NPV was deemed especially relevant for scenario comparison. Changing sets of parameters influences the amount of time that is being used for production, or the project as a whole. The NPV and IRR for the base case cash flow model including grade selection, have been calculated to be as follows:

- NPV (Can$, Thousands): $47,657
- IRR: 42%

When mining would start in the lower grade VSW target, the NPV is about half of the value displayed above. The annual discount rate applied for the NPV calculation was set on 8%.

9.3 Sensitivity analysis

The objective of creating the cumulative cost and cash flow curves was to provide a means of expressing the sensitivity of the project costs and revenues on different parameters. A sensitivity analysis has been performed on selected assumptions and will also serve as a method for financial evaluation of the extraction scenarios. The sensitivity analysis has been performed for deposit characteristics and economic parameters separately.

The deposit characteristics sensitivity analysis entails the scenario evaluation for different extraction scenarios. Cutting performance and extraction rates have been regarded in this analysis. Each parameter is changed incrementally while keeping the other parameters constant. A sensitivity analysis is particularly useful to identify the parameter(s) with the most influence on the NPV of the project. On the horizontal axis the percentage of the assumed value of the parameter is plotted, on the vertical axis the corresponding NPV.

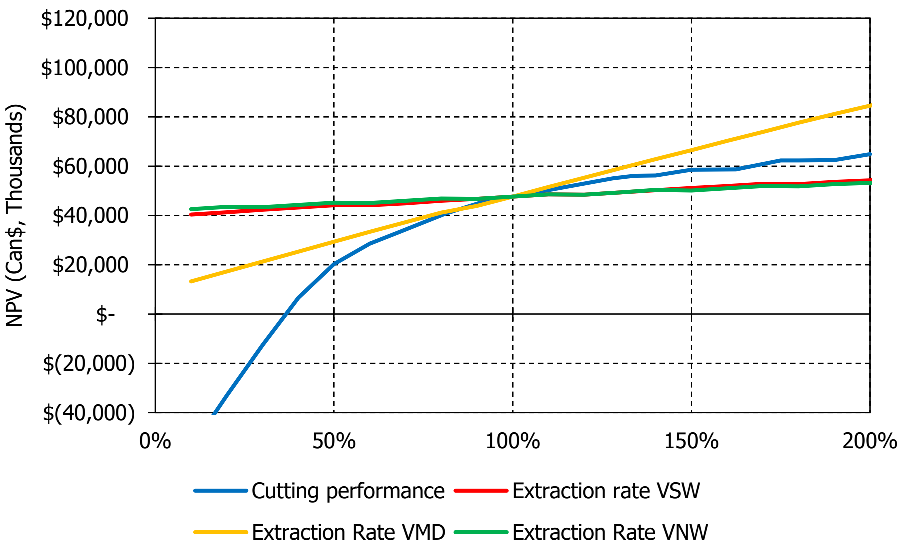


Figure 9.3: Sensitivity analysis for deposit characteristics and extraction variations

Figure 9.3 demonstrates the sensitivity analysis of the NPV in relation with changes in deposit parameters or extraction parameters. It can be concluded that the cutting performance has the great influence on the NPV of the project when it decreases. The extraction rates of the VNW and VSW targets have surprisingly lower influence. The extraction rate of the VMD target, being the both large and of high grade, has the greatest influence of the three targets.

The next paragraphs will address a more detailed analysis of the parameter variations. Paragraph 9.3.1 addresses the extraction rate sensitivity. Because the established extractions scenarios are associated with unique extraction rates, the extraction rate sensitivity analysis also evaluates the scenarios financially. The cutting performance which has the apparent largest influence on the value of the project is discussed in more detail thereafter.

9.3.1 Extraction rates

Significant effort has been done in order maximize the extraction rates during the stability assessment and the optimization of the task schedule. Figure 9.3 suggests that the influence of the extraction rate is, however, rather limited. The figure determines sensitivity of the extraction rates while keeping other parameters constant. Hence, the extraction rate of one specific target is varied while keeping the recovery at the other two constant at the value initially determined. However, increasing the extraction rate at one target location in fact can be expected to cause a similar increase of the extraction rate in the other target locations. For instance, if it is found that the rock stability influenced less than anticipated and trench can become deeper, it is likely that the recovery increases for all targets similarly. Figure 9.3 is, therefore, considered not to be reflective of the true variations in extraction rates. Table 7.5 clearly demonstrates this hypothesis. When the extraction rate at a specific target increases due to a different scenario the other target areas show a similar increase in extraction rate.

It is suggested that the extraction rates of the targets are evaluated together, and not for each target separate. When applying similar variations, evaluating the extraction rate can also serve as the evaluation of different scenarios. Extraction scenarios are largely distinguished of each other by different recoveries and the application of backfill. Financial evaluation of the extraction scenarios can, therefore, be related to the evaluation of the extraction rates. Figure 9.4 displays the average extraction rate per scenario and the NPV of the scenario. The blue dotted line shows a fitted trend between a theoretical extraction rate, one that is applied to each target, and the NPV.

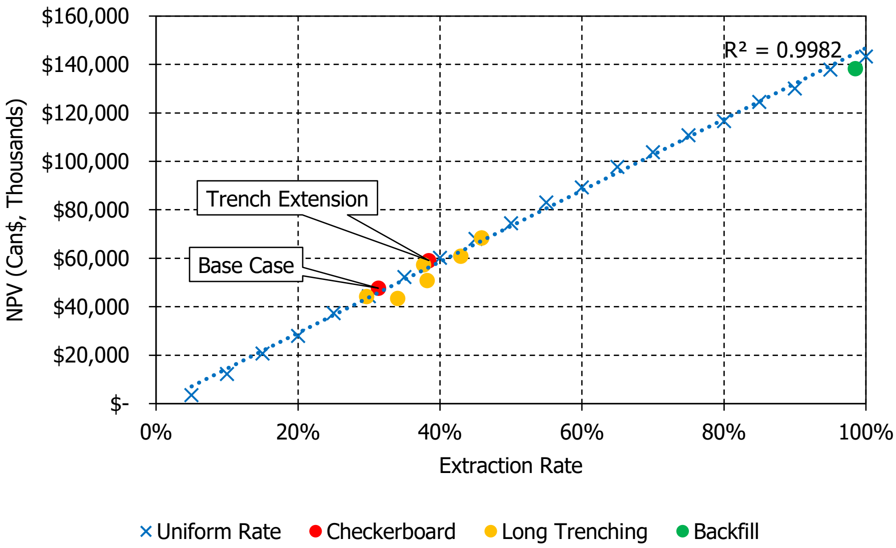


Figure 9.4: Financial evaluation of extraction scenarios

The line that expresses the linear trend between the extraction rate and the project value shows that, every increase in extraction rate results in a rise of the project value. Optimizing the overall extraction rate can have significant influence on the NPV of the project. The trench extension scenario, a variation on the checkerboard, clearly demonstrates this observation. Trench extension increases the average extraction rate of the base case checkerboard with only 6%, but the NPV of the project shows an increase of 24% from the base case.

The long trenching scenario shows considerable variation in extraction rate and value. There are two outliers that do not fit the trend of uniform extraction rates for the three targets. The outliers are a result of the limitation of the maximum cutting depth to improve stability. Extraction at the VMD target greatly influences the NPV of the project, as was already demonstrated by Figure 9.3. The VMD target extends the most in depth, has a high total tonnage and the highest average grade. Reduction of the maximum cutting depth to 72 m, renders more than half of the target inaccessible. The influence of reducing the maximum cutting depth is smaller for the other targets. This is why attempting to improve the stability by reducing the maximum cutting depth does not fit the observed trend, the measure does not affect the recovery in all targets equally. The financial influence of applying backfill to stabilize the long trenches is of limited influence.

The backfilling scenario with the highest extraction rates, returns the highest NPV of the considered scenarios. The value of this scenario lies slightly below the fitted trend line due to the additional costs of backfilling.

9.3.2 Cutting performance

The cutting performance is the parameter that has the single greatest influence on the financial value of the project. Figure 9.3 displays that, when all other parameters are kept constant the NPV of the project would become negative at a cutting performance of 16 t/h (density = 2.51 t/m^3). For the base case cutter size, this cutting performance occurs with a rock strength well above 100 MPa. Technological advancements, like the ones discussed in Chapter 5, can increase the NPV significantly with increasing rock strength (Figure 9.5).

A lower cutting performance is generally caused by stronger rock. Cutting performance in itself evaluates the amount of material that is produced per time unit. It can be related to variations in intact rock strength. However, the sensitivity of the project NPV on variations in rock strength should also include other factors. It is likely that the amount of wear that occurs on the cutting teeth will be greater with a rise in the rock strength or changes in abrasiveness and brittleness, resulting in an additional cost rise. The increase of costs of the wear parts due to higher rock strengths has not been accounted for in this study.

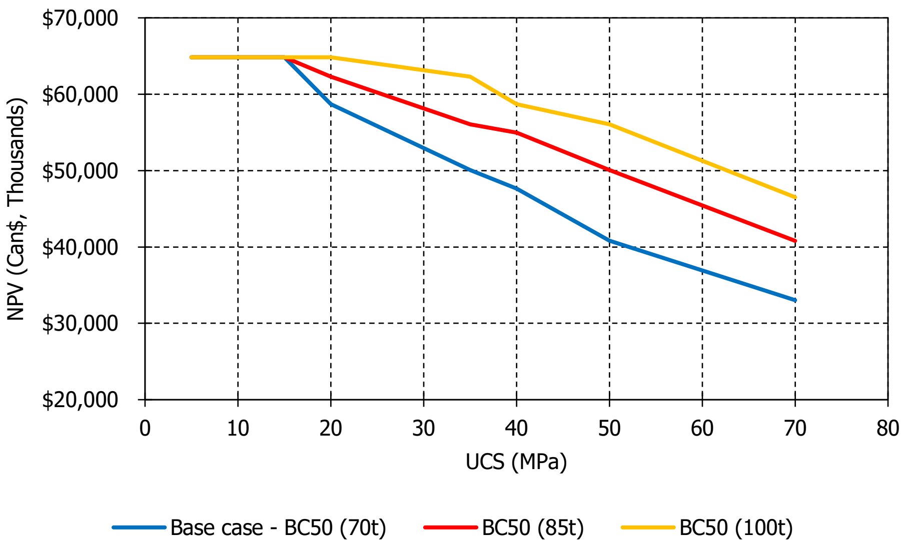


Figure 9.5: Relation between intact rock strength and NPV

Figure 9.5 demonstrates the relation between the intact rock strength and the cutting performance of the BC 50 for its current weight (70 tonnes) and potential developments of increasing the cutter weight without considering changes in wear. A kimberlite rock mass with an intact rock strength of about 20 MPa would return a project value approximately 15% greater than the base case. For all rock strengths below 70 MPa the project has an NPV positive result. According to Figure 9.5 the rock strength is allowed to increase drastically before the NPV reaches zero. The influence of the rock strength on the NPV is surprisingly low. Especially when the cutting performance increases, the NPV virtually remains the same. This is partly due to the fact that the total income remains the same regardless of cutting performance. Greater production rates do not result in greater incomes, only the timing of the income.

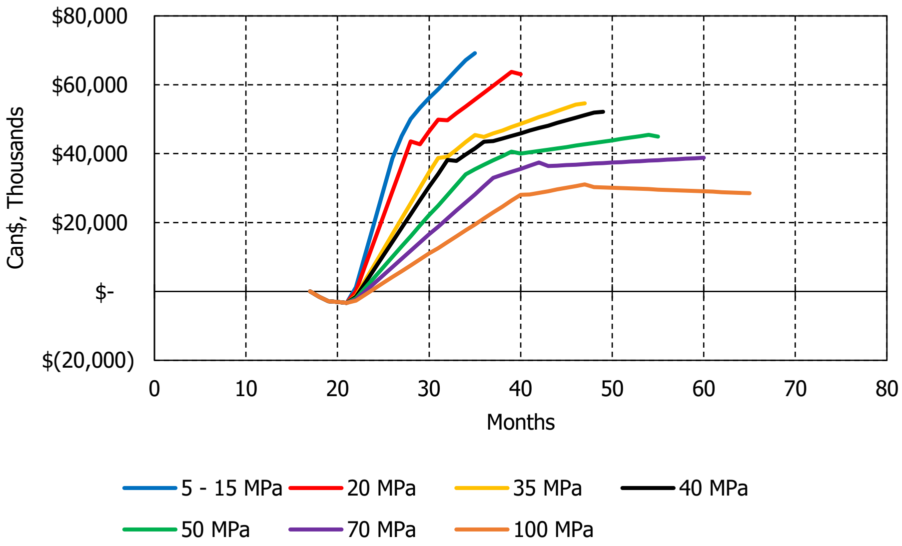


Figure 9.6: Cumulative cash flows for average rock strengths

The influence of the cutting performance on both the time and cumulative cash flow can better be demonstrated by plotting the cumulative cash flows of the different cutting performances (Figure 9.6). An obvious result of greater rock strengths and subsequent lower cutting performances is the increased time required to extract all kimberlite from the targets. When the cutting performance is reduced due to an increase in rock strength, the operating costs will surmount the kimberlite value of lower grade targets as demonstrated by the 100 MPa graph.

Recent reports have shown that the rock strength of the kimberlite in the target zones is in fact 100 MPa and over, instead of the 39 MPa that had been established before. It has been generally accepted that rock strengths of 100 MPa and above are a deal-breaker for the project. Extrapolating the relation between the rock strength and the cutting performance as displayed in Section 5.3 suggests a cutting performance of 21 t/h for the base cutter in 100 MPa rock. Figure 9.6 displays the cumulative cash flow graphs for different cutting performances of the cutter, corresponding with the average strength of the rock. According to the cumulative cash flow for a rock with average rock strength of 100 MPa, a higher strength rock does not necessarily result in a net negative project. However, no additional wear on the cutter teeth has been accounted for.

9.3.3 Economic parameters

The presented sensitivity analyses regarding extraction rate and cutting performance have been based on assumed economic parameters regarding grade and value per carat. The parameters have been chosen arbitrarily and are based on experience. In order to see the influence of these parameters, they are added to the sensitivity analysis for the base case extraction scenario. Figure 9.7 displays the sensitivity analysis for the economic parameters that have been used throughout the financial model.

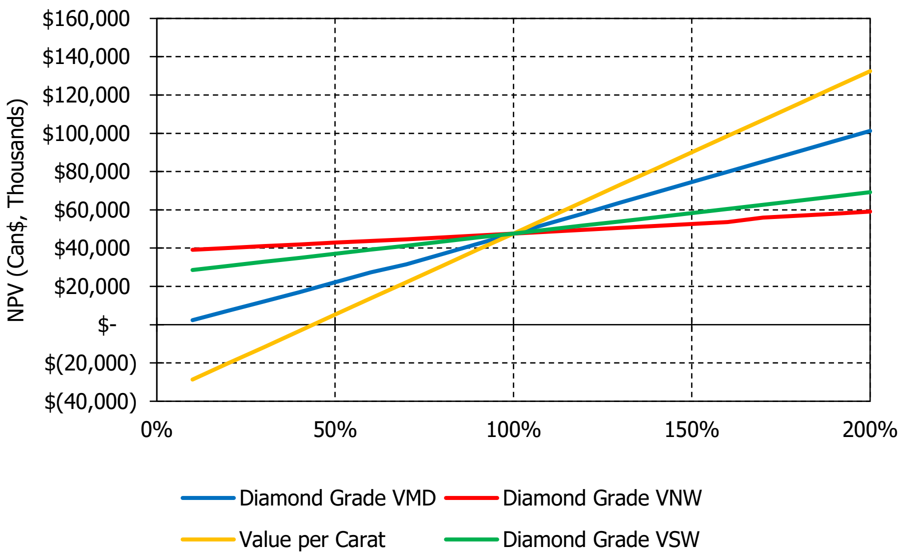


Figure 9.7: Sensitivity analysis for economic parameters

The economic parameters, especially the value per carat, have a significant influence on the calculated NPV's of the project. Their influence is greater than the cutting performance and extraction rates. However, the likelihood of these parameters to double during the project life is limited. The extraction rates of the VNW and VSW targets have virtually the same limited impact on the project value, due to their volume (VNW) or grade (VSW).

Due to the updating of the geotechnical model it is estimated that the UCS of the kimberlite in the target zones is in fact over 100 MPa, instead of 40 MPa. Assuming the updates are correct and the designed extraction scenarios are not subject to change, it has been investigated what values the economic parameters would need to take in order to result in a NPV positive project. Due to the limited influence on the project value of the grades of VNW and VSW no minimum values are applicable for them. Table 9.5 summarizes the minimum values for the grade of the VMD target and the value per carat. Both values are lower than the currently estimated values as the NPV of the project is calculated to be positive, even for kimberlite with average strength 100 MPa. The grade of the VMD target can assume a value about 50% of the estimated grade, the value per carat roughly 69% before the project has a NPV of zero with a cutting performance of 21 t/h.

Table 9.5: Minimum values for economic parameters at 100 MPa rock strength

Average Grade (Victor Main - southern TL)	0.256 ct/tonne
Value per Carat	Can$ 391.57

9.3.4 Production rate

Based on the total operating costs during production the minimum tonnage per month can be determined. The average calculated monthly production costs are $1,400,000. In order to offset the production cost with the revenue, a total of 7,500 tonnes at a grade of 0.300 ct/t are required to be mined monthly. The minimum production requirement corresponds with an hourly cutting performance of 15 t/h. At this production rate no additional costs are recovered by revenue.

9.4 Conventional mining

The previous section of the Financial Evaluation Chapter have demonstrated that vertical cutting is a financially viable alternative mining method. Even for increasing rock strengths this study demonstrates that the project has a positive net present value. In order to demonstrate the preference of vertical cutting over conventional methods the financial evaluation is compared to an assumed cost to extract the same amount of kimberlite that is extracted by vertical cutting.

Chapter 4 addressed the options of extending mine life with conventional mining methods. It was stressed that based on the stripping ratio of 55:1, alternative mining methods are sought after to continue production. The stripping ratio expresses the amount of material of no value that has to be moved for one unit of ore. In general, increasing stripping ratios are in direct relation with elevated costs per tonne of ore mined. Mining operations with stripping ratios in the order of 55:1 are often quickly regarded as unattractive. However, in order to develop vertical cutting as a suitable alternative mining method, the alternative method is compared to the applied conventional methods. The conceptual cut 3 design is used for this comparison. Cut 3, as shown in Chapter 4, extracts all the ore planned for extraction with vertical cutting (including the ore potentially left behind in pillars and crown pillars) and comprises a total volume of 66.5 million m^3. For the pit design, similar design parameters that were applied to cut 2 were assumed.

Assessing mine life extension in full detail is an extensive operation and not fully within the scope of this thesis. However, using order of magnitude estimations provided by for instance CostMine [62], an attempt can be made to quantify the approximate financial implications of continuing operations using conventional methods. Estimates for an operation with stripping ratio 55:1 may not be well represented by extrapolation of costs that are dependent on the stripping ratio. However, for the sake of argument, the mining cost per tonne processed is considered linearly proportionate to the stripping ratio. In order to estimate a mining cost of the Victor Extension, the mining costs per tonne processed of the Gahcho Kué mine are approximated. Gahcho Kué, the new De Beers mine that opened early 2017 in the North-western Territories of Canada, is fairly comparable with Victor in the sense that it is a diamond mine in Canada with similar daily production rates [63].

Gahcho Kué was anticipated to operate at a mining cost of 33.24 $/t processed at a stripping ratio of 10.2:1, or 2.97 $/tonne material. Assuming the Victor pit is able to be extended with a similar mining cost per tonne material, the total cost of mining would be in the order of Can$ 533 million for cut 3. In order to accommodate for the expansion the hauling fleet among other things, would be required to be expanded, involving additional capital costs.

This rough estimation of costs associated with conventional mine life extension is intended to provide a financial justification for the development of alternative methods. Open pit mining is regarded as the conventional that has the lowest operational cost per tonne. However, if open pit mining requires the movement of large volumes of waste material the total cost of conventional mining may significantly arise the anticipated revenue. Vertical cutter mining is able to produce only ore. As such, all costs are directly made for the production of ore, opposed to open pit mining where waste material is required to be moved. The total project cost of vertical cutting amounts to a rough Can$ 41.5 million, less than a tenth of the estimated costs to mine cut 3. The latter cost does not include capital investment of for instance fleet expansion to achieve the operational cost, whereas estimates for vertical cutting currently include transportation, equipment cost in the form of a lease, operation.

10

Environmental and Social Impact Assessment

Conventional mining is a high-impact industry, influencing its surroundings in both positive and negative ways. The spatial footprint of an open pit mine is one of the effects of mining that can be quantified the easiest. Satellite imagery clearly shows the land that is used for mining operations. Corporate responsibility of publicly listed mining companies has grown substantially and reducing the long lasting negative impact of mining operations is one of the opportunities mining companies can continuously strive towards. Deloitte [29] identifies "re-earning the social license to operate" as a clear industry-wide trend. Communities impacted by the mining industry are worried about the potential negative impacts of the mining industry on their environment. Severe mining accidents or physical damage to the environment in the form of chemical pollution comprise only part of the concerns. The emission of greenhouse gasses, noise and dust and water usage are also reasons for public resistance against mining.

Continued growth of the demand for raw materials, however, does not lead to the downsizing of the mining industry [30]. Vertical cutting as a mining methods can accommodate for the deficit between the demand for raw materials and the impact of mining on its surroundings. Mechanical cutting of the rock produces less vibrations and noise due to omitting drilling and blasting from the production system. In addition, a vertical cutting system can replace multiple pieces of mining equipment.

This chapter addresses the environmental and social impact of vertical cutting as a production method of kimberlite. A preliminary assessment of environmental and social indicators for vertical cutting is made. Indicators of interest consist of a CO_2 balance, the spatial footprint of the operation, water usage, operational safety and stake holder engagement.

10.1 Fuel consumption and GHG emissions

The vertical cutting system operates on a diesel engine. Combustion of the engine emits greenhouse gasses into the atmosphere. Carbon dioxide is emitted by burning fossil fuels in for instance the engines of mining trucks, service vehicles and hydraulic shovels. Besides the emission of greenhouse gasses one of the largest input costs of a mining operation are related to the fuel or energy required to extract and process the ore [64]. Reducing the high energy cost of the mining equipment or the dependency of the mining equipment can therefore have substantial positive effects on the balance sheet of a mining operation as well as on the impact on the atmosphere.

Conventional mining in open pit mines operates in a multiple stage process of drilling, blasting, loading and hauling (Figure 10.1). Thereafter, when the mineral ores are transported outside the pit they are processed and concentrated. The production cycle of mineral ores requires the continuous progress of activities for the efficient handling of the material. The activities within this cycle in turn often involve a multitude of machines that perform the work. With mechanical rock cutting several steps can be replaced by one single piece of production equipment. Drilling, rock fracturing (blasting) and crushing can be executed by a single vertical cutter. Elimination of the equipment necessary for the drilling of blast holes, placing of the explosives, loading and crushing has significant positive benefit on the cost of the operation and the emission of greenhouse gasses.

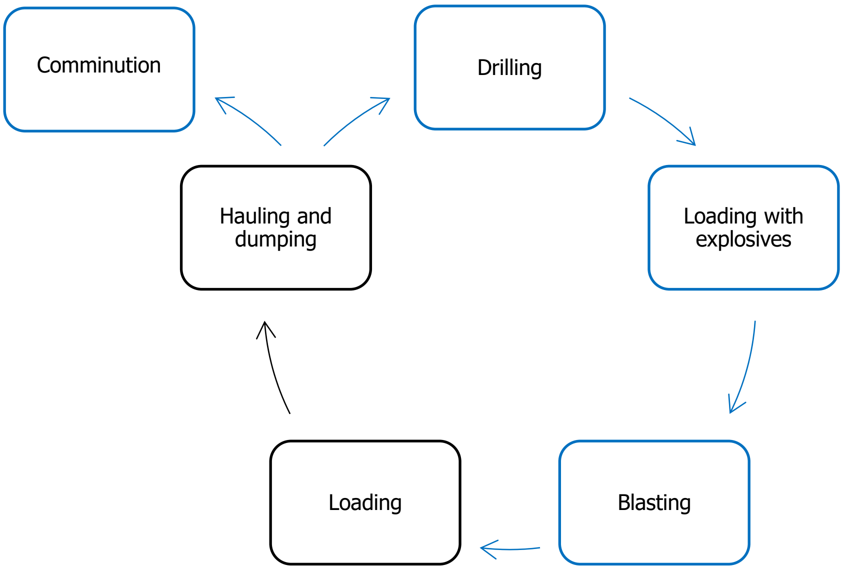


Figure 10.1: Production cycle conventional open pit mines. Processes in blue can be substituted by vertical cutting

Greenhouse gas emissions of the vertical cutter system are estimated to be 1,600 tonnes of CO_2 per year (t/a) for the extraction of the ore. This estimate was based on a diesel consumption of 100 L/h of the vertical cutter, or 2.64 kg/h of CO_2 emissions. For the total extraction and haul process minor additional emissions from a hauler and the micro blasting for the superficial cut would increase the amount of annual CO_2 emissions. A GHG emission rate of 1,600 t/a is virtually negligible compared with the current emissions at Victor. The emissions from fuel consumption were estimated in 2014 at 72,400 tonnes of CO_2 for the main mining operations phase. Roughly 55% of the emissions originate from on-site diesel fuel consumption. Off-site power generation and truck transportation to and from the mine site have comprise the rest of the total GHG emissions. [53]

Crushing is a common first step of the processing of excavated ores. During the crushing step the initial size of the rock is reduced in order to increase the ease of liberating the ore minerals at later steps [65]. The amount of comminution, or crushing, is often a balance between the required energy put in initially to reduce the particle size and the recovery of the mineral ores at a later stage. Comminution is, besides smelting of metal-bearing ores, highly energy intensive [66]. The power used for processing, is supplied from an off-site power plant. The ability of reducing the energy required for comminution due to cutting may translate in off-site GHG emission and energy cost reduction.

10.2 Spatial footprint

Mining activities often compete with other land users over large tracts of land. Especially in more densely populated areas or areas where communities live in the vicinity of the ore deposit there needs to be extensive engagement of the local peoples and room for negotiation regarding impact benefit agreements. More traditional uses of the land like farming but also undisturbed land may suffer infringement due to the arrival of heavy industry, not necessarily due to physical pollution. The reduction of available land on the location of a prospected mine may already provide enough reason for concern.

The spatial footprint of a mine can easily be seen using aerial photographs. Open pit mines require the area of the pit itself but also use land for a number of other purposes like waste disposal, processing and tailings treatment. When applying vertical cutting at late project stages for continued recovery of residual ore the spatial footprint of the existing mine will not be decreased. However, vertical cutting provides the ability to increase the recovery without the need for further pit expansion or increased surface infrastructure.

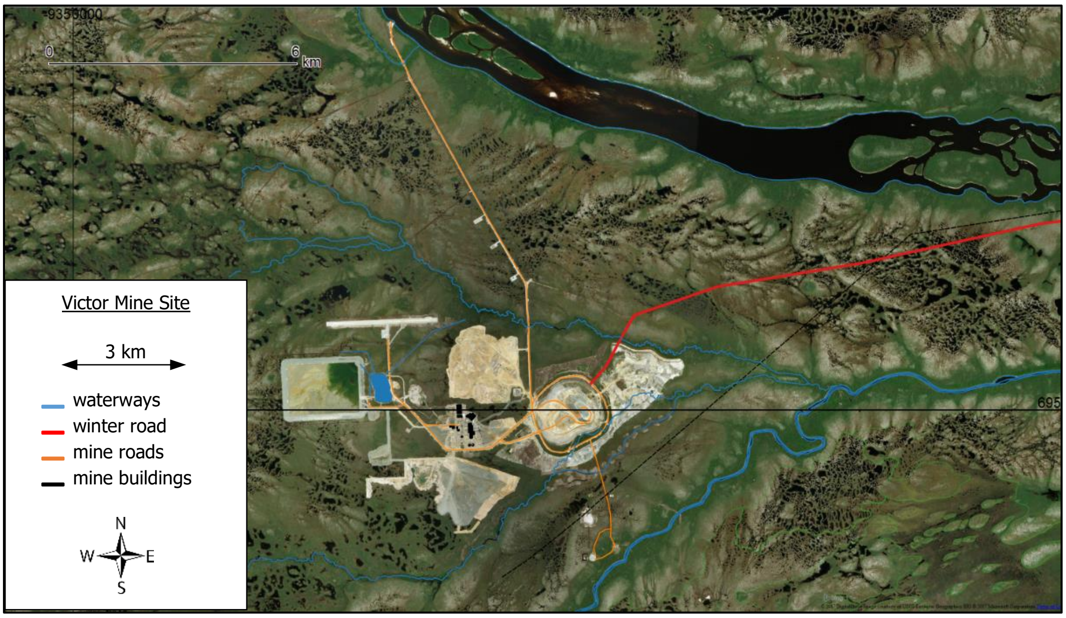


Figure 10.2: Overview Victor Mine site

The enlargement of the open pit is associated with the objective of continuing ore extracting vertically deeper in the ore body. Unless the overall slope angle of the pit can be adjusted, vertical extension of the mine is associated with lateral expansion. Vertical cutting is designed such that lateral expansion is not required. By moving the vertical cutter system inside the pit, so that it is directly on top of the ore no adjustments of the pit slope should be required. Therefore, more ore can be extracted vertically without enlarging the spatial footprint of the mine, preventing removal of the vegetation, top soil and waste rock.

For instance, assume an open pit mine exhibits a perfectly conical shape. Provided that the same slope design parameters can be applied during pit extensions, the pit widens for every extension in the vertical direction. For an overall slope angle of 45° every vertical meter corresponds to an additional horizontal meter in all directions at the crest of the pit. If the average slope angle, or inter-ramp angle (IRA), is 50° like at Victor, extending the pit with an additional bench of height 10 m enlarges the pit with 8.4 m. The conceptual cut 3 (Chapter 4) assumed an increase of 103,000 m^2 in order to extract the identified targets with conventional open pit mining.

10.3 Water usage

Open pit mines have to be dewatered to ensure dry working conditions, maximal efficiency and slope stability [67]. It is common that when creating a large excavation the regional water table will be crossed at some stage during excavation. A ring of dewatering wells surrounding the excavation is able to lower the groundwater table such that the excavation itself remains dry. Besides lowering the water table inside the excavation, the drawdown of the water table also extends into the surrounding lands around the mine. Extensive water pumping can cause harmful impacts including depletion of water resources, pollution of surface watercourses and mine waste leachates [68]. Especially in regions rich in wetlands and waterways, such as the James Bay lowlands, unregulated water pumping can cause long lasting impact on the ecosystem.

During trench cutting the rock that has been cut inside the trench is transported to surface via a mud hose. The trench itself is filled with cutting fluid that serves as the transport medium to bring the fines to surface as well as the stabilizing fluid for the trench walls. In essence, vertical cutter mining only requires the existing open pit to remain dry. The trench cutter, mounted on the base machine, is driven on tracks inside the pit which should therefore be kept dry. The trenches are below the water table and therefore wet. The groundwater can directly be applied as cutting and stabilizing fluid, making it redundant to further lower the groundwater table. Additional water that is located within the hose system and desanding units on surface can originate from the pumping wells around the excavation.

In Chapter 5 variations in the base machine have been addressed. One of the methods of extraction was identified as cutting using a barge-mounted system. At Victor this scenario was decided to have too great of an impact on other in pit processes. In addition, barge-mounted cutting requires different evaluation methods of stability, and financials and was therefore not considered for the Victor case. However, it is worth mentioning that with vertical cutting from a floating vessel, part of the open pit is allowed to be filled again with water. Partial infill of the pit with water allows for the decrease in the pumping rate at the mine dewatering wells, potentially providing considerable reduction in costs and environmental impact earlier in the project stage. Barge-mounted cutting is the extraction scenario that may lead to significant reduction in water abstraction with its associated costs, if found applicable.

10.4 Operational safety

Operational safety is and has been for little over a century, a primary concern for mining companies. Safety statistics have made enormous improvement since the beginning of the previous century [69]. Changes in the nature of the work, and continuous improvement of the safety conditions at a work place have contributed to the decrease in serious injury rates and fatalities [70, 71].

Mechanical cutting is often reported as a safe method in underground mining. Mechanical cutting replaces the need for explosives, can create smaller underground openings, can install support measures during extraction and combines drilling, loading, excavation and hauling equipment inside the mine (adapted from Vogt [31]). In addition, due to the high level of mechanization virtually no personnel is required to enter the production locations, keeping them away from the hazardous environments.

According to the ICMM [72], 70% of the mining fatalities in 2015 could be attributed to one of the following three factors:

- Fall of ground in underground mines

- Machinery
- Transportation

Although safety in mining continues to improve each year, not a single fatality is accepted by the industry or society. Mechanical rock cutting may contribute to the further improvement of the safety conditions in an underground mine, preventing employees to be hit by 'fall of ground' and reducing the amount of transport equipment and other machinery.

Mechanical cutting is currently mostly applied in underground mines. However, vertical cutter mining may provide similar risk reduction measures in open pit mining. Vertical cutting reduces the required fleet in the mining operation. Besides a reduction in required energy, the decreased amount of mobile equipment can contribute to the reduction of the machinery/transportation related accidents. In theory, for some operations, vertical cutting may eliminate large scale hauling completely by maximizing transport through mud hoses.

The handling or detonation of explosives has not been reported by ICMM as a cause of fatalities in the mining industry for 2015. However, in 2013 and 2016 accidents with fatal injury occurred in respectively 4 and 1 cases, in the US alone [73]. The handling of explosives and rock blasting is still a hazardous step in the mining cycle. But besides fatalities, blasting has much more impact on its surroundings and workforce. Noise and vibration may impact the long term health of the employees as well as reduce their ability to work efficiently [70]. Although a majority of these impacts can be mitigated by the proper closure of the blast area, it appears many accidents occur due to failure to do so [74, 75]. In addition, closure of the blast area requires the complete area to be evacuated, interrupting production during that time. Vertical cutting in essence eliminates blasting, unless the superficial cuts need to be trench-blasted (Section 5.2). The application of trench blasting techniques should be further investigated, however, due to the scale of these blasts it is expected the blast area and safety risk is rather limited.

10.5 Stakeholder engagement

Communities that surround a mining operation are always affected. In most cases communities have been present long before the mining operation arrives. Evidently this implies that with the arrival of a mine, land is freed up and local inhabitants will notice an increased amount of activity. A large portion of the effects that reach the communities can be very positive. The creation of a job opportunities and engagement of local contractors are two ways economic benefit can flow back to the local communities.

The total group of stakeholders is much larger than just affected communities. Governments, NGOs, contractors, employees as well as on a larger scale the corporation are just some of the stakeholders that could be applied on any Canadian diamond mine. For many of the stakeholders continued operation and sustained high value output of diamonds is a desired outcome of mine expansion. However due to increase in social responsibility of all stakeholders, this desired outcome is put into perspective with other impacts.

The inability to obtain buy-in from the community of Attawapiskat FN illustrates the level of influence stakeholders have on a mining operation. Expansion of the mine either outward, which is unattractive due to increased stripping ratios and increased required water pumping, or to another pipe in proximity of Victor cannot take place without support of Attawapiskat FN [40]. However, large cutbacks in workforce as well as reduced revenue are no obvious decisions for as long as the Victor pipe and its surrounding pipes are rich in kimberlite ore. Obtaining or sustaining the social license to operate can appear a challenge for mining operations all over the world. Stakeholders have the influence to close mining operations in extreme cases like the Bougainville copper operations in 1989 as well as initiate lengthy evaluation periods even when production has occurred in a specific area for over 20 years [76].

Vertical cutting is not expected to provide an amount of personnel positions to sustain the current workforce. However, by being able to sustain the production of high quality kimberlites without increasing or even reducing the mines impact, backing of the stakeholder may increase for continued mining.

11

Discussion

This study intended to demonstrate the potential of vertical cutter mining as a complementary mining method at a late project stage. In order to do the technical feasibility, viability and the environmental benefit have been investigated for the Victor Diamond Mine, Canada. A large portion of this research has been based on geotechnical data of samples located outside the zone of interest, with a relative low data density and with expected mistakes. Application of the geotechnical data required simplification of the geotechnical data. Simplification of the geotechnical data influences the prospected cutting performance as well as the models used in order to assess the pit stability.

The financial evaluation has been performed with the objective of comparing different extraction scenarios and in order to assess the sensitivity of the project on specific parameters. In truth, the financial evaluation of implementing vertical cutting technology at the Victor Diamond mine is dependent on a vast and complex set of costs. It is expected that the operation as a whole requires adjustments to accommodate vertical cutting. The processing plant, which was not included within the scope of this study, is expected to be one of the production steps needing substantial adjustment.

11.1 Strength analysis

The geotechnical assessment of the Victor pipes was based on the combination of three datasets involving the rock strength. The geotechnical core logging, UCS measurements and the Point Load Test data provided the data on the kimberlite strength. The three data sets displayed notable inaccuracies that rendered them not ideal to assess an intact rock strength distribution that can be applied throughout the target zones of Victor. However, it is believed that the combination of the three provides the best possible depiction of the rock strength that can currently be made.

The determination of the conversion factor K that calculates the UCS from the point load index strength is highly debated. The conversion factor can take a number of values theoretically ranging between 14 and 24 according Singh et al. [51] or 20 and 25 according ISRM standards [77]. A conversion factor of 20 has been determined based on 4 samples within comparable range of borehole depth and geotechnical assessment of other kimberlite pipes in the Attawapiskat cluster [52]. It is judged that the obtained distribution of rock strengths provides a decent representation of the expected strength distribution at Victor, corresponding to the geotechnical drillhole data.

The total number of strength tests is far greater than merely the tests on kimberlites described in this study. A large portion of the open pit slopes have been created in country rock. The mudstone and limestone surrounding the pipes has been extensively studied for the purpose of slope design. Because there is no correlation between the strength of the country rock and the strength of the kimberlites, no country rock samples have been analysed for vertical cutting. The difference in strength between the kimberlite and the granitic basement can be important in order to determine whether a trench has been cut to full depth. By analysing the production rates of the cutter one would be able to determine the bottom of the trench due to a change in the cutting performance.

The fact that little to no samples were collected for strength testing were directly located inside the target zones was initially estimated to be of little influence to the strength assessment. Due to textural differences in the kimberlite facies some increase in the rock strength was expected towards the root zone of the pipes, however, not in the order of magnitude established on the updated 2017 strength samples. This study has established that high rock strengths of 100 MPa are not necessarily reason to believe that cutting cannot take place at Victor. However, the doubling of the average rock strength clearly illustrates the uncertainty of the kimberlite intact rock strength at Victor and can be cause for the need of further research.

Kimberlites often contain a high portion of xenoliths. The presence of xenolith samples on which point load testing was performed has been investigated during this research by comparing the geology logs and geotechnical logs. It has not been found that the presented measurements were performed on xenolithic rock samples. However, xenoliths may still be present in the kimberlite rock units, with potentially greater intact rock strengths. Although this study has only considered average strengths and average rock strengths of the kimberlite, the presence of xenoliths and their IRS will affect the overall cuttability of the rock.

11.2 Production performance

The cutting performance estimation provides the basis for the production rate that is expected. The cutting performance has been determined to be predominantly dependent on the intact rock strength of the rock. In reality, cutting performances are dependent on a complex combination of other factors as well. Abrasiveness and brittleness are two of the most common rock properties referred to in literature to be also of influence on cutting [31].

Average rock strengths have been calculated in order to determine cutting performances. The influence of the rock strength on the NPV of the project has been determined with a sensitivity analysis. However, the sensitivity analysis again assumes a uniform rock strength for all target zones at Victor. Cutting performance is rarely determined by the average rock strength. The rock fragment of the highest strength will determine the cutting performance [Pers. Comm., Rougier, 2017]. In general, it is expected that the spatial variability of the strength within the kimberlite of a facies, is rather limited. The strength variability as a result of the presence of xenoliths is likely to be greater. For the scope of this research a cutting performance based on an average rock strength should provide a satisfactory assessment of productivity.

11.3 FE model validation

The finite element modelling comprised an important tool for the macro scale stability assessment. Finite element modelling highlighted the risk of slope failure in the existing pit slopes when undercutting the slope with a trench. Some of the limitations of the used FE model have already been mentioned with the model creation, the most important ones being:

- The analysis of the stresses and strains in 2D (plane strain) cannot accommodate for rock interactions that occur around the excavation in 3D. Especially with cuts the size of one primary panel, it is expected that analysis of the stresses in 3D will prove to be highly valuable for the overall stability assessment.
- Finite element modelling as applied with this study assumed uniform rock conditions. Changing rock strengths or elastic moduli and jointing have not been accounted for.

The model has been validated to ensure applicability of the results. Rock properties of the kimberlite have been validated in order to check whether the rock behaves as anticipated according material type. Secondly, the macro scale simulation has been validated according the determined factor of safety for the pit slopes that has been determined during previous slope design investigations. Rock property validation has been performed based on a simulated UCS measurement. A rectangular rock sample of 0.25 by 1 m was modelled in RS^2. Residual rock properties were assumed at a disturbance factor of 0.7 of the peak rock properties [78, 79]. Stress-strain curves of the simulated rock cylinder suggest the rock behaves as anticipated within the models. The curves and a more detailed analysis are shown in Appendix C.

The macro stability has been calibrated based on the designed ultimate pit without a trench. It was attempted to recreate the pit, including the geology, in the finite element software using the similar parameters that were applied during the slope stability assessment performed by Golder [57]. The conclusions of the 2016 pit stability investigation highlighted a factor of safety for the ultimate pit varying between 1.7 and 1.8. The calculated factor of safety of the pit cross-section 1-1' was determined to be slightly higher, at 2.28 during this research. Section 1-1' is in a different location than some of the simulations performed in 2016. It is judged that the factors of safety are within reasonably comparable magnitudes. Determination of the exact parameters has not always been clearly stated during for the 'updated stability analyses' [57], however, it is suggested that recent geological drillholes provide the basis for a range of GSI values. It has been chosen to apply the same parameters for all bedrock formations due to the current stable conditions in the mine. Kimberlite rock parameters have been adjusted towards the averages obtained from the geotechnical assessment reported in this study. This involved the slight decrease of the intact rock strength (MPa), unit weight (kN/m^3) and elastic properties. For the limit equilibrium analysis and finite element analysis of circular and non-circular failure planes Golder [57] applied bedding and cross-joints to the limestone and Red Head Rapids formations. No bedding and cross-joints have been included in this study.

11.4 Project financials

For the purpose of this study a limited financial evaluation was performed. The objective of the financial evaluation was to establish project sensitivity on defined parameters and evaluation of the extraction scenarios. It is judged that a number of cost factors are missing from the currently presented evaluation. This research has attempted to find a balance between reporting project costs and costs that are relevant for involved parties. This research does not provide a true profitability analysis. It is expected that the true total projects costs will increase compared with the model presented in Chapter 9.

Sensitivity analyses have shown that the influence of the cutting performance is rather limited with increasing rock strengths. It was expected that the sensitivity would show different results for the productivity of the cutting system. Inclusion of project costs on a more detailed level may increase the influence of the cutting performance.

11.5 Environmental and social impact

The environmental and social impact assessment has been based on a theoretical application of vertical cutting at the Victor site. During this study no measurements were done towards the current impact of the mining operation. All data of the Victor mine site originates from published and unpublished reports. This study, however, demonstrates the significant difference between the impact of conventional mining methods and vertical cutting.

Calculations of the total GHG emissions have been based on an average fuel consumption rate of 100 L/h. It is expected that the total amount of emitted greenhouse gasses may increase due to auxiliary processes. Comparison of the calculated 2014 total emissions and the emissions of the vertical cutter demonstrate the gains that can be made by developing alternative mining methods while sustaining the output of ores. GHG emission considerations, as well as other environmental impacts have been assessed assuming a single operating cutting system. Average tonnages produced by cutting differ significantly from conventional open pit mining, environmental and safety benefits must be viewed in this light.

12
Conclusion

The purpose of this thesis is to aid in the development of vertical cutting mining as a new mining method. In order to do so the objective was to investigate the technical feasibility and viability, and environmental benefit of vertical cutting as a complementary mining method. The investigation was performed for the case of the Victor Diamond mine in northern Ontario, Canada. Open pit mining is planned to end when the ultimate pit is reached, although additional kimberlite resources exist below the pit limits. The costs of mine closure, site abandonment and potential future reopening of the site, are expected to surmount the residual value significantly. Therefore, all kimberlite resources are desired to be extracted fully before closure.

Conventional mining methods do not provide the solution to sustain extraction of the kimberlite. There is a clear need of alternative methods that can produce ore without the need of enlarging the mine or of constructing extensive underground infrastructure. The amount of residual resources is limited such that the investments cannot be justified.

Cutting technology has experienced a surge in development for underground mining in soft rock conditions. Vertical cutting is intended to provide the mechanical cutting solution for surface mines. By extracting the ore with vertical cuts from directly on top of the ore, no large quantities of waste rock have to be moved. Four extraction scenarios have been designed with varying recovery. The base case checkerboard scenario, achieves roughly 30% recovery by creating a series of non-connected trenches. Long trenching is a variation of checkerboard mining, where all cuts are connected in rows by secondary trenches. In between the rows, kimberlite remains in so-called crown pillars. Extraction of the crown pillars requires the backfilling of the previously cut trenches, comprising the third extraction scenario. The fourth option involves the flooding of the pit and extracting the ore with a trench cutter mounted on a barge.

For the Victor mine three scenarios are considered relevant. Pit flooding impacts the current processes in the mine too greatly to be of interest at this stage. Vertical cutting can achieve average extraction rates varying between 30% and 46%, and up to 98% when backfill is applied. The creation of long trenches imposes severe risk of instability in the rock surrounding the cuts. Instability of the existing pit slopes as well as the collapse of the crown pillars are expected when long, interconnected trenches are created. Measures to improve stability exist but involve decreased recovery or increased secondary costs. The base case checkerboard scenario is expected to have less influence on rock stability due to the interconnected kimberlite in between the trenches and is expected to be an efficient and safe extraction method. Especially when limited trench extension is applied, checkerboard mining proves to be an attractive mining scenario for vertical cutting.

Financial evaluation of the extraction scenarios and cutting performance was specifically performed in order to determine the sensitivity of the project value on the mentioned parameters. Cutting performance has great influence on the NPV of the project, especially when the production rate decreases. It is commonly referred to that rock strengths greater than 100 MPa are a deal-breaker for commercial ore production using mechanical cutting. The sensitivity analysis performed during this study did not indicate project failure when the rock strength of the kimberlite approaches 100 MPa. However, the wear rate of the cutter consumables have been calculated per unit volume produced. The wear rate may increase if the rock strength, abrasiveness and brittleness change. Evaluation of the extraction scenarios has shown that all scenarios provide a positive NPV. In essence, based on the linear relation between the material recovery and NPV it is concluded that every tonne of extracted material results in an increased project value.

Assessment of the environmental and social impact indicators suggests that vertical cutting does not lead to an increased negative impact on the environment for all indicators. In the case of GHG emissions and energy consumption vertical cutting reduces emissions significantly by combining several conventional production processes in one operating machine. It is anticipated that the support basis by stakeholders will expand in comparison with conventional mining methods. Without increased negative impacts and sustained output of the highly valued Victor diamonds, vertical cutting may provide the solution to prevent mine closure by the end of 2018 or beginning of 2019 for the Victor mine. Technical feasibility assessment and financial evaluation of the project justify future and more detailed studies into the development of vertical cutting as an alternative mining method for the Victor mine as well as other kimberlite deposits.

13 Recommendations

In order to further develop vertical cutting as a mining method future research is required. It has been determined that large scale investment for the complete application of vertical cutter mining at Victor, will be postponed due to associated risk of the project. Reducing the risk is paramount before trial mining can occur. This thesis has clearly established that the project value is sensitive to cutting performance and extraction rates. Reducing the risk associated with vertical cutting will require cutting trials over extended volumes. Cutting performances are estimated based on interpolated data. Assessment of the validity of the interpolations will require extensive testing in different rock conditions.

Intact rock strength, which in turn influences the cutting performance, is investigated to a limited level in hard rock open pit mines. Design of the open pit slopes and blast design are commonly the only studies requiring intact rock strengths. For these studies a limited amount of samples in addition to geological observations suffice. Assessment of the feasibility of vertical cutting requires an extensive study of rock strengths, abrasiveness and brittleness per geological zone, as well as the presence of xenoliths and their properties when mining kimberlites.

It is recommended that the level of detail of the stability assessments is increased. This thesis has highlighted the risk of slope and pillar instability due to cutting. Further investigation into the rock stability should greatly increase the level of detail and quantification of failure risk, as well as possible mitigation methods. An important addition to the stability assessment is expected to be the 3D simulation of the rock behaviour. A limitation of this study was the assessment of slope stability in plane strain. Study of the slope stability in 3D may suggest that the rock mass surrounding the trenches is less affected by the cuts. The checkerboard scenario, which has been referred to as stable due to the rock bridges in between the trenches, should be assessed similarly in 3D.

Lastly, it is recommended that financial evaluation is performed for a greenfield vertical cutter mining project. The aim of this thesis was to establish sensitivity of the project value to several parameters and compare extraction scenarios. However, it is considered that the net present value may be overestimated due to the exclusion of several cost factors. Performing a financial evaluation on a greenfield project should provide a more detailed and better estimation of the costs of developing vertical cutting as a mining method. The Tango Extension kimberlite, located in the vicinity of the Victor mine, may provide an ideal ore body to perform such study on.

Barge mounted cutting which was considered to be too invasive for current operations at Victor, may provide to be of great value for the development of vertical cutter mining at the Tango pipe but also other diamond deposits around the globe. The presence of water inside the pit provides additional stability to the pit slopes decreasing the risk of macro scale deformation. Considering the benefits for environment, communities and company, the recovery of residual resources from flooded open pit mines with vertical cutting may prove valuable.

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Personal Communication

Personal communication has been cited, stating the name, function and company of the person the communication was with, and on what date the communication occurred. Communication cited according order in the text:

- S. K. Schwank (Exec. Dir. Mining Solutions, Bauer Maschinen GmbH), 17 March 2017a
- H. Qureshi (Mine Planning Analyst, De Beers Group Canada), 1 June 2017
- B. Rausch (MHEPO Consultancy), 31 May 2017
- M. Rougier (P. Eng., Golder Associates Ltd.), 6 June 2017
- S. Kurszlaukis (P. Geo. Kimberlite Petrology Unit, De Beers Group Canada), 31 January 2017
- S. K. Schwank (Exec. Dir. Mining Solutions, Bauer Maschinen GmbH), 17 August 2017b

A

Surface Area Evaluation

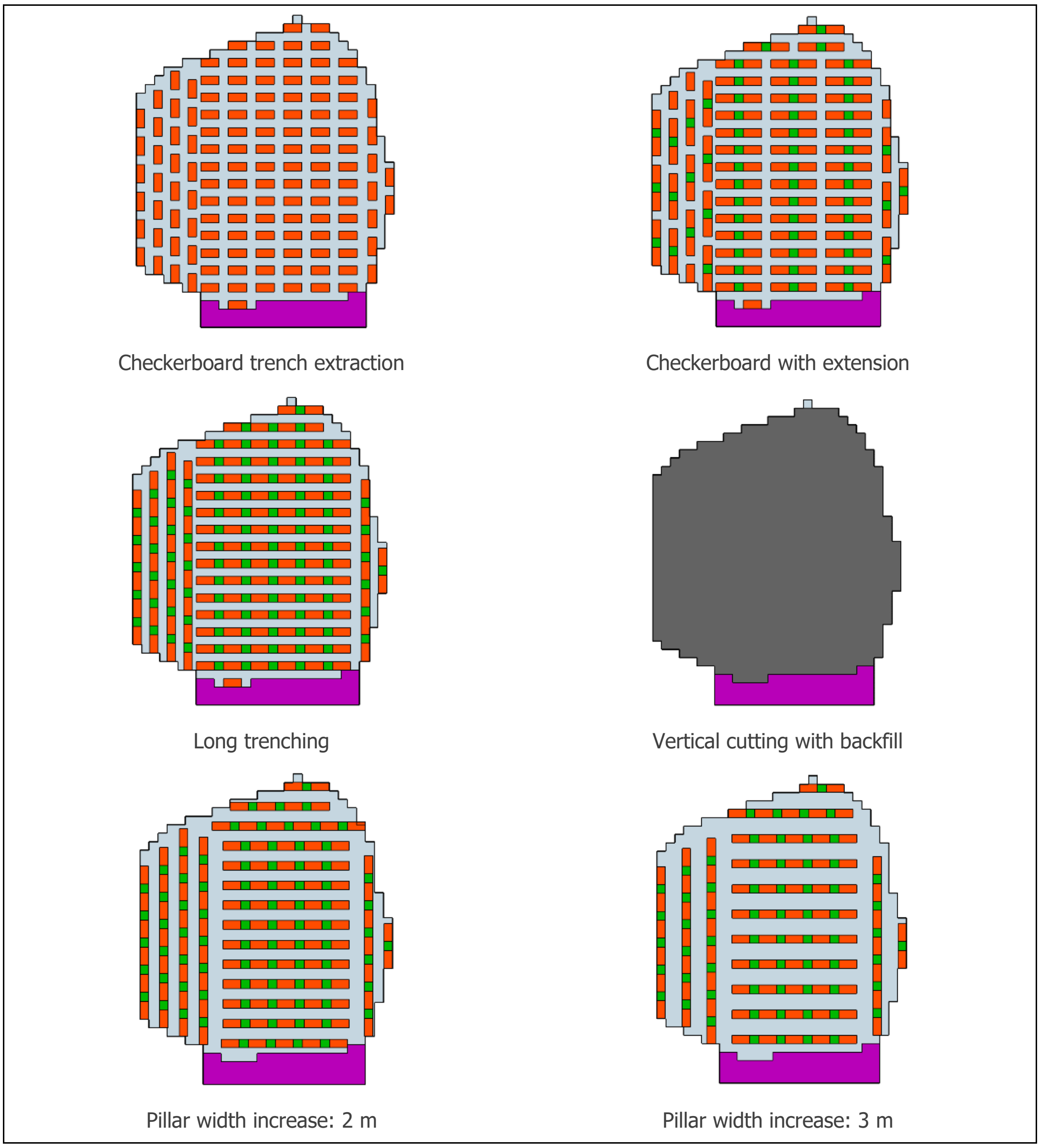
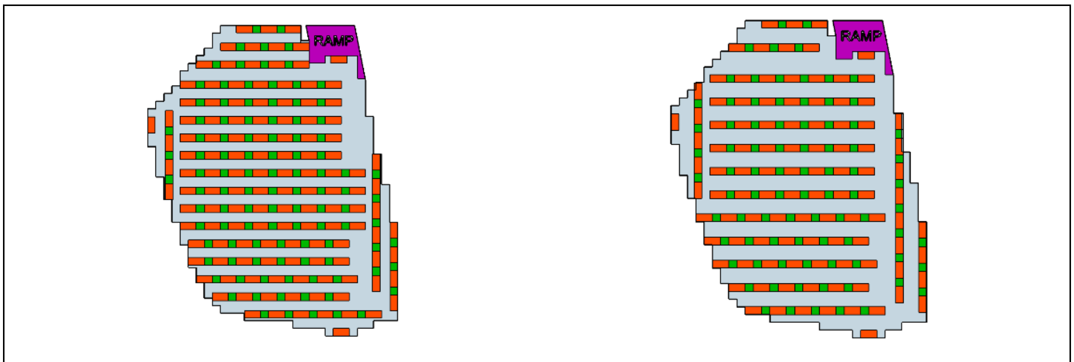


Figure 1: Schematic representation of the extraction scenarios at the VNW target. Primary trenches are displayed in orange, secondary trenches in green, unrecovered material in light blue, the ramp in purple and backfill in dark grey.



Pillar width increase: 2 m   Pillar width increase: 3 m

Figure 2: Schematic representation of the variations in pillar width for the long trenching scenario at the VMD target

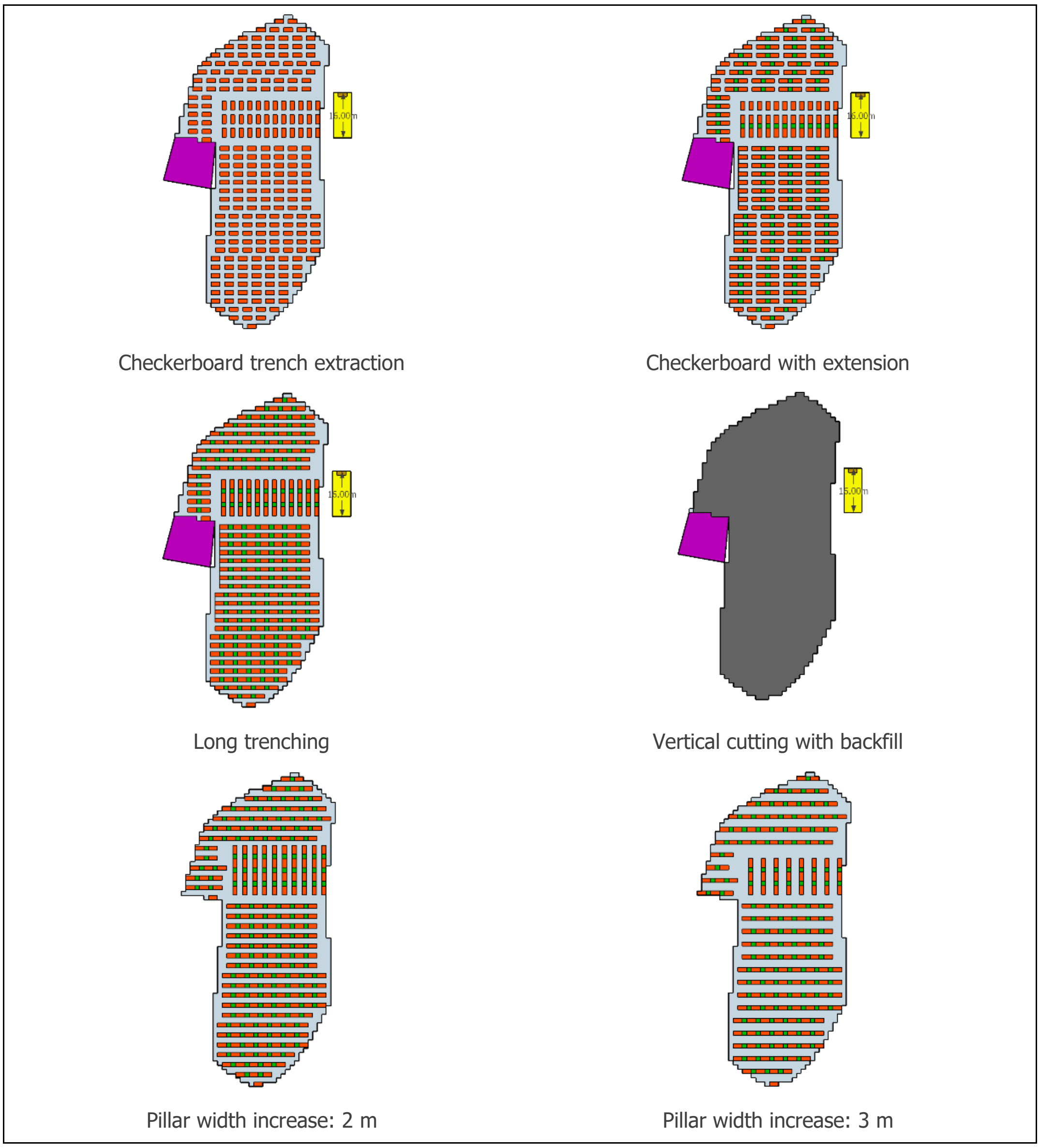


Figure 3: Schematic representation of the extraction scenarios at the VSW target. Primary trenches are displayed in orange, secondary trenches in green, unrecovered material in light blue, the ramp in purple and backfill in dark grey.

B

Material Properties

Table 1: Field stress as simulated in RS^2, from [59]

Field stress Type	Gravity
Total Stress Ratio (horiz/vert in plane)	2
Total Stress Ratio (horiz/vert out-of-plane)	1.5
Locked-in horizontal stress (in plane) (MPa)	5
Locked-in horizontal stress (out-of-plane) (MPa)	1

Table 2: Kimberlite rock properties as simulated in RS^2, modified from [57]

Rock Name	**Kimberlite**
Unit weight (MN/m^3)	0.0251
Failure Criterion	Generalised Hoek-Brown
Intact Comp. strength (MPa)	39
Geological Strength Index (peak)	65
Intact Rock constant mi (peak)	10
Disturbance Factor (peak)	0
Young's Modulus (MPa)	6450
Poisson's Ratio	0.25

Table 3: Basement rock properties as simulated in RS^2, from [57]

Rock Name	**Basement**
Unit weight (MN/m^3)	0.027
Failure Criterion	Generalised Hoek-Brown
Intact Comp. strength (MPa)	75
Geological Strength Index (peak)	65
Intact Rock constant mi (peak)	10
Disturbance Factor (peak)	0
Young's Modulus (MPa)	24000
Poisson's Ratio	0.3

Table 4: Overburden properties as simulated in RS², from [57]

Rock Name	Overburden
Unit weight (MN/m³)	0.019
Failure Criterion	Mohr Coulomb
Tensile strength (MPa)	0
Friction Angle (deg.)	35
Cohesion (MPa)	0
Young's Modulus (MPa)	200
Poisson's Ratio	0.3

Table 5 Limestone rock properties as simulated in RS², from [57]

Rock Name	Limestone
Unit weight (MN/m³)	0.027
Failure Criterion	Generalised Hoek-Brown
Intact Comp. strength (MPa)	45
Geological Strength Index (peak)	45
Intact Rock constant mi (peak)	10
Disturbance Factor (peak)	0
Young's Modulus (MPa)	24000
Poisson's Ratio	0.3

Table 6: Red Head Rapids rock properties as simulated in RS², from [57]

Rock Name	Red Head Rapids
Unit weight (MN/m³)	0.027
Failure Criterion	Generalised Hoek-Brown
Intact Comp. strength (MPa)	16
Geological Strength Index (peak)	40
Intact Rock constant mi (peak)	10
Disturbance Factor (peak)	0
Young's Modulus (MPa)	5000
Poisson's Ratio	0.3

Table 7: FE Model dimensions for macro scale stability assessment

Section	Span X	Span Y
1-1'	1700 m	1650 m
2-2'	1700 m	1650 m
3-3'	1700 m	1650 m
4-4'	1700 m	1650 m

C

FE Simulation Results

For the stability assessment performed during this research finite element software was used. Finite element analysis for the determination of rock behaviour numerically computes the forces, stresses and movement of rock. The method divides a macro scale problem into smaller parts which are referred to as the finite elements.

For the finite element analysis performed during this thesis research the software RS^2 from Rocscience Inc was used. The objective of the analysis was to determine the influence of undercutting a mine slope with a trench to maximum trench or resource depth. It was expected that the creation of trenches at the pit bottom would cause deformations of the slopes causing potential instability. Simulations as described in Section 7.3 indicated the likelihood of such deformations to occur. This appendix includes the FE analysis results for additional scenarios that were investigated as well as the results of the cross sections other than 1-1′. In addition this appendix includes validation of the rock behaviour based on stress-strain curves of a simulated UCS measurement.

Stability Measures

The following three figures show the maximum shear strain for the critical slope of section 1-1′. Figures have been shown for SRF = 1, implying with the rock properties as given to the model.

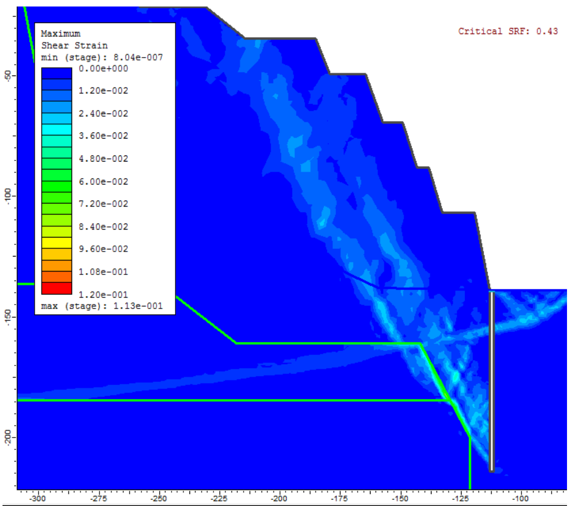


Figure 1: Maximum shear strain with SRF =1 in critical slope of section 1-1′, first meter of trench reinforced with liner (Young's modulus = 30000 MPa, Poisson's ratio = 0.2, thickness = 0.1 m). SRF calculated by RS^2 is 0.43.

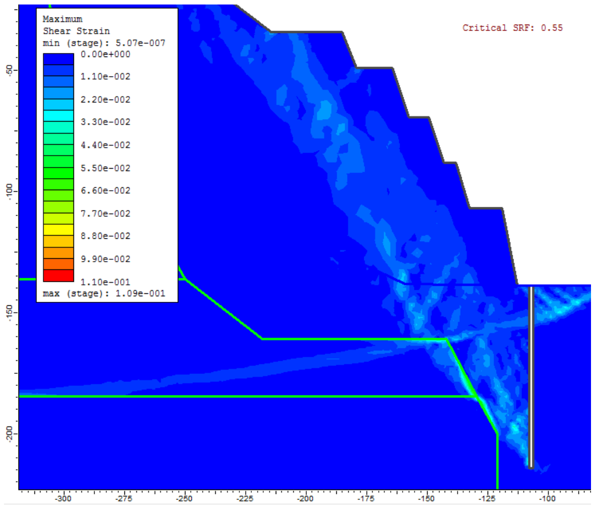


Figure 2: Maximum shear strain with SRF =1 in critical slope of section 1-1', trench moved away from the toe of the slope with 10 m. SRF calculated by RS² is 0.55.

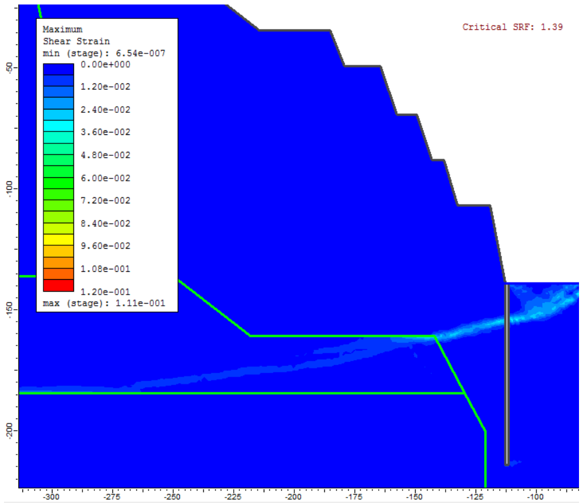


Figure 3: Maximum shear strain with SRF =1 in critical slope of section 1-1', trench filled with heavy density fluid (specific weight = 20 kN/m³). SRF calculated by RS² is 1.39.

The effects of applying a liner (Figure 1) and moving away (Figure 2) are very limited. The calculated SRFs of the macro scale slope stability with stability measures are the same as the SRF without (SRF = 0.45). Application of a heavier density fluid, like a bentonite slurry, however, has significant effect, increasing the SRF to 1.39.

Pit cross sections

A total of four cross sections through the ultimate pit were determined to assess the macro scale stability of the Victor open pit. The cross sections were chosen such that they are perpendicular to the trenching direction. Every cross section is specifically simulated for a single trench cutting target. Trenches were created in the model at the base of the critical slope and modelled with a ponded water load representing the hydraulic stress exerted by the water inside the trench on the trench walls. Simulation results have been specifically shown for this critical slope.

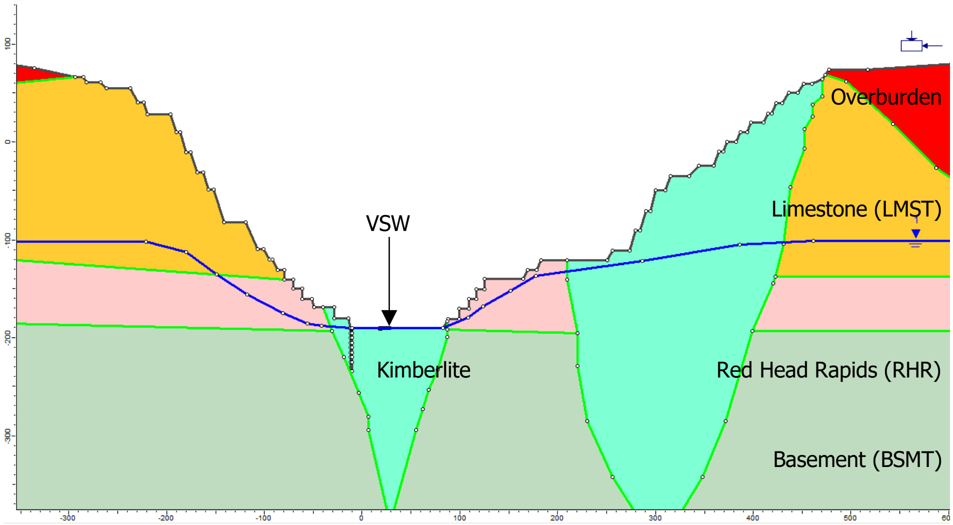


Figure 4: Geological cross section 2-2′ including rock units and phreatic surface indicated by the blue line

The calculated SRF for cross section 2-2′ is greater than 1, indicating stable pit conditions. This is partly due to the relative shallow trench that is cut at the toe of the slope. As indicated by Figure 7.11b slope failure in section 1-1′ occurred after 60 m of cutting.

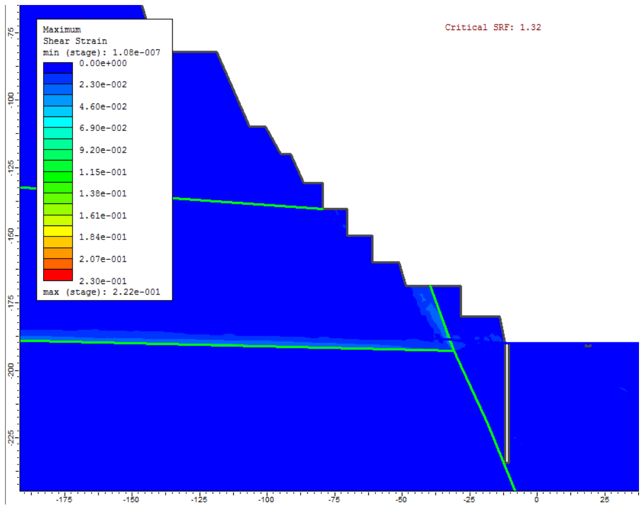


Figure 5: Maximum shear strain with SRF =1 in critical slope of section 2-2′, SRF calculated by RS² is 1.32.

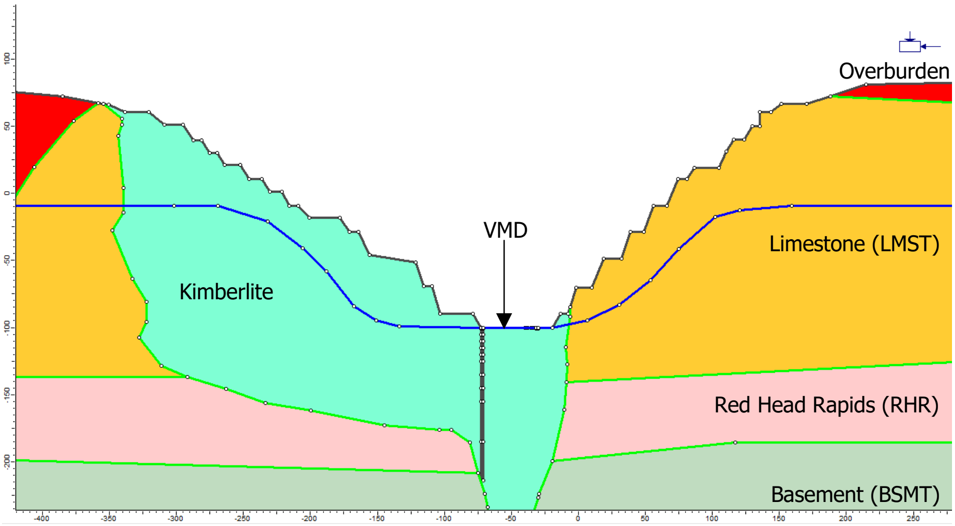


Figure 6: Geological cross section 3-3' including rock units and phreatic surface indicated by the blue line

Cross section 3-3' was created through the VMD target zone. This target zone has the greatest vertical continuity and is therefore be cut with the deepest trenches. As demonstrated by an SRF of 0.53, failure occurs in the slopes. Based on the analysis of the maximum shear strain in Figure 7 two failure planes occur through the rock mass inside the pit slope. One through the kimberlite rock mass, the other through the Red Head Rapids (RHR) formation. The RHR formation exhibits weaker rock than the kimberlite in the slope.

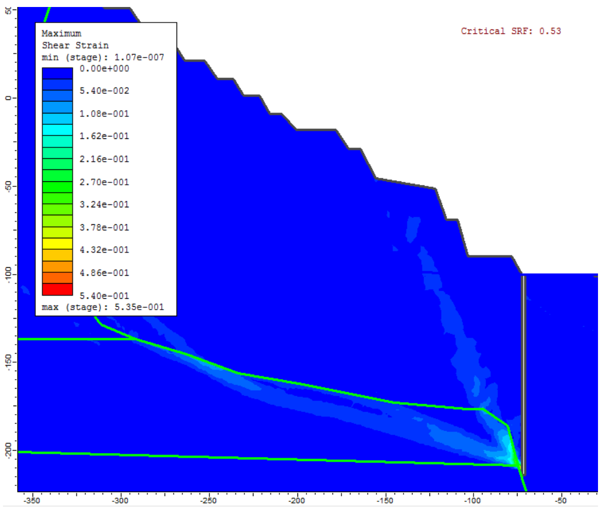


Figure 7: Maximum shear strain with SRF =1 in critical slope of section 3-3', SRF calculated by RS² is 0.53.

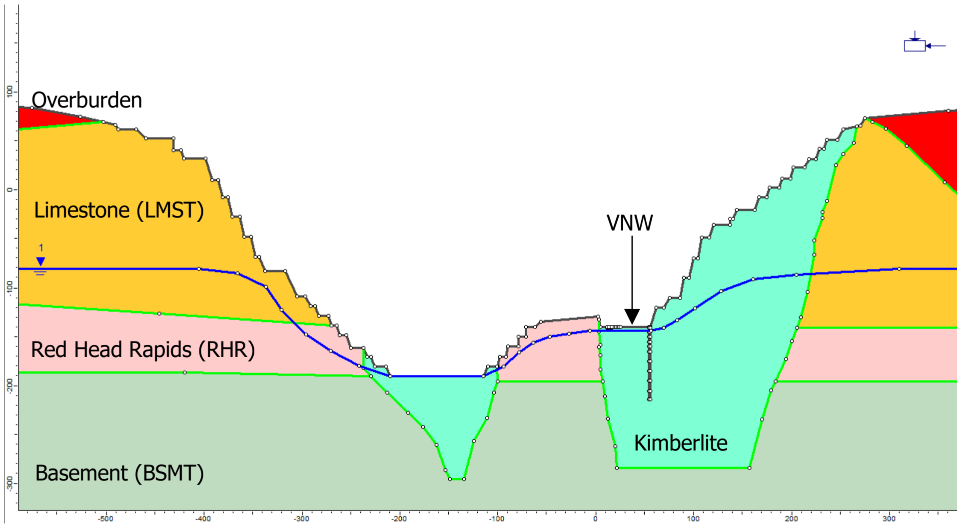


Figure 8: Geological cross section 4-4' including rock units and phreatic surface indicated by the blue line

The section through the VNW target has the lowest calculated SRF of the targets. Figure 9 shows the failure planes in the northeast slope. Part of the failures appear to continue towards the middle of the pit and the VSW target.

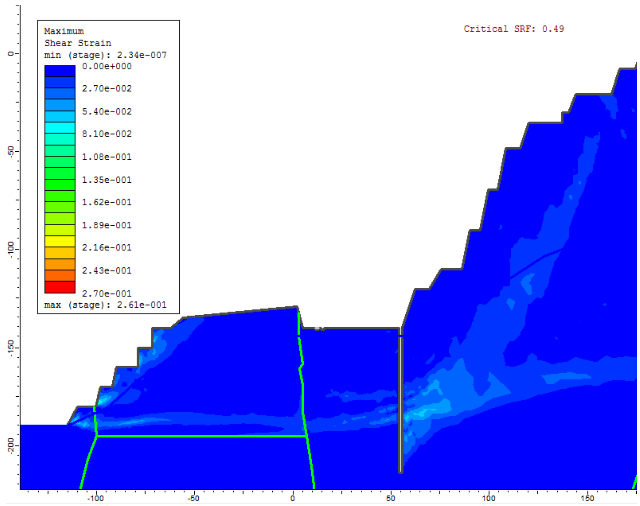
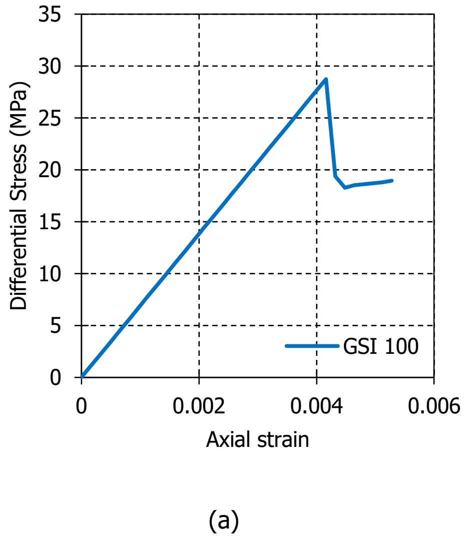


Figure 9: Maximum shear strain with SRF =1 in critical slope of section 4-4', SRF calculated by RS² is 0.49.

SRF calculations of the macro scale stability for trenches at the toe of the slopes are in comparable range and exhibit similar results. Section 2-2' is an exception, showing a considerable higher SRF. The difference is particularly caused by the difference in trench depth. Risk of slope failure during trench cutting is high for all target locations at the Victor mine.

FE model validation

As discussed in the Discussion (Chapter 11) the FE model has been validated by simulating a UCS measurement and comparing the SRF of the undisturbed ultimate pit with earlier performed stability assessments. The simulation of the UCS measurement was performed in order to validate the rock behaviour in the macro scale FE analysis. A rectangular model of dimensions 0.25 by 1 m was modelled in RS^2. By forcing the model to perform an axisymmetric analysis the simulation creates a cylinder of diameter 0.5 m (or with base to height ratio of 1:2). The UCS test simulation was modelled for both an intact rock sample and a sample adjusted according the geologic strength index used in the generalized Hoek-Brown failure criterion. Figure 10 demonstrates the stress-strain curves of the two simulations. The curves both clearly depict the elastic behaviour of the rock before the load to failure and plastic deformation thereafter.



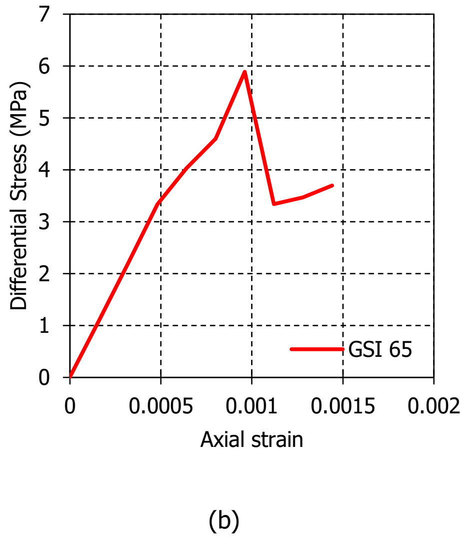


Figure 10: Stress-strain curve of kimberlite rock based on simulated UCS test. (a) Intact rock sample (GSI =100), (b) Strength reduced sample according in-situ estimations (GSI =65), curves plotted for upper right corner of sample

The stress-strain curves in Figure 10 exhibit good representation of an elastic-brittle failure mechanism (a) to slight strain softening in (b). The influence of the GSI reduction is significant. For the intact rock sample (GSI = 100) the peak strength does not reach the intact compressive strength of the rock. It is considered that this is caused by the simulation method and the influence of size of the model. The GSI reduced sample behaves as anticipated by the generalized Hoek-Brown failure criterion. A reduced GSI results in a lowered effective strength of the sample anticipated to be approximately 5.7 MPa. Kimberlite with reduced GSI is applied in the macro scale model, suggesting the rock behaves as anticipated within the models. Rock properties (peak and residual) have been assumed similarly for the other rock units in the macro scale model.

D

Financial Model

The excel file containing the financial model has been submitted with this thesis as a digital file. The excel file consists of the following tab sheets:

- Project overview, containing a model summary and project planner;
- Base case – Cash Flow, providing the complete cash flow calculation per month;
- Cumulative Cash Flow (graph);
- Cumulative Cost (graph);
- Cutting, summarizing the cutting performances of the BC 50 in t/h and (vertical) m/h as well as the cumulative cash flow variations for different cutting speeds;
- Sensitivity Analysis, containing the sensitivity analysis for all parameters mentioned in the report;
- Production, providing an overview of the extraction rates per scenario as well as the associated NPV with the scenario;
- Costs, summarizing an extensive list of project costs; and
- Parameters, displaying additional parameters that have been used to calculate the cash flows and production performances.