Techno-economic optimisation of mining cut for Zimbabwe Platinum Mines' steeply dipping tabular orebody

Maida Wailesi

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

Johannesburg, 2019

DECLARATION

I declare that this research report is the original work of my research. The report has been entirely written by me and has not been submitted for any previous degree or professional qualification. References have been provided for literature and resources used in this report.

Signature ______________________

29th **day of** July **year** 2019

ABSTRACT

Zimbabwe Platinum Mines (Zimplats) extracts Platinum Group Metals (PGMs) in Zimbabwe along the Great Dyke. The orebody being mined at Zimplats is tabular, synclinal in nature and plunges gently at approximately 1.5º towards the north. Layers of igneous rocks within the deposit dip at 9° to 20° near the limbs and flatten near the centre to form a flat-lying floor. Mineral Resources at the mine are classified into 'flats' (Mineral Resources dipping shallower than 9º) and 'steeps' (Mineral Resources dipping at an angle > 9º). Current mining operations are limited to the 'flats' portion of the deposit but as mining progresses towards the north, the ratio of 'steeps' to 'flats' increases. The current mining layout design was based on the 'flats' with a mining cut of 2.5m. In order to exploit the steep dipping orebody limbs, modification of the current mining layout was done in 2016 and trials were conducted on the new design. Results from the trials showed that the design resulted in a lower actual head grade than planned which was attributed to a sub-optimal mining cut.

This research study was done to determine an optimum mining cut at Zimplats mine for the 'steeps' portion of the deposit, focussing on the Mineral Resources dipping between 9° to 14° to create optimum value from extracting the steep Mineral Resources. The study was carried out at P4 mine and two blocks out of the available six blocks were randomly selected for the purpose of this study. In the optimisation process, six mining cuts were selected, 2.00m, 2.10m, 2.20m, 2.30m, 2.40m and 2.50m. Mine designs were generated for each mining cut in Vulcan 3D software and Mineral Reserve calculations were done for each option. Mining schedules were generated through the application of the Vulcan Gantt Scheduler. A techno-economic evaluation was done on the six mining cuts focusing on, Mineral Reserve tonnage, grade, extraction ratio, cost and revenue generated. A discounted cash flow (DCF) analysis was done which incorporated all the evaluation parameters to determine the optimum mining cut which yields the highest Net Present Value (NPV). A sensitivity analysis was done to determine the reliability of the chosen optimum mining cut.

It was found that the optimum mining cut varied depending with the optimisation criteria. However, from the DCF analysis, the 2.10m mining cut option produced the highest NPV of US$95 million among the other possible mining cuts. Therefore, the

research study concludes that the optimum mining cut for the ‘steeps’ Mineral Resources at Zimplats mines is 2.10m. It is being recommended that the company adopts 2.10m for the exploitation of its ‘steeps’ resources. It is also recommended that the company should design a mining method for the ‘steeps’ resources with dip angles greater than 14° for maximum Mineral Resource utilisation.

ACKNOWLEDGEMENTS

The completion of this thesis would not have been possible without the unwavering support of Mr. Tinashe Tholana whose guidance and encouragement inspired me to produce this research report to the best of my ability. My gratitude is extended to my peers at Zimplats for their advice and constant motivation which drove me to pursue a Master of Science degree in Mining Engineering and inspires me to go beyond. I would also want to thank my beautiful wife and the mother of my children, Sibongile Wailesi for her unending patience, stalwart positivity and fierce commitment to my success in this endeavor. Above all, all thanks and praise is to the Most High God for the mercy and wisdom granted in the pursuance and completion of this degree.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

LIST OF ACRONYMS

Au	Gold
BMSZ	Base Metal Sub-Zone
DCF	Discounted cash flow
Fe-Ni-Cu	Iron-nickel-copper
GRF	Grade reduction factor
Ir	Iridium
LHDs	Load haul and dump machines
LOM	Life of mine
LP	Low profile
MSZ	Main sulphide zone
NPV	Net present value
OEM	Original equipment manufacturer
OR	Operations research
Os	Osmium
Pd	Palladium
Pt	Platinum
Rh	Rhodium
Ru	Ruthenium
TMMs	Trackless mechanised machines
XLP	Extremely low profile
Zimplats	Zimbabwe Platinum Mines

1 INTRODUCTION

1.1 Chapter overview

This chapter provides the introduction of the research study. Background information on Zimbabwe Platinum Mines' (Zimplats) operations is also discussed. Further, background information on the type of orebody being mined, production rate and equipment currently in use at Zimplats is also provided. The mining method used at Zimplats is also described where the apparent angle concept is described. This chapter also describes the mining sequence at the mine, which involves development and stoping. The problem statement and research question are outlined in this chapter. The chapter also outlines the study aim and objectives as well as the research scope and the research significance. The chapter ends with a summary of the chapter and introduces the next chapter.

1.2 Background information on the mine

Zimplats is a platinum mining company with operations in Mhondoro Ngezi in Zimbabwe, 150km south east of the capital city, Harare. The company has mineral rights along Ngezi tenements in the Sebakwe sub-chamber that form part of the Great Dyke of Zimbabwe (Zimplats, 2012). The orebody being mined at Zimplats is tabular, synclinal in nature and plunges gently at approximately 1.5º towards the north. The mineralised zone known as the main sulphide zone (MSZ) is located between the mafic and ultra-mafic sequence in the Great Dyke of Zimbabwe (Zimplats, 2012). Figure 1.1 shows the general stratigraphy within the Ngezi mine area.

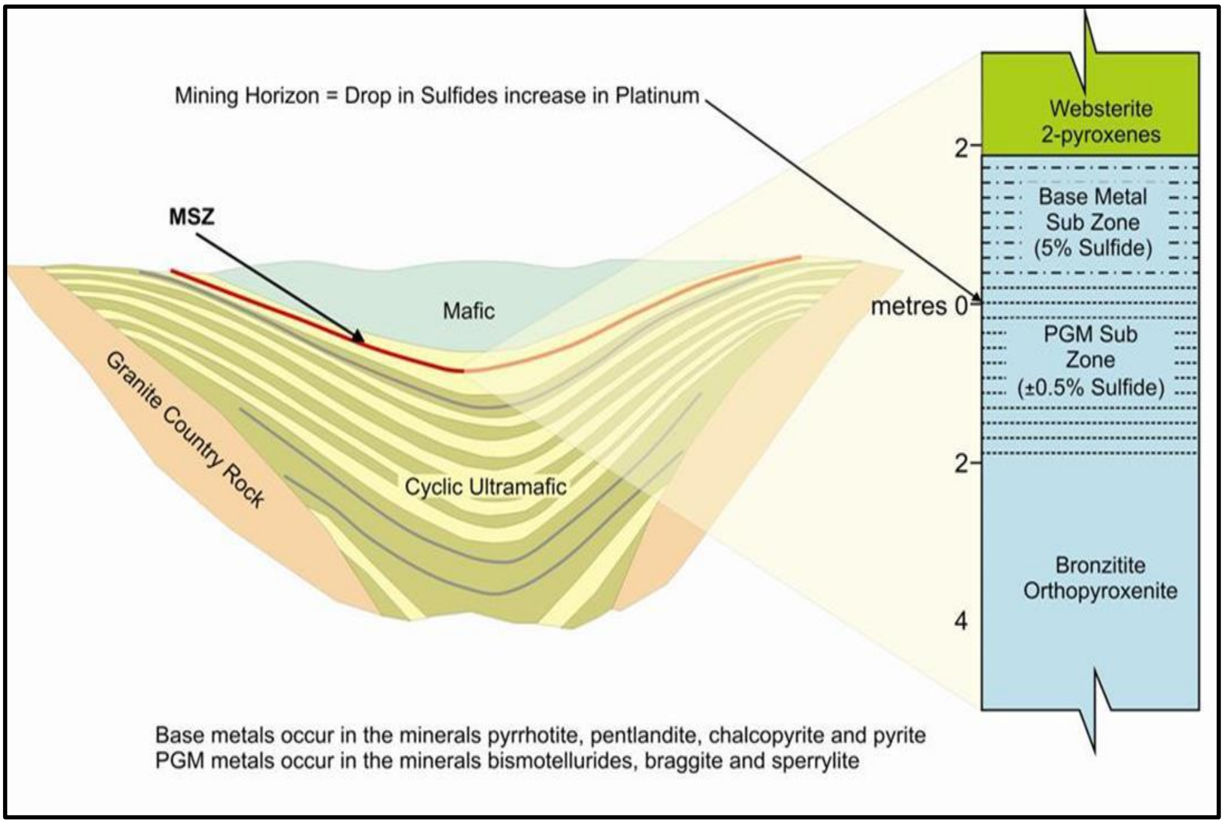


Figure 1.1: Generalised stratigraphy within the Ngezi Mine area

(Source: Zimplats, 2012)

According to Zimplats (2012), the MSZ host economically exploitable PGMs and related base metal mineralisation. The MSZ is a continuous layer with a thickness ranging between 2m and 10m forming an elongate basin. Layers of igneous rocks within the basin dips at 5° to 20° near the limbs and flatten near the centre to form a flat-lying floor (Zimplats, 2012). The MSZ can be visually identified by the presence of iron-nickel-copper (Fe-Ni-Cu) sulphides disseminated mainly on the top part of the MSZ, termed the Base Metal Sub-Zone (BMSZ). From the base of the BMSZ is a 1 m to 5 m thick zone of elevated PGM values commonly referred to as the PGM Sub-Zone (Zimplats, 2012). According to Zimplats (2012), the base of the MSZ can be located visually in a drill core. The PGM mineralisation is spatially linked to the peak sulphide zone in a consistent manner. The change in sulphide mineral content is used as a marker for Mineral Resource evaluation and mining grade control (Zimplats, 2012). The change in sulphide mineral abundance can be identified in drill cores and on underground mining faces where it is used to guide sampling and mining. The PGM mineralisation includes major concentrations of platinum (Pt), palladium (Pd), rhodium

(Rh) and gold (Au), and minor concentrations of iridium (Ir), ruthenium (Ru) and osmium (Os) (Zimplats, 2012). Figure 1.2 shows the location of the mine in relation to the Great Dyke and other platinum mining operations in Zimbabwe.

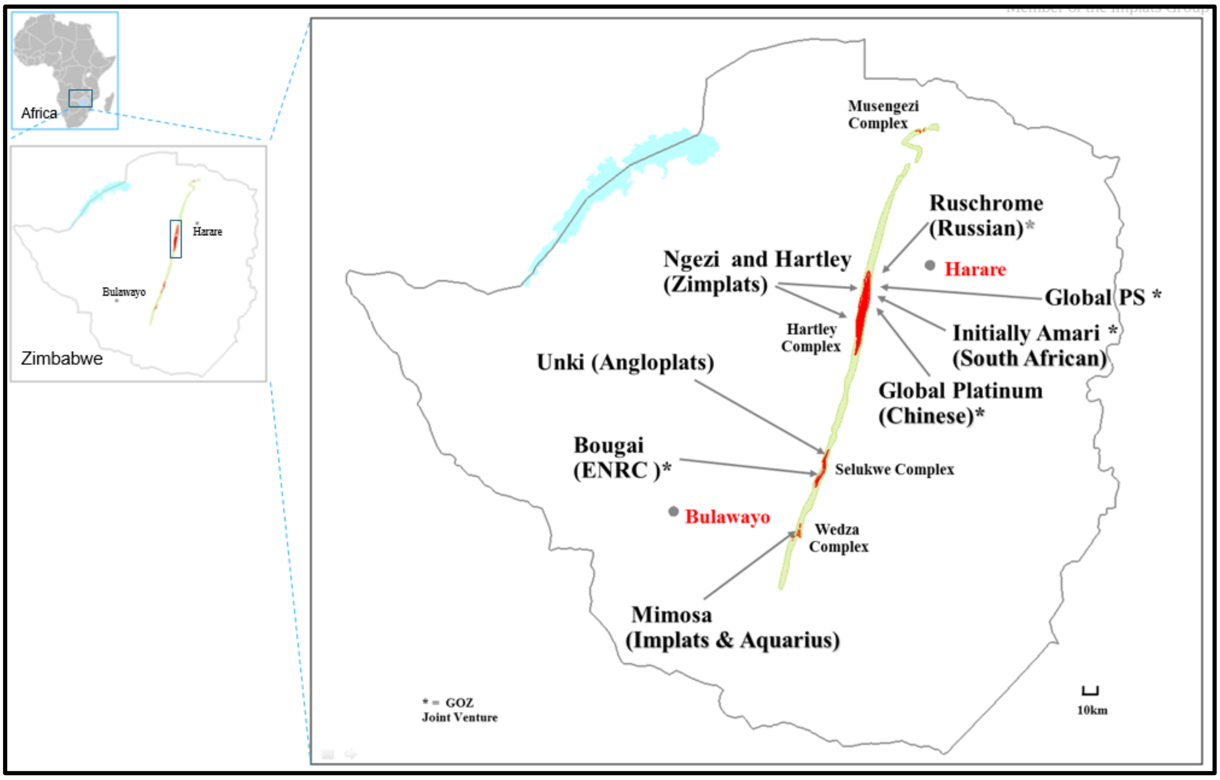


Figure 1.2: Zimplats location

(Source: Zimplats, 2012)

Zimplats is currently operating five underground mines producing 6.2 million tonnes of ore per annum. Mechanised room and pillar mining method is used at all the five mines. Trackless mechanised machines (TMMs) used in the underground operations are low profile (LP) machines with a minimum operating height of 2m. The TMMs comprise of 10t load haul and dump machines (LHDs), 30t dump trucks, face rigs and support rigs. According to the original equipment manufacturer (OEM), the equipment used at Zimplats optimally operates on dips shallower than 9^{o}. The orebody dip has informed the broad classification of the Mineral Resources by the company into ‘flats’ (Mineral Resources dipping from 0^{0} - 9^{o}) and ‘steeps’ (Mineral Resources dipping at an angle > 9^{o}) as illustrated in Figures 1.3 and 1.4. In this research study, the flat and steep Mineral Resources are also referred to as ‘flats’ and ‘steeps’, respectively.

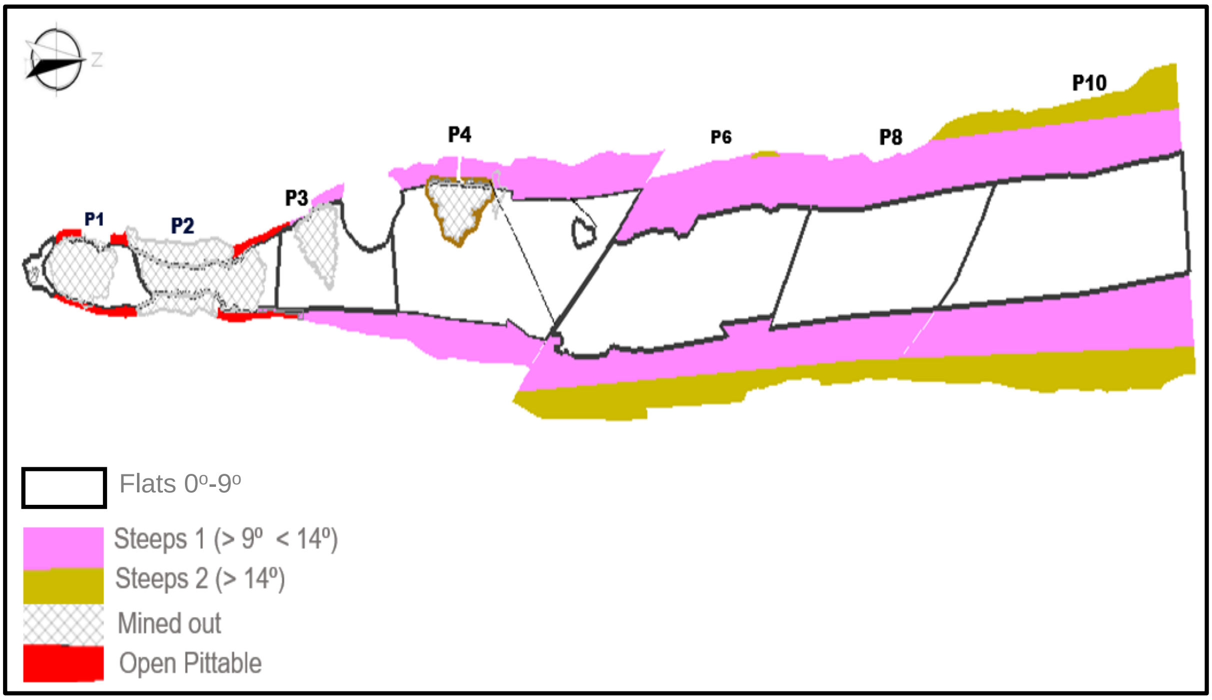


Figure 1.3: Zimplats Mineral Resources categorisation

(Diagram produced by the author in Vulcan)

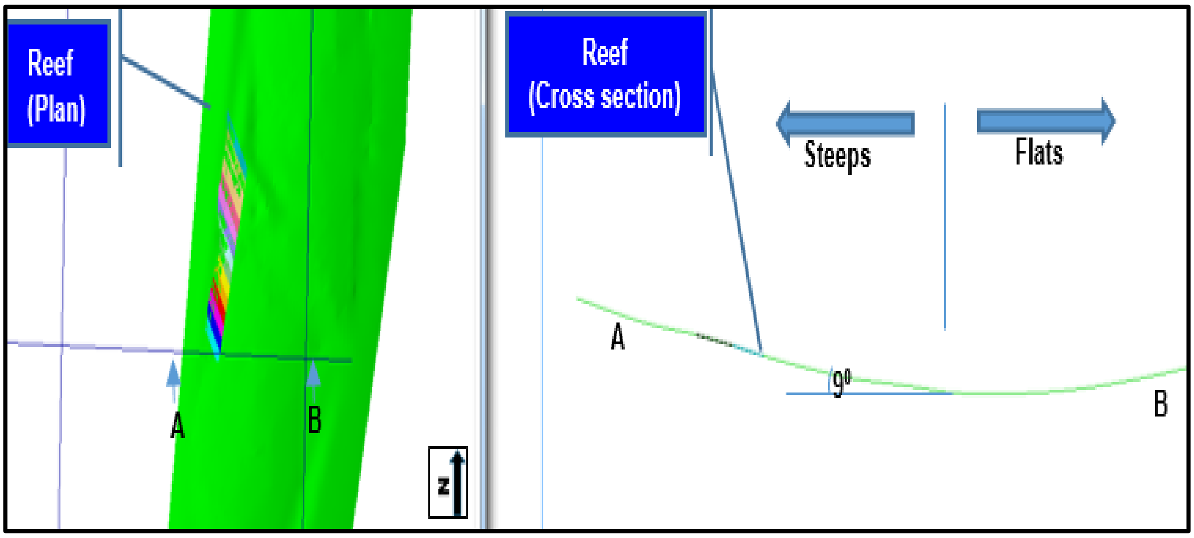


Figure 1.4: Reef on plan and cross section in the 'steeps' section
(Source: Zimplats, 2012)

Mining of the deposit is done from the south to the north with P1, P2, P3 and P4 being the mines currently in production. Current mining operations are limited to the 'flats' portion of the deposit. However, the ratio of 'steeps' to 'flats' is increasing as mining operations progress further north. Figure 1.5 shows the proportion of flat and steep Mineral Resources from south to north.

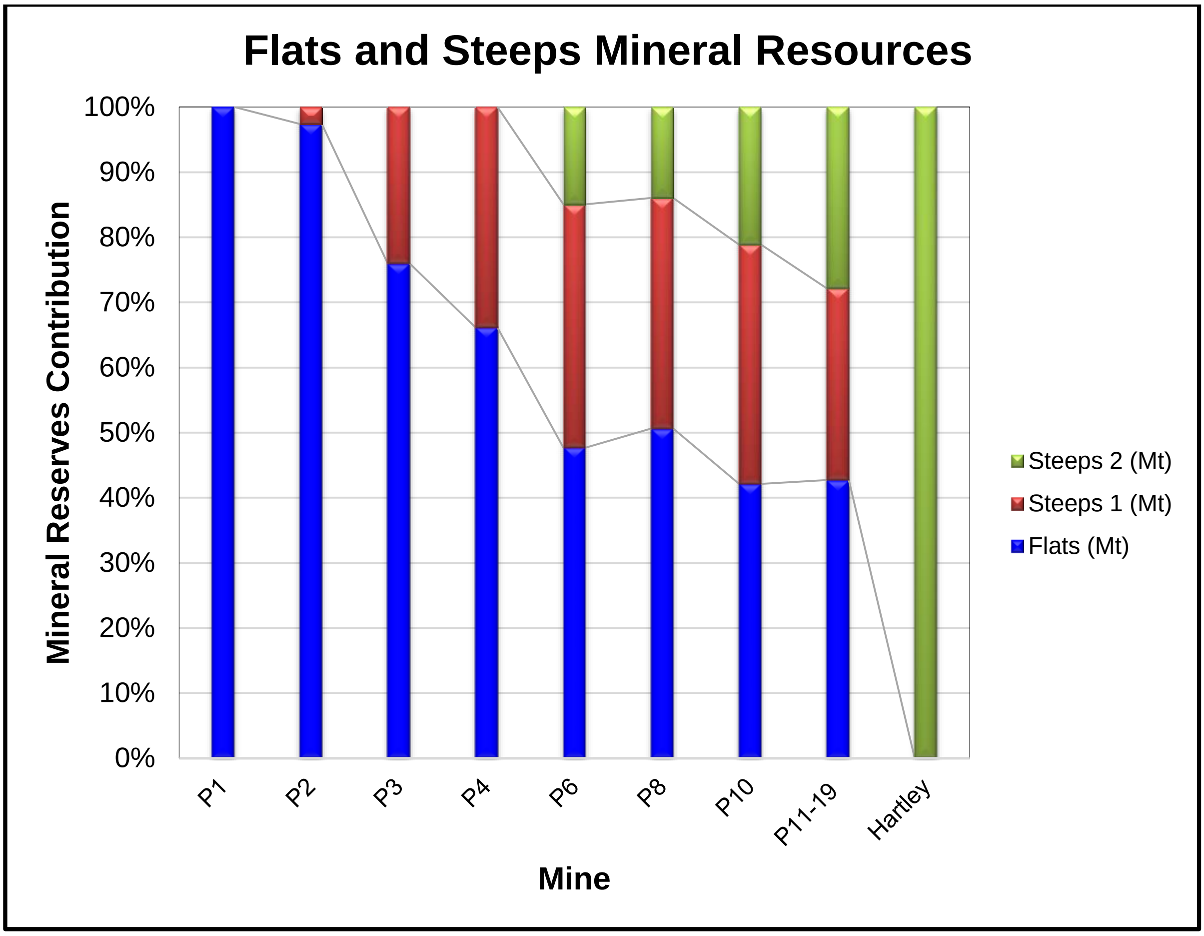


Figure 1.5: 'Steeps' contribution from south to north

(Source: Zimplats, 2017)

The figure shows that the ratio of 'flats' is decreasing while the ratio of 'steeps' is increasing from P1 (south) to Hartely Mine (north). In total, the 'steeps' constitute approximately 53% of the company's total Mineral Resource base. Table 1.1 shows that current mines (P1, P2, P3 and P4) have more of 'flats' but future mines will have an almost equal proportion of 'flats' and 'steeps'.

Table 1.1: Mineral Resources grouped by dip category

(Source: Zimplats, 2017)

Portal	Flats (Tonnes)	Steeps by Dip (Tonnes)		
	0-9 º	9 -14 º	>14 º	Total Steeps
P1	9 109 796			
P2	5 851 056	156 603		156 603
P3	32 491 522	10 287 127		10 287 127
P4	93 235 733	47 741 408		47 741 408
P6	89 489 512	70 110 933	28 014 936	98 125 869
P8	67 979 776	47 565 576	18 737 150	66 302 726
P10	62 617 536	54 662 728	31 420 308	86 083 036
P 11-20	466 041 421	320 958 000	304 000 579	624 958 579
Hartley			217 662 698	217 662 698
Total	**826 816 352**	**551 482 374**	**599 835 672**	**1 151 318 046**

Since the current operating mines have more 'flats' resources, the current mining layout design was based on the 'flats' with a mining cut of 2.5m. In order to exploit the steep dipping orebody limbs that started being encountered from P4, modification of the current mining layout was done in 2016 and the new layout trials were done. The 'steeps' Mineral Resources in the area marked for the trials dips from 9° to 18°. These dip angles presented a challenge to the existing TMMs which cannot operate efficiently and safely at such angles. An angle of 45º to reef strike was then considered in the new layout with an apparent angle of 6º - 10º to accommodate the existing TMMs in the 'steeps' areas. This apparent angle concept is illustrated in Figure 1.6.

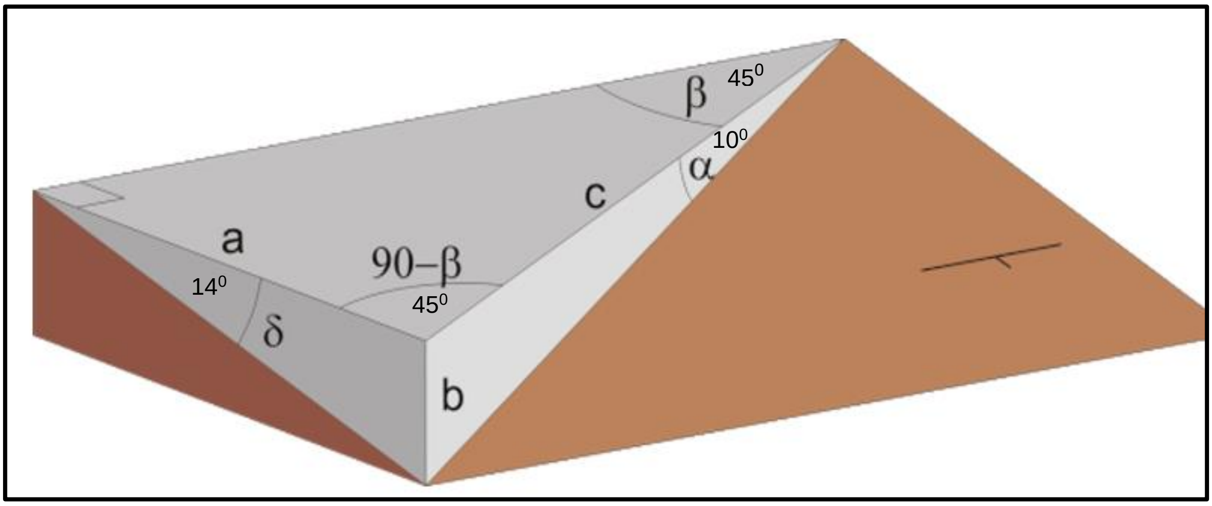


Figure 1.6: Apparent dip angle concept

(Source: Jorge, 2012)

δ = real dip

α = apparent dip

β = angle between strike direction of the plane and the apparent direction of dip direction

However, the new layout still uses the current mining cut of 2.5m. Mining is primarily in the north east direction for ease of LHD operation. Figure 1.7 shows the modified layout undergoing trials at Zimplat's P4 Mine.

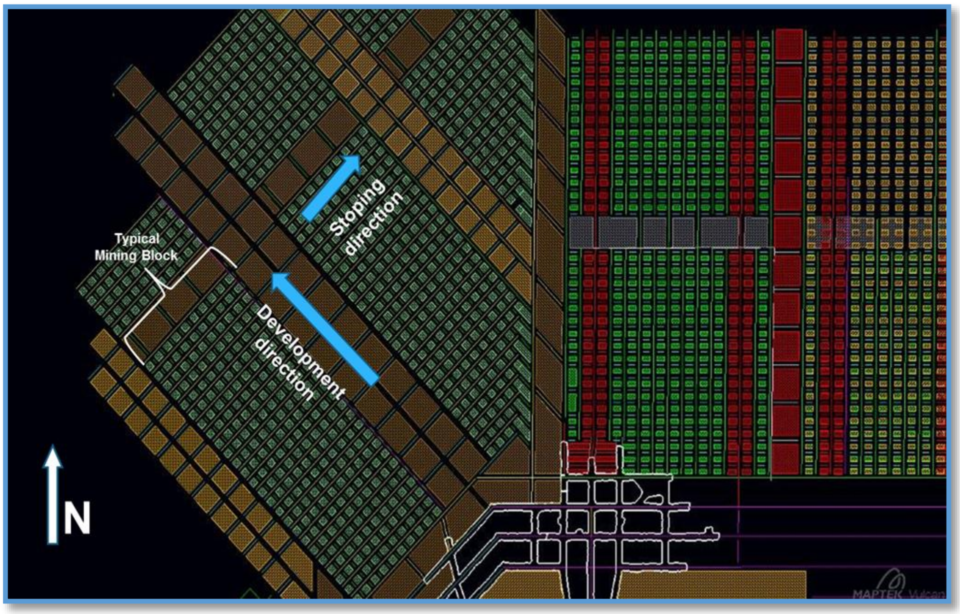


Figure 1.7: Modified mining layout for the 'steeps' section
(Source: Zimplats, 2017)

1.3 Problem statement and research question

During trial of the new layout, it was found that the layout consistently delivered results below the planned head grade of 3.24g/t. The new design has a mining face profile that results in lower head grade than planned because of mining at an angle less than the true dip with the new layout still using a mining cut of 2.50m. Figure 1.8 shows the head grade performance trend in the ‘steeps’ section resulting from the new layout design.

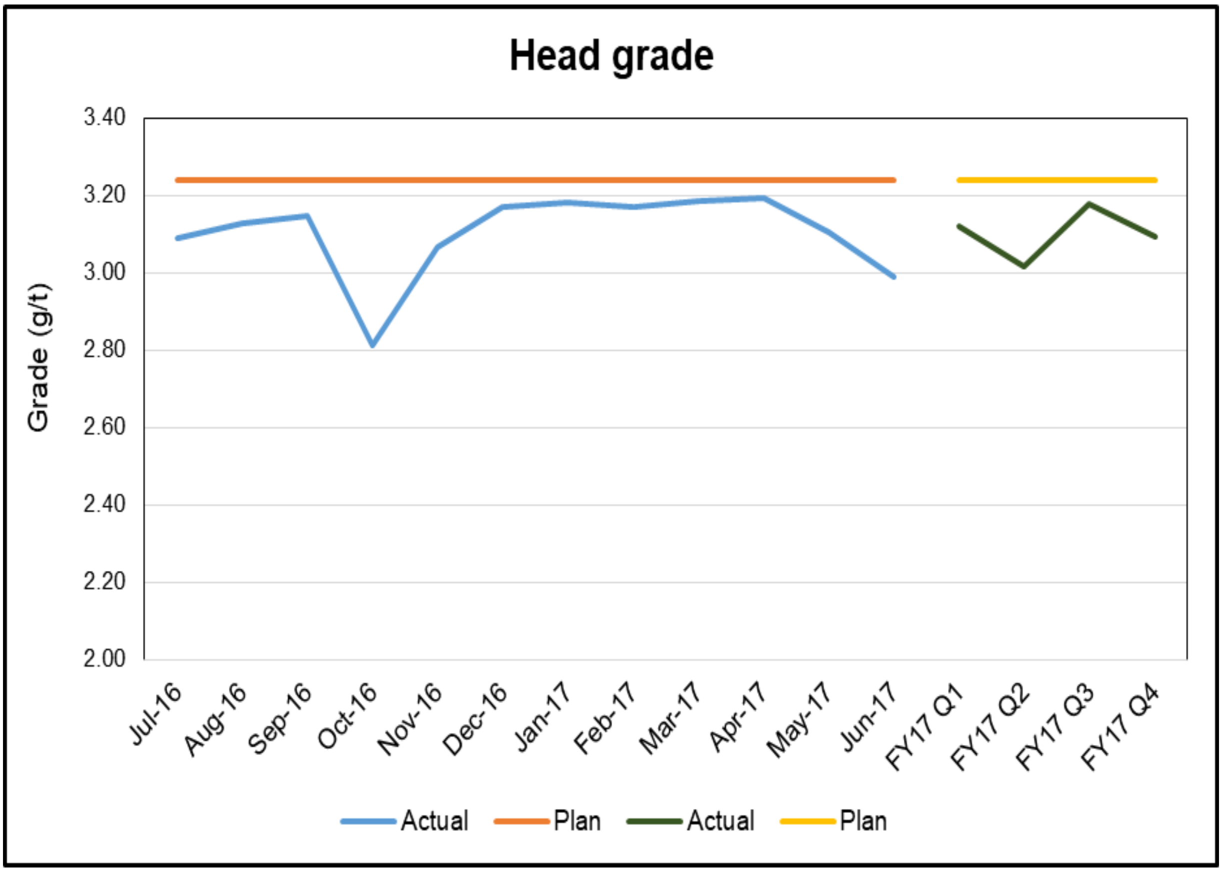


Figure 1.8: Head grade trend for the new mining layout
(Source: Zimplats, 2017)

The trend in Figure 1.8 is not optimum for the company if it persists, because of the negative effects of dilution on economic value. The head grade achieved in October 2016 was worsened by a part of the orebody with low in-situ grade that was mined during that month. Dilution lowers metal recoveries, increases operating costs leading to lower profits and net present value (NPV). To optimise value from mining the ‘steeps’ a new optimum cut must be determined and a new layout must be designed with a new mining cut. Due to capital constraints the new layout must be compatible with the current TMMs. This raises the research question:

> *What is the optimum mining cut that can be applied to the current ‘steeps’ design to optimise NPV whilst still meeting the 90 000t per month production at a minimum head grade of 3.24g/t?*

1.4 Research aim and objectives

The aim of the research is to determine the optimum mining cut for ‘steeps’ mining at Zimplats. The research objectives are to:

- Conduct a literature review on underground mine optimisation in general and mining cut optimisation in particular;
- Understand Zimplats orebody orientation;
- Understand the grade profiles of Zimplats mines orebody; and
- Conduct a techno-economic evaluation to determine an optimal mining cut.

1.5 Research scope

The research focuses on ‘steeps’ mining at P4 on the western side boundary of the existing Mineral Reserves. The western boundary of the ‘steeps’ block is defined by the 40m contour line, which is 400m from the eastern ‘steeps’ boundary as shown in Figure 1.9. The focus of the study is on ‘steeps’ dipping at 9^0 to 14^0 only.

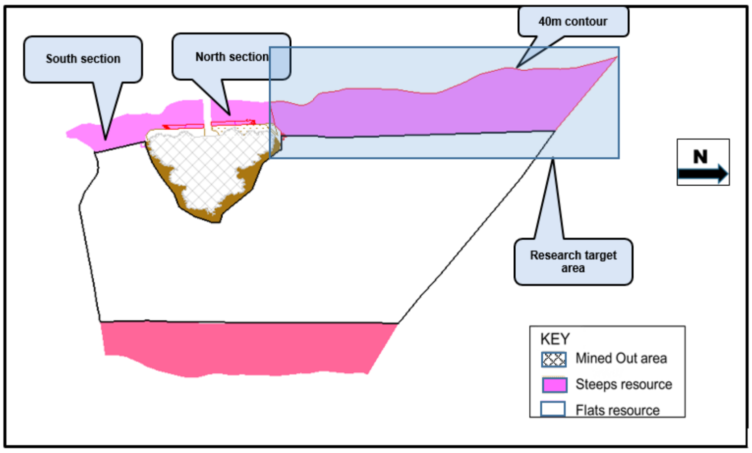


Figure 1.9: Research scope

The north and south declines divide the Mineral Resources into the north and south 'steeps' Mineral Resources, respectively, and the research focuses on the northern part of the mine. Even though the research focuses on P4, the results however will be applied on all the other mines with 'steeps' Mineral Resources at Zimplats.

1.6 Research significance

Figure 1.8 indicated a sub-optimal grade performance, and there is a need to determine an optimal mining cut in the 'steeps' section to optimise head grade. The current mining cut for the 'steeps' design is set at 2.50m and determining an optimal mining cut will avail the 'steeps' Mineral Resources for economic mining. Successful optimisation of the mining cut design means the Mineral Resources on the western and eastern flanks of the Zimplats orebody that are characterised by steep dips can now be economically extracted. The success of the trial mining project of steep dipping Mineral Resources at P4 will avail the Mineral Resources for mining and the existing ore-handling infrastructure at P4 in that area can be used to exploit the 'steeps' while the mine ramps up to full production. If the 'steeps' resources are included in the Mineral Reserve boundary of a mine, it quickens production ramp up to full production. This is due to early access to the orebody compared to the current designs for P1, P2, P3 and P4, whose main decline development intersected the orebody at the 9^{o} horizon for the mining of the flat portions of the orebody. The life of mine (LOM) can be prolonged by economically exploiting 'steeps' Mineral Resources without substantial capital requirements because the required mining infrastructure already exist. 'Steeps' resources constitute a significant portion of the Zimplats Mineral Resources and successful determination of the optimum mining cut results in maximum Mineral Resource recovery and prolonged LOM which has immense benefits to Zimplats and Zimbabwe's economy.

1.7 Chapter summary and report outline

The research report is structured in five chapters, Chapter 1 provided introduction of the study including background information on Zimplats. Zimplats operates five underground mines along the Zimbabwe Great Dyke. The orebody being mined at Zimplats is tabular, synclinal in nature, plunging gently towards the north and having steep dipping limbs. Equipment used at the mine include 10t LHDs, 30t dump trucks, low profile face and support rigs. Zimplats Mineral Resources are classified according to dip, into ‘steeps’ which have dip angles exceeding 9^o and ‘flats’ with dip angles less than 9^o. ‘Steeps’ contribution to the total Mineral Resources increases from south to north with ratio of ‘steeps’ to ‘flats’ increasing significantly for the mines north of P4. The current layout design is based on the flat Mineral Resources which is not suitable for the steep Mineral Resources. Therefore, a new layout was designed for the steep Mineral Resources. The new layout resulted in a change to the principal mining angle to have an apparent orebody dip ranging between 6^o and 10^o. The drawback of the new layout is that it resulted in below planned head grade performance. Therefore, it is important that an optimum cut for the new layout be determined to optimize NPV. The significance of the research include benefits such as an increase in LOM, improved capital utilisation and maximisation of economic value. The remainder of the report is briefly described below.

Chapter 2 reviews literature on mining optimisation in general and mining cut optimisation in particular. Chapter 3 describes the research methodology used in this project. Chapter 4 presents and analyses the results. The chapter presents the techno-economic evaluation results of the six mining cuts done and the results of the discounted cash flow (DCF) analysis done to validate the research findings. Chapter 5 concludes the study and recommends on the optimum cut for Zimplats ‘steeps’ Mineral Resources.

2 LITERATURE REVIEW

2.1 Chapter overview

This chapter reviews literature on underground mine design and the significance of optimisation in mine design and planning. Mine optimisation and optimisation of underground mines are also reviewed in the chapter. The chapter further reviews the main underground mine components that are amenable to optimisation that include stope layouts in general and mining cut in particular. The importance of techno-economic evaluation of mining cut is discussed in the chapter. Various Operations Research (OR) techniques are available to optimise mining cut and these are reviewed in the chapter. Mining optimisation criteria is reviewed and a chapter summary concludes this chapter.

2.2 Underground mine design

Mine design is defined as a system of mining elements and practices which include aspects such as mining methods applied, means of accessing the orebody, services required such as ventilation, power, water, material and personnel handling and other necessary requirements that allows mine planning to begin (SAMREC, 2016). There are broadly two methods of mining, underground mining and surface mining. In surface mining, the excavations are made on surface and in underground mining, the excavations are made sub-surface and they can be backfilled or left open depending on the particular underground mining method chosen. This study focuses on underground mine design because most of the Great Dyke is amenable to underground mining. An underground mine is basically a collection of ramps and drives connecting various points of access at each required level of the orebody to a surface portal (Brazil *et al*., 2000).

The mine design and planning process has evolved with time. Traditionally, the process relied mostly on the intelligence of the mine designer or planner for its decision making. However, the advent of technology now enables a fully integrated mine design and planning process which supplements the intelligence of the designer with expert knowledge and sophisticated data analysis, modelling, optimisation and simulation tools (Morin, 2001). Morin (2001) defined a traditional mine design and planning process as activities restricted to defining the methods for accessing and then extracting Mineral Reserves. He stated that based on experience or current practice, the engineer would determine the best plan for extracting the ore while the planner would determine and schedule the required resources to implement the engineer's plan. In the modern day mine design and planning process there is improved use of mine design and planning software which makes it easier to develop accurate mine designs and plans. Increased computerisation of the traditional process can only help the designer or planner. However, if greater design or planning efficiency and productivity are the goals, new components need to be added to obtain the full benefits of computerisation (Morin, 2001). Mineral Resources are non-renewable and therefore optimum value should be derived from their extraction. This optimum value is derived through optimisation of the mine design and planning process. Mine design and planning optimisation enable mine planners to discover areas of improvement which offer added value to the mining industry (Smith, 2015). Mine design and planning requires learning proved design and planning optimisation methods that provide a professional competitive advantage in today's competitive environment utilising industry standard software for modelling and optimisation (Smith, 2015). The following section reviews mine optimisation in general and opportunities for optimisation in the mine design of underground mines.

2.3 Optimisation

According to Burke (2017), optimisation can be defined as maximising or minimising an objective function versus a set of particular constraints. The function allows a comparison of the different alternatives for determining the best one. Burke (2017) highlighted that the objective function can be minimising cost or maximising profit. Optimisation contributes significantly in the success of mines, when new mines are being planned, and old ones upgraded, the option is there to optimise production in a way that would have been unthinkable (Abb, 2017). Mining presents a source of optimisation challenges that cover an extensive array of applications because it is a complex industry, which requires enormous input from various fields (Caccetta, 2007).

Caccetta (2007) highlighted that problems arising in these applications are difficult to solve because of the following reasons:

- Huge number of constraints and variables;
- Multi-criteria objectives which require rigorous analysis of multiple differing criteria in making decisions, for instance mining quality versus mining volume decision making;
- Non-linearity of decision variables; and
- The need to complete an exhaustive sensitivity analysis.

In optimising a mine’s grade performance the most crucial input is the planning and geology knowledge. As highlighted by George and Richard (n.d.), the amount of initial planning that is worthwhile depends upon the knowledge derived from experience and theory about the geological characteristics of the orebody, in particular its pattern of occurrence.

Mine optimisation has significant benefits to any production set up, be it an underground or a surface mine operation. It does not need to be an expensive exercise or take much time, but it has high potential of adding value to a mining operation (Chadwick, 2009). However, to realise full benefits from optimisation there is need to look at the entire operational process. Chadwick (2009) highlighted that focusing on a specific area of the value chain may result in a transference of costs or reduction in revenue in a different area of the value chain leading to achievement of sub-optimal value for the entire project. A balanced approach is therefore necessary where synergies amongst different stages of a mining project have to be understood before the optimisation process is implemented (Chadwick, 2009).

Optimisation has proved to work in many cases but in some cases this has to be followed by management development programs. In a Zhungeer coal mine case study in China, a production deadlock was overcame by optimisation resulting in a massive improvement in equipment utilisation and a rise in production volumes. According to RWE Power International (2008), Zhungeer Coal Mine in inner Mongolia China employed continuous mining technology to mine lignite, but the mine failed to achieve performance specified by the equipment manufacturers limiting the amount of coal that could be produced. RWE Power International (2008) engineers undertook a comprehensive technical evaluation of the structure and equipment, as well as mining and maintenance procedures, to identify areas for improvement. This analysis led to the design and development of on-the-job and classroom courses to develop managerial and skilled staff to successfully operate a mine resulting in production improvement. This shows that optimisation does not work in isolation to improve production performance.

As highlighted by Caccetta (2007), careful planning and management determines the financial viability of a contemporary mine which is affected by the declining average ore grade trends, increase in mining costs and environmental management considerations. It therefore means, better solutions can considerably impact profitability. As Caccetta (2007) highlighted, planning and management of a mining business is complicated by the larger scale of operations; legal and regulatory requirements; technological improvements which produce ever increasing data volume that requires thorough evaluation and analysis across all levels of an operation. Caccetta (2007) further mentioned that this ever increasing complexity has speeded the need for generation of sophisticated mathematical methods of improving efficiency, decision making and effective planning. Trends show that, this hurdle has been surmounted by application of considerably enhanced computational models and immense technology advancement which aims at coming up with optimum solutions to problems.

The effective operation and design of industrial systems needs mathematical analyses which considers all the critical features such as, quality and quantity of deliverables, resources and time constraints. System behavior is investigated by the subsequent mathematical models and identify the critical parameters affecting performance of the system (Caccetta, 2007). With all this knowledge, Caccetta (2007) concluded that, focus can be put on the problem of system performance optimisation. To optimise mining operations and improve on the bottom line, companies must utilise their assets in a way that exploits their potential while reducing capital and operating outflow (Ebbels, 2016). Successful application of optimisation techniques to enhance performance in many mining processes such as mine design and management have been applied (Caccetta, 2007). These techniques have been widely applied in surface mines compared to underground mines (Little *et al.,* 2012). Underground operations and its mining limits (boundaries) are less amenable to optimisation techniques (Shahriar *et al.,* 2007). This study focuses on underground mines because of the little study done in the area of underground mine optimisation.

2.4 Optimisation of underground mines

There are various techniques of optimising mining operations and optimisation is done at various stages of a business plan, as well as at different levels of confidence (Smit and Lane, 2010). There are a number of ways of mine optimisation that can be successfully applied to solve problems that arise in the management and planning of a mine (Caccetta, 2007). Morin (2001) stated that the main mine design and planning elements for underground mining are; orebody evaluation, mining method selection, layout and development design and production rate determination. The key mine design and planning components that offer opportunities for optimisation, are discussed in the next sub-sections.

2.4.1 Orebody evaluation

A key process in mine design and planning is the modelling of an orebody which is the foundation upon which subsequent mine design and planning decisions are based on (Pandey, 2018). An orebody according to Pandey (2018) has three distinct components:

- The physical geometry of the geological units that formed and host the orebody;
- The attribute characteristics in terms of assays and geo-mechanical properties of all materials to be mined; and
- The value model of the orebody in terms of economic mining.

A successful mineral exploration program depends on the correct estimation and valuation of the extent of an orebody (SGS, 2018). With the volume, grade and thickness of each block known, it is possible to transform this information and other factors such as price and cost into an economical aspect (Moharaj, 2014).

2.4.2 Mining method selection and optimisation

Anglo American (2018) defined mining method as the method of extracting ore from the surface or below the surface of the earth. It also defined an optimal mining method as one in which ore is mined safely, economically and with as little waste as possible. The selection of an optimal mining method takes into account the conditions of the ground that determine the dimensions of any excavation, the extent and orebody type, and the amount of waste in the scheduled stope (Alford *et al.*, n.d.). Pickering (2004) mentioned that it is the stoping method that is at the core of a mining system. Nelis *et al.* (2016) also emphasised that the main objective of underground mining is to maximise economic value from the extraction of an orebody and maintain high level of safety in the operation. To attain maximum economic value and high level of safety, economical and geo-mechanical factors have to be taken into account in the selection of an optimal mining method with the optimum stope cut or mining height. Figure 2.1 shows an approach which can be used to model and optimise a mining method to create optimum value for an organisation.

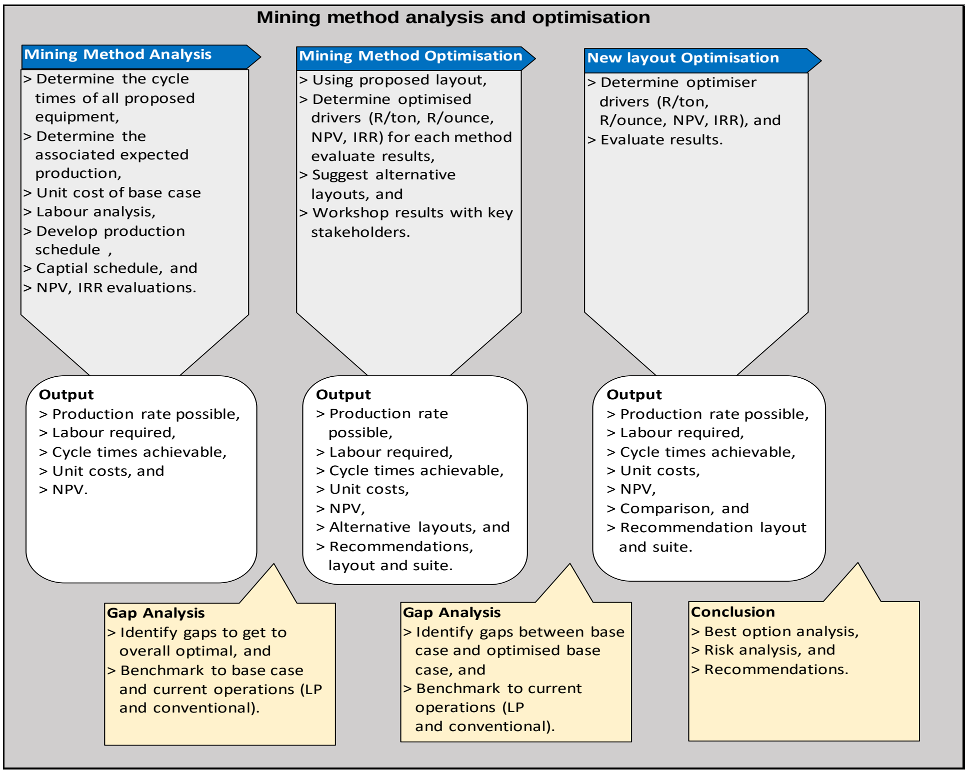


Figure 2.1: Mining method analysis and optimisation

(Source: Valicek *et al*., 2012)

Using this approach Valicek *et al.,* (2012) outlined mining parameters which can be changed and their effect measured:

- Face length or redundancy (Optimal face availability);
- Stope width - room and pillar LP and mechanised breast stoping extremely low profile (XLP);
- Equipment against mining method;
- Required development;
- Availability of resources; and
- Configuration or resource magnitude.

By varying and evaluating these parameters, the interactions that exist are understood and the mining method can be optimised to achieve optimal face availability and equipment configuration with the objective of upholding high safety standards and economic production (Valicek *et al.,* 2012).

2.4.3 Stope boundary optimisation

According to Nhleko *et al.* (2018), in underground mining, a stope refers to a mining area which comprises of several economic mining blocks. A stope boundary is the outline for blocks that can be mined economically by underground stoping methods (Alford *et al*., 2007). An optimal stope boundary guarantees optimum value over LOM for an underground mine, thus, stope boundary optimisation is one of the critical elements in underground mine design and planning optimisation process (Don *et al.*, 2015). To optimise the stope boundary problem, some algorithms have been developed. Most of the algorithms consider the dimension of the stope as one of the constraints and optimise designs in three dimensional space (Nhleko *et al.*, 2018). Selection of a good combination the stope boundary available directly affects the profitability of an operation (Topal and Sens, 2010).

2.4.4 Layout optimisation

Nhleko *et al.* (2018) defined an underground mine layout as the general arrangement of crosscuts, adits, stopes, raises, winzes, declines, crusher chambers, ore transportation system and any other infrastructure in an underground mine. At its simplest, underground mine design and planning involves developing an optimum layout for the underground workings that results in maximum ore production, minimum cost, and high level of safety (Hem, 2012). An optimal layout according to Hem (2012) must fulfill five fundamental objectives:

- Maximum production level over the LOM;
- Good ground support system and rock engineering criteria that results in uncompromised ground stability and minimise rock related risks;
- Efficient ventilation system which results in a good working environment at the minimum cost;
- Cost effective and efficient men and material handling system; and
- The layout must optimise economic value, that is, it must result in high production volume, low cost and must result the highest NPV possible.

Figure 2.2 shows a typical operating layout for an underground platinum mine.

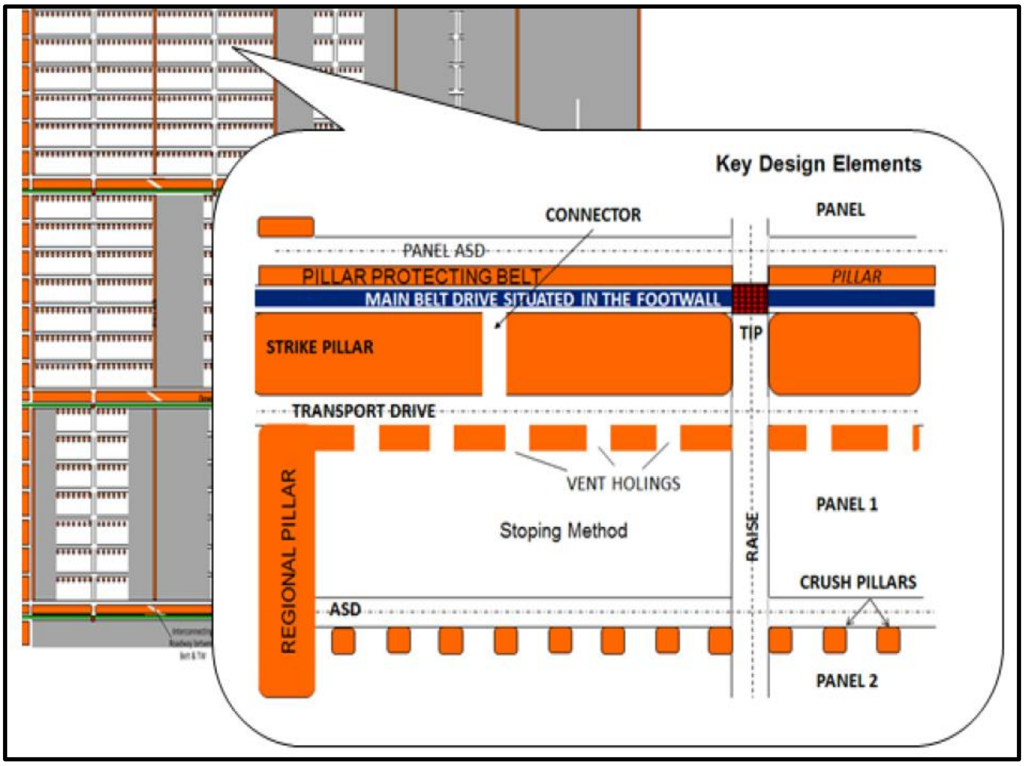


Figure 2.2: Detailed design XLP breast stoping mining section

(Source: Valicek *et al*., 2012)

Elements of a stope layout include stope width or mining cut, face length, stope profile and stope spacing in relation to pillars and other development structures like loading bays, roadways and crusher chambers. Based on the mining method concerned (board and pillar), stope width is referred to as mining width. This research study focuses on stope width optimisation as stope width has great impact on mined grade performance.

2.4.5 Production schedule optimisation

According to Ebbels (2016), schedule optimisation means utilising the available mine design, present equipment, personnel and other available resources in varying blends to generate the best schedule that produce the best NPV of a project. The problem of schedule optimisation for underground mining determines when a detailed mining activity should be executed in the future (Ebbels, 2016). Underground mine scheduling choices have been centered on production volume requirements, the mine planner's instincts on the best scheduling strategy and equipment efficiencies and productivities for a mine to optimise the project NPV (Ebbels, 2016). Ebbels (2016) also stated that in the recent years, advanced software programs have been developed to produce optimised underground production schedules centered on NPV.

2.5 Mining width optimisation

Mining width is the vertical distance between the hangingwall and footwall in an underground mining excavation synonymous to bench height in an open pit mine. Figure 2.3 shows the mining width in a room and pillar mine.

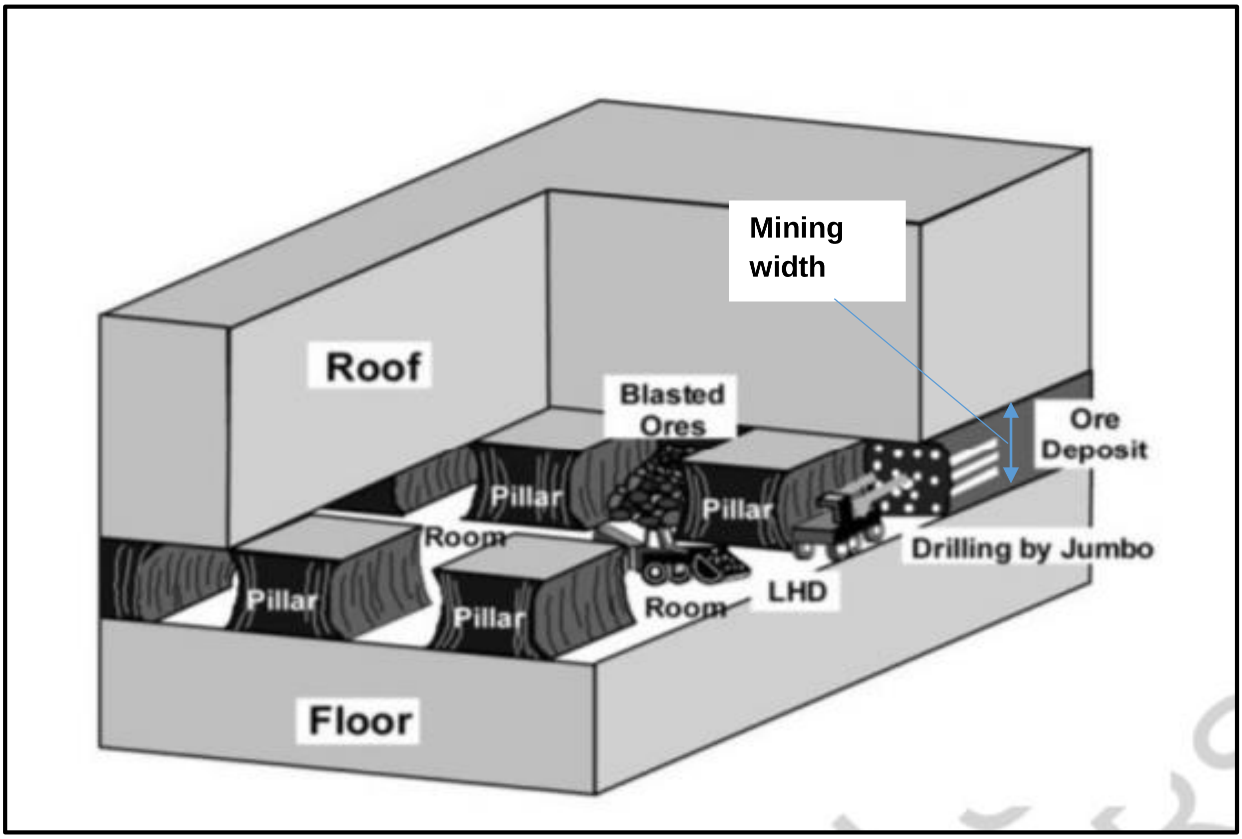


Figure 2.3: Mining width in a room and pillar mine

(Source: Okubo and Yamatomi, 2002)

An optimum mining width is defined as the width of the extracted ore deposit which maximise the NPV subject to a number of other economic and physical constraints (Pourrahimian *et al.,* 2018). Mining width can be largely controlled by producing a quality mine design and enforcing good stoping practices (Soyer, 2006). Zimplats (2016) highlighted that, analysis of the optimum mining width is based on achieving the maximum revenue per cut calculated from the value of recovered metal in concentrate.

Mining width has an impact on ore dilution in underground mines. One of the most crucial aspects affecting the economics of mining projects is mining dilution (Ebrahimi, 2013). Ebrahimi (2013) stated that during times when operations must operate with maximum efficiency, it is critical to determine, analyse and understand parameters, including dilution, influencing the project economics. Dilution cannot be totally avoided during mining, but it can be measured and managed (Ebrahimi, 2013). Ebrahimi (2013) mentioned that the starting point in optimising grade recovery is to quantify dilution and mitigate it by changing the mine design where possible.

According to Ercikdi *et al.* (2018) processes that contribute to dilution are orebody delineation, mine design and scheduling, stope development and practices relating to drilling and blasting. They highlighted that, dilution is defined and calculated with respect to a planned stope limit. This makes the determination of an optimum mining width crucial during mine design and planning. In addition to the above causes, mining at an angle lower than the true dip of an orebody also results in significant footwall dilution of the ore grade which lowers revenue generated from produced ore thereby negatively affecting the value of a mining project (Zimplats, 2016). Grade optimisation is one of the main value adding processes that can be applied to a metalliferous mining operation (Orelogy, 2017). Hence, there is a need to establish the optimum mining width which addresses both equipment operability and minimisation of ore dilution.

Production rate is one of the constraints in an optimisation process. As indicated by David (2017), the rate of output possible with any stoping method largely depends on the size of the orebody, especially the width (thickness), and the mining cut. Variations in mining cuts will vary the mining grades achieved from stopes. A study was conducted by Anani (2016) which introduced a modeling framework that incorporated the dynamics of the loading, hauling and dumping cycles in the stope width selection. The framework describes how to incorporate the variable cut sequences for each individual stope width, as well as how to optimise sections of the stope width (with distinct duty cycles for material handling equipment) independent of the remaining stope. He stated that the early tools applied in mine system optimisation were based on trial and error but for the past decades, numerous techniques have been developed that make mining system optimisation more efficient. He highlighted OR as one of the main techniques used today. Scientific methods are applied systematically to obtain optimal levels of operation based on the state of the current system (Sharma, 2009). Optimisation techniques applicable to mining cut optimisation are reviewed in the next section.

2.6 Optimisation techniques

There are a number of optimisation techniques which can be applied in optimising mining cut. These include techno-economic evaluation, integer programming, linear programming, mixed integer linear programming, binary programming, branch and bound and metropolis algorithm.

2.6.1 Techno-economic evaluation

One of the optimisation technique that can be applied in mining cut optimisation is the techno-economic evaluation. This involves an analysis of technical and economic factors. The technical factors considered in techno-economic evaluation include, Mineral Reserve calculations, grade and tonnage trends analysis, extraction ratios and mine stability factors, the important ones among them being pillar and panel width to height ratios and pillar safety factors (Musingwini, 2016). According to Diering *et al.,* (2012), a Mineral Reserve is an estimation of the ore tonnages and ounces to be produced stated in a business plan, as derived from the Measured and Indicated Mineral Resource scheduling. There is a direct relationship between mining cut and Mineral Reserves for a room and pillar mine. Key in optimising mining cut would be minimising dilution, maximising ore recovery and supplying a consistent metallurgically balanced feed to the process plant (CIM, 2003). In terms of grade and tonnage, an optimum mining cut aims to achieve maximum metal content recovery from minimum ore tonnage. An economic evaluation requires input such as tonnage and grade information (Wellmer *et al*., 2008). Minnitt (2004) mentioned that in a mining operation, optimal grades can be attained when mining capacity related factors are incorporated in cut-off grade calculations, and in a tabular platinum orebody, the mining capacity is determined by the mining cut. When the mining cut is increased, the tonnage increases but the grade decreases and when the mining cut is decreased the tonnage decreases and the grade increases. Mining cut has an impact on extraction in a room and pillar mine. Variations in mining cut affects pillar strength which is a function of pillar geometry and the strength of the material comprising the pillar (van der Merwe, 1990).

The economic factors considered in a techno-economic evaluation include revenue and cost calculations, profit and loss analysis and NPV computations. In this optimisation technique, economic performance indicators such as processing costs, market price, metallurgical and mining recoveries, tonnage and grade must be taken into account to maximise the NPV of the operation (Nieto and Zhang, 2013). Various mining cuts have different effects on mining cost, revenue generated and the resultant cash flows. An optimised mining cut in a room and pillar mine would result in a maximum NPV among the mining cut options selected for the study.

2.6.2 Branch and bound algorithm

In the Branch and bound approach, the principle is that a set of possible solutions can be divided into smaller subsets of solutions from which the best solution can be determined after systematic evaluation (Taylor, n.d). Taylor (n.d) explained that mixed integer linear programming challenges can be resolved using the branch and bound method through application of the basic steps applied in solving integer problems. Ovanic and Young (1995) applied the branch and bound technique in determining optimal stope layouts. They applied type-two special ordered sets (SOS2) approach in their stope boundary model optimisation. The basic steps for the algorithm according to Gelman *et al.* (2013) are:

- Consider a two-dimensional non-linear function y = f(x);
- The solution is on the curve (x, f(x)); and
- The solution is influenced by two points if it is between the two discrete points, and if it is on a discrete point, then it is affected by that point only. In this case, each discrete point on the curve, $(x_i, f(x_i))$, is given a special variable, λ_i, which signifies the weight of the point in the solution such that:

$$\begin{cases} 0 \leq \lambda_i \leq 1 \\ \qquad\qquad and \\ \sum \lambda_i = 1 \end{cases}$$

The two possible outcomes for the solution can be defined as (Gelman *et al*., 2013):

- One of the discrete points is the solution, in which case the weights given to other points are equal to zero and the weight of that point, Λ_1, is equal to one; and
- The point between two adjacent break points on the curve is the solution. In this case, the weights of these points add up to one and are non-zero. The precise position of the solution, depends on the weights of the two nearby points represented by ($x_i \Lambda_i + x_i +1\Lambda_i+_1$, f($x_i \Lambda_i + x_i+_1 \Lambda_i+_1$)).

The outline of the stope can be determined by optimising its starting and ending locations. Two separate sets are therefore defined for the two end points of the stope.

2.6.3 Metropolis algorithm

The metropolis algorithm is a technique that can be applied to optimise mining cut. Among the members of the computational types that used to be called the "Monte Carlo Method", the metropolis algorithm has been the best and most influential (Beichl and Sullivan, 2000). The Metropolis algorithm is a random walk adaptation, combined with acceptance-rejection sampling which converges on a specified target distribution (Gelman *et al.*, 2013). The basic steps for the Metropolis algorithm according to Gelman *et al.* (2013) are:

1. Choose a starting point θ^0 from a starting distribution, where $p(\theta^0|y) > 0$. This may be a very rough estimate;
2. For all t values 1, 2, …,n sample a proposal θ^* from a "proposal distribution", which must be a symmetric distribution. θ^* is sampled at time t, $J_t(\theta^*|\theta^{t-1})$;
3. Calculate the density ratio using Equation 2.1; and

$$r = \frac{p(\theta^*|y)}{p(\theta^{t-1}|y)} . \qquad (2.1)$$

4. Set θ^t as in Equation 2.2:

$$\theta^t = \begin{cases} \theta^* & \text{with probability min (r, 1)} \\ \theta^{t-1} & \text{otherwise} \end{cases} \quad . \qquad (2.2)$$

2.6.4 Linear programming

In linear programming, one or two variables are involved. For two optimisation variables, solution can be obtained by plotting contours of constraint functions and the objective function (Bhatti, 2000). (Bhatti, 2000) stated that, based on previous experience, some of the variables can be fixed and an idea of the optimum solution may be obtained by a graphical solution for the two most critical variables. Swain (2016) highlighted the following areas of application of linear programming in mining industries:

- Blending of ores and production scheduling;
- Transportation problems;
- Assignment problems;
- Inventory management and optimisation; and
- Project management.

A typical optimisation problem involves finding the best element from a given set. In order to compare elements, A criterion called an objective function f(x) is needed to compare elements. The feasible set is usually defined by Equation 2.3.

$$\{ x \in Rn \mid g_i(x) \leq 0, i = 1, . . . , m \} \qquad (2.3)$$

The optimisation problem is then formulated as:

Maximize f(x)

Subject to g_i (x) ≤ 0, i = 1, . . . , m.

Among the four methods outlined in this section, techno-economic evaluation is more applicable to mining cut optimisation. This study will employ the techno-economic evaluation of the 'steeps' design to determine the optimum mining cut. The drawback of the other three optimisation techniques, is that they are more complex and they do not optimise overall project value as compared to techno-economic evaluation. Techno-economic evaluation considers numerous factors that affect project value (Martinez, 2010).

2.7 Optimisation criteria in mining

Optimisation criteria according to Dowgielewicz *et al.* (2006) are criteria on which the optimisation should be set upon. Usually cost and time oriented criteria are applied in a production process. There are different optimisation criteria applied in mining and the most common ones are; profit maximisation, NPV maximisation, production output maximisation and cost minimisation. Brazil *et al.* (2000) stated that, cost reduction for mining operations is a critical issue for mine developers and operators faced with enormous competitive mineral commodities market place. Efforts were made in the past to apply mathematical methods to maximise the total profit or maximise the net present value of mining projects (Ding *et al.*, 2004). Ding *et al.* (2004) highlighted two conditions under which profit maximisation can be achieved in a mine development project, given the nature of the orebody and production size as well as the development system and these are:

- Every stope to be mined contribute profit; and
- All stopes that generate non-negative profit may be developed and recovered.

The profit maximisation problem can be expressed by Equation 2.4 (Ding *et al.,* 2004):

$$Max, TP = \sum_n Vn \qquad (2.4)$$

s.t. $V_n > 0$, $n \rightarrow$ maximal

Where,

V_n – potential value of recovered mining block;

TP – total development profit; and

n – number of mining blocks.

Other optimisation criteria used in underground mines include technical and safety requirements and production output maximisation. The technical requirements include selective mining for the purpose of maximising grade, which is made possible by a reduction in development spacing; extraction ratio optimisation, and productivity enhancement by development spacing reduction. The safety aspects may include reducing span or coverage of the production areas to improve supervision and reduce safety risks by minimising unsupported spans for better ground stability, which is also achieved by decreasing development spacing (Musingwini, 2016).

The major drawback with these optimisation criteria is that they do not optimise overall project value. For instance in profit maximisation, resource utilisation is not optimised as only high grade ore is targeted to maximise profit (Nhleko *et al*., 2018). Table 2.1 shows the various algorithms for stope layout optimisation and the optimisation criteria that are applied by these algorithms.

Table 2.1: Algorithms and optimisation criteria

(Source: Nhleko *et al*., 2018)

Algorithm	Classification	Mining method	Dimensional space	Optimization criteria	Constraints	True optimality
Dynamic Programming	Exact	Block caving	2D	Economic value	Stope dimension; draw control	No
Downstream Geostatistical	Exact	Cut-and-fill; sublevel stoping	2D	Geometry	Stope dimension; hangingwall and footwall slope angle	No
Branch and Bound	Exact	All	1D	Economic value	Stope dimension (stope length)	Yes
Octree Division	Heuristic	All	3D	Grade/vein	Stope dimension, cut-off grade and mining cost	No
Floating Stope	Heuristic	All	3D	Economic value	Stope dimension	No
Multiple Pass Floating stope	Heuristic	All	3D	Economic value	Stope dimension	No
Maximum Value Neighbourhood (MVN)	Heuristic	All	3D	Economic value	Stope dimension	No
Topal and Sens heuristic	Heuristic		3D	Economic value	Stope dimension	No
Network Flow	Heuristic	Sublevel stoping	3D	Grade/Economic value	Stope dimension, footwall angle and hanging wall angle	No
Sandanayake	Heuristic	Sublevel stoping	3D	Economic value	Stope dimension, pillar separation and level location	No

From Table 2.1, it can be observed that economic value and stope dimension are the commonly applied optimisation criteria and constraint by most of these algorithms, respectively. The economic value criterion was successfully applied in most underground optimisation problems and it is for this reason that it has been selected as the optimisation criterion for this research study.

Mining is a complex business that demand a constant valuation of risk as the mine project value is affected by numerous fundamental economic and physical uncertainties. These uncertainties include grade, commodities prices, costs, tonnages, schedules, and environmental requirements, among others, which are known with less certainty at project onset (Martinez, 2010). Economic value as an optimisation criterion, involves maximising NPV by maximising profits and minimising cost. In some cases, this is achieved through optimisation of cut-off grade and extraction sequence (Myburgh *et al.*, 2014). According to Martinez (2010), when an option exist for a new project, the value has to be determined as this influences the final investment decisions on a project.

2.8 Chapter summary

Optimisation is carried out in underground mines to maximise project economic value and this is done at various levels of the business. Underground optimisation starts from the generation of a geological model which is the foundation upon which subsequent mine design and planning processes are done. Mining stope boundaries are optimised to generate maximum economic value considering the physical and geotechnical constraints of the orebody. Production schedules are also generated using the selected mine designs and available resources with the best schedules generating optimum net present values. Factors to consider when optimising layouts include LOM production output; rock engineering; ventilation; logistics and economics. All these factors must be carefully considered and incorporated in the creation of safe production plans. Other areas which are also considered in optimisation of underground mines are sound equipment maintenance and replacement systems and policies.

Some algorithms have been developed to date for optimising stope layouts. These are broadly classified into rigorous and heuristic algorithms. Mining cut was also reviewed and not much work has been done in this area which is the area of focus for this research study. From the literature review, mining cut is a major value driver in underground mines. There are various optimisation criteria employed in underground mine optimisation. These include profit maximisation, cost minimisation, production maximisation and economic value optimisation. Most optimisation work applies economic value as the optimisation criterion and this has been selected for application in this research. The methodology which was applied in this research study is described in the next chapter.

3 RESEARCH METHODOLOGY

3.1 Chapter overview

This chapter describes the research design and methodology that was used in this study. The methodology includes various research techniques or instruments, and include both present and historical information. This chapter outlines the research design, sampling, research procedure, data collected and the data analysis technique that was used in this study.

3.2 Research design

The critical process that converts an idea, interest, or question into a significant and focused investigation of social or physical process is the research design (Kaapanda, 2011).

There are various research designs which can be employed in conducting research studies and this study employed the exploratory research design. Characteristics of this research design include (RBS Guide, 2010):

- An investigation of phenomena or situations that are not familiar as is the case with steep resource mining in platinum mining; and
- It provides a description to understanding or explaining a situation.

This research design is appropriate to this study as no major work was done on the optimisation of the ‘steeps’ layout in mechanised environments before, meaning it is an investigation of situations that are not familiar.

3.3 Sampling of mining blocks for analysis

The current ‘steeps’ section at Zimplats’ P4 mine is comprised of six mining blocks. Assuming that these mining blocks are homogeneous, random sampling was employed to select two blocks required for use in the study. Random sampling is easy and less time consuming and is most appropriate in situations where sampling units exhibit uniform characteristics. Out of the six blocks, ‘Steeps’ block 1 to ‘Steeps’ block 6, only ‘Steeps’ block 3 and ‘Steeps’ block 4 were selected.

3.4 Data collected

Data on the following parameters was collected for use in the techno-economic evaluation:

- Fixed and variable costs (US$/t),
- Processing costs, that include, concentrating and smelting cost (US$/t),
- Transport costs for ore and smelter matte (US$/t),
- Shared services cost (US$/t),
- Royalty for platinum, palladium, rhodium and gold (US$/Oz),
- Tax payments ($),
- Real discount rate (%),
- Metal prices for palladium, platinum, rhodium and gold (US$/Oz),
- Concentrator recoveries (%),
- Mining production rate (t/m), and
- 4E head grade – Pt (g/t); Pd (g/t); Rh (g/t) and Au (g/t).

The parameters above were collected because they are key input into the techno-economic evaluation optimisation process. Data on mining costs was obtained from the mine’s monthly operating cost reports. Average operating costs (fixed and variable) per tonne of ore mined was determined from past production performance reports. The same approach was applied in the determination of processing costs (smelter and concentrator). Transport costs include ore haulage cost from the underground mines stockpiles to the mill and the transportation of smelter matte from Zimbabwe to South Africa. Mining shared services costs include costs of services from departments that service all the mines. The departments include central human resources, technical services and finance departments. All the costs used in this research report were sourced from Zimplats finance department. Royalty rates and taxes were also considered as determined by the Government of Zimbabwe (GoZ). Metal prices applied in this model were obtained from Zimplats’ website. Ore production and grade schedules were derived from the created mine designs through the application of the Vulcan Gantt scheduler (VGS).

3.5 Data analysis

In this study, a number of research instruments were used, and these include: use of 3D mine design software, trends analysis and financial valuation techniques. Six designs were produced in Vulcan software. The designs had different mining cuts which are 2.00m, 2.10m, 2.20m, 2.30m, 2.40m and 2.50m cuts. Mineral Reserves for each option were determined. In order to successfully achieve this, the mine's geological block model created by the company's exploration team was used in this study. Zimplats' modifying factors were applied for each mining cut option, that is, mining loss, seam dilution width, extraction ratio and specific gravity.

The design blocks and results from the Mineral Reserve calculations were imported into VGS for scheduling. After scheduling the blocks, graphs were produced for each option and analysed. Trends analysis focused on variation of grades and volumes to be extracted with variation in mining cuts. Costs and revenue analysis was also done for each mining cut. Total costs for each option was calculated from mining, processing and transport costs. The option with the lowest costs was considered the optimum mining cut. Total revenue was derived from PGM grades, PGM prices, tonnage produced for each mining cut option, processing recoveries and payability. The option with the highest revenue was considered the optimum mining cut. This evaluation produced different optimum mining cuts depending on the optimisation criteria used.

The DCF method was then used to consolidate results from the techno-economic evaluation to come up with the optimum mining cut. The revenue per mining cut was calculated using the Zimplats financial model and commodity prices as updated on the Zimplats website. Scenarios for the different mining cuts were then analysed and cash flow analysis carried out to determine the option with the highest NPV. Sensitivity analysis was performed on the following parameters for the schedule:

- PGE price: -20%, -10%, +10%, +20%;
- Discount rate: -20%, -10%, +10%, +20%; and
- Processing recovery: -20%, -10%, +10%, +20%.

The above ranges were used as they are the standard ranges adopted by Zimplats after having carried out risk profiling for all the input parameters to the organisation's financial model.

3.6 Ethical considerations

A clearance letter from Zimplats to obtain access to technical data for use in this research was obtained. The objectives of the research project, its methodology and associated timelines were outlined in the letter. The letter was presented to all relevant authorities to ensure transparency and accountability of all accessed data and generated results. The clearance letter was submitted with the research proposal.

3.7 Chapter summary

The research methodology which specifies how the study was conducted was discussed in this chapter. The research employed a quantitative approach with various research instruments used to collect and analyse data. A random sampling method was used where two production blocks were selected for this research. The chapter also discussed data collection and outlined all the data collected for use with the Zimplats financial model. The techno-economic evaluation model used in determining the optimum mining cut was described in this chapter. The chapter also highlighted the sensitivity analysis that was done to test the robustness or sensitiveness of the NPV of the determined optimum mining cut to changes in the input parameters of the financial model used for DCF analysis. A clearance letter was obtained from Zimplats and confidentiality was enforced as specified in the letter. The clearance letter was submitted with the research proposal. Analysis of the findings from this study is done in the following chapter.

4 TECHNO-ECONOMIC OPTIMISATION RESULTS AND ANALYSIS

4.1 Chapter overview

The chapter gives a brief review of the room and pillar mining method employed at Zimplats mine. The modified layout for the mining of ‘steeps’ is also discussed. The general grade profile for P4 is analysed to determine the position of the mining cuts on the mineralised horizon of the orebody. An analysis of the results from the Mineral Reserve calculations is also done for each mining cut option and graphs are produced showing the tonnage and grade profiles. Comparisons are made of the grade and tonnage schedules and the estimated life of mine for each mining cut option. Mining costs, revenue generated for each mining cut and extraction ratio analysis is also done in this chapter. Economic valuation using the discounted cash flow technique is done for all the mining cut options. This process determines the best mining cut which should be employed in the mining of the ‘steeps’ orebody at Zimplats. The optimum mining cut should have the highest NPV compared to other investigated mining cut options but satisfying all constraints. The optimum mining cut design must also be robust enough to withstand any changes in key input variables to the financial model and this is determined by a sensitivity analysis which is done in this chapter.

4.2 Room and pillar mining method

As mentioned in Chapter 1, the orebody at Zimplats is mined using room and pillar method. Figure 4.1 shows the room and pillar mining method typical design at Zimplats.

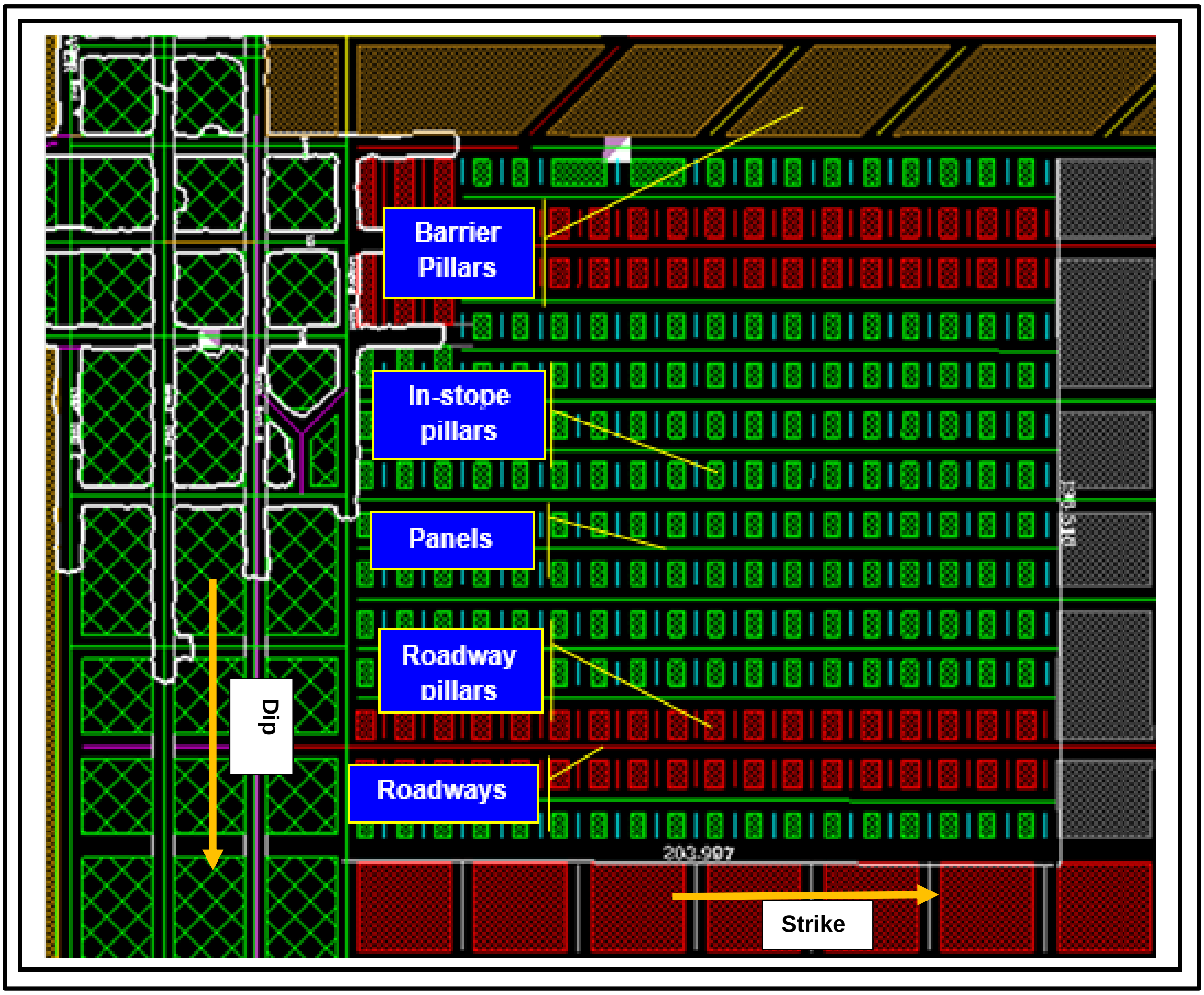


Figure 4.1: Zimplats room and pillar mining method

(Generated in Vulcan software)

The room and pillar mine design is based on the paddock pillar system. The 'steeps' section includes regional pillars at 200m spacing and in-stope and roadway pillars with dimensions shown in Table 4.1. The overall extraction ratio for this mine design is 67%. An empirical formula to establish the minimum pillar dimensions in different ground classes is applied in designing the mining method. The mine is divided into paddocks of approximately 205m x 200m, bounded by large pillars on all four sides. The width of roadways and panels is 6m. Decline pillars must have a minimum dimension of 21m x 20m and the decline panels measure 6m x 3.5m high. Ends mined in bad ground have a maximum panel width of 6m and the mining is guided by site specific rock mechanics recommendations from the rock engineering department.

Table 4.1: Zimplats room and pillar mine design dimensions

Excavation	Width (m)
Panels	6.0
Panel holings	6.0
Roadways	6.0
Roadway holings	6.0
Pillar	**Dimension (m) (Dip x Strike)**
In-stope – minimum dimensions	6.0 x 6.0
Roadway – minimum dimensions	7.0 x 7.0
Barrier	25.0 x 26.0

4.3 'Steeps' mining layout

As mentioned in Section 1.2, the orebody's mineralisation include Pt, Pd, Rh and Au and the concentration of each mineral varies across the mining face from the footwall to the hanging wall. Figure 4.2 shows a grade profile across a typical mining face for P4 mine. The figure was drawn from data in Appendix 7.1.

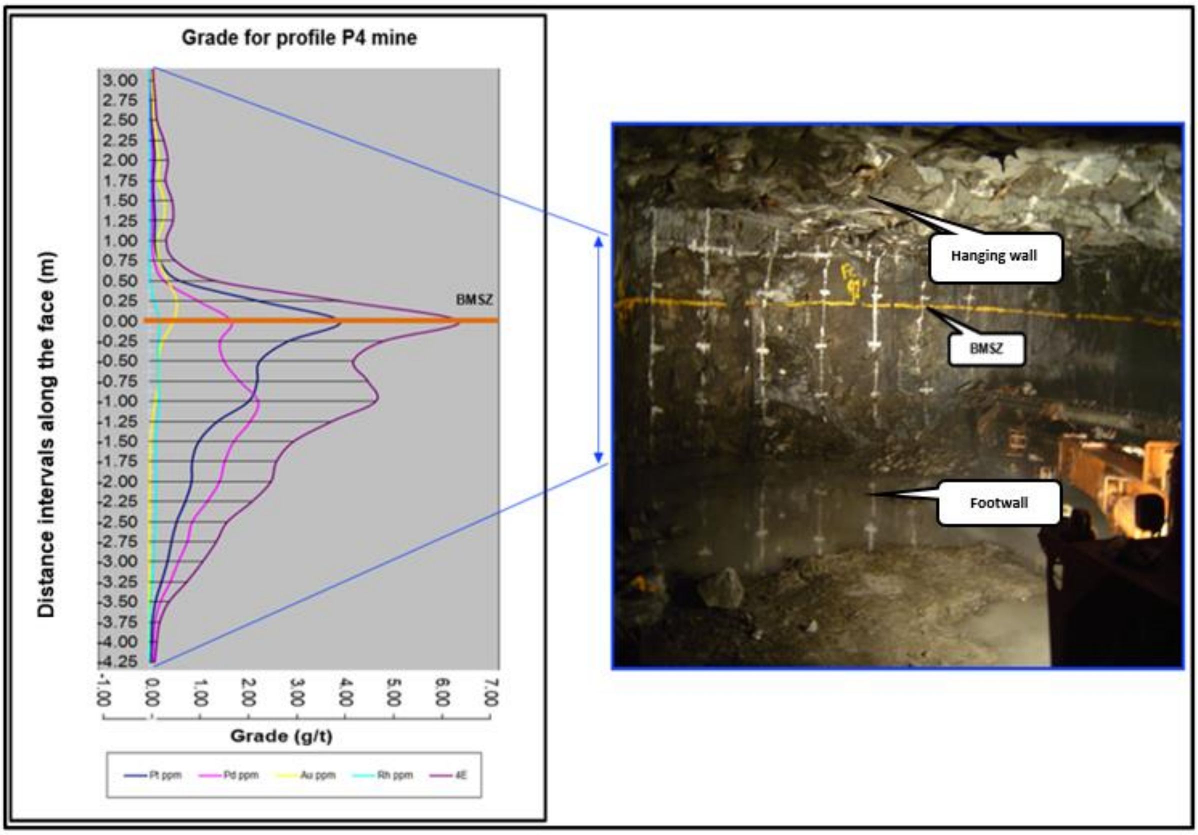


Figure 4.2: P4 mine grade profile

From the grade profile, it can be observed that there is a sharp drop in PGM grade values from the BMSZ (0.00m) to the upper seam (0.75m). Beyond 0.75m above the BMSZ, the PGM values gradually drop to zero. Any mining above 0.75m from the peak seam would only include more waste into the ore stream, thus this study fixed the upper stope limit at 0.75m above the BMSZ and varied the mining cut by varying the stope footwall position. Fixing of the hanging wall position at 0.75m above the BMSZ also assist in guiding the mining teams in mining the correct ore horizon as any further reduction in the distance between the hanging wall and the peak platinum seam may result in the loss of the peak platinum seam into the hanging wall. This will consequently have a negative impact on the overall 4E grade achieved. Figure 4.3 shows the selected mining cut options (i.e. 2.50m, 2.40m, 2.30m, 2.20m, 2.10m and 2.00m) analysed in this study as explained in Chapter 3.

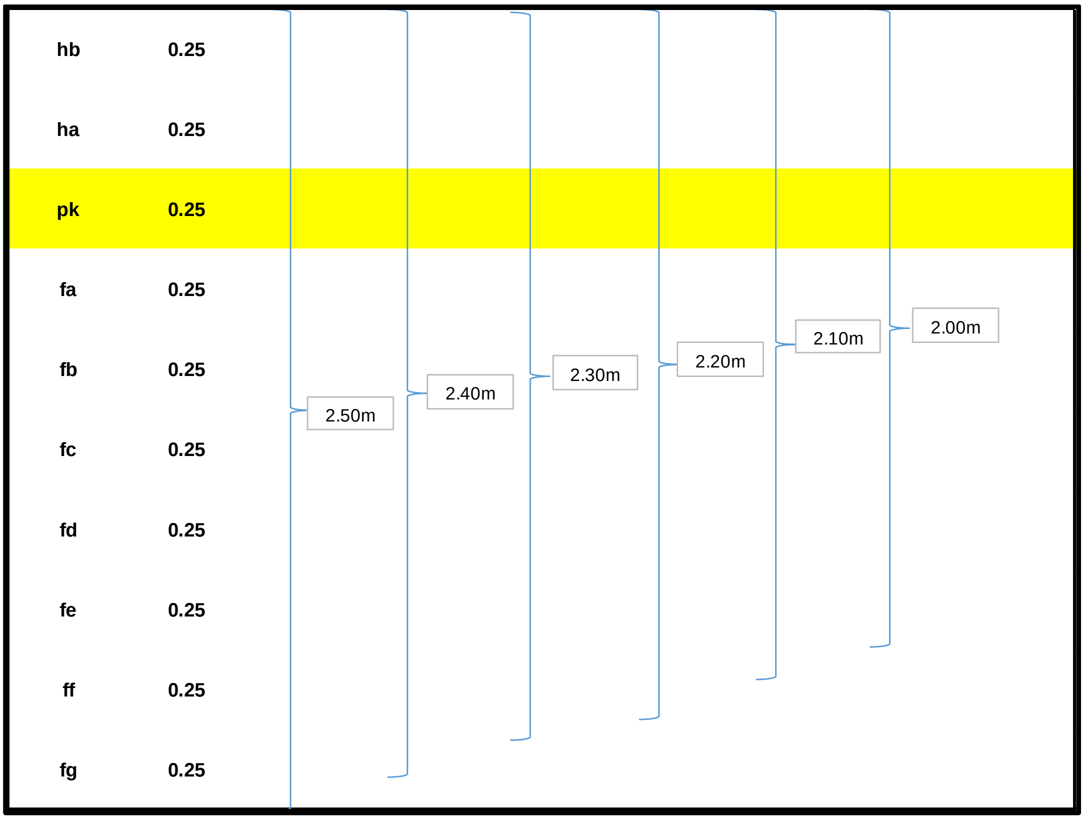
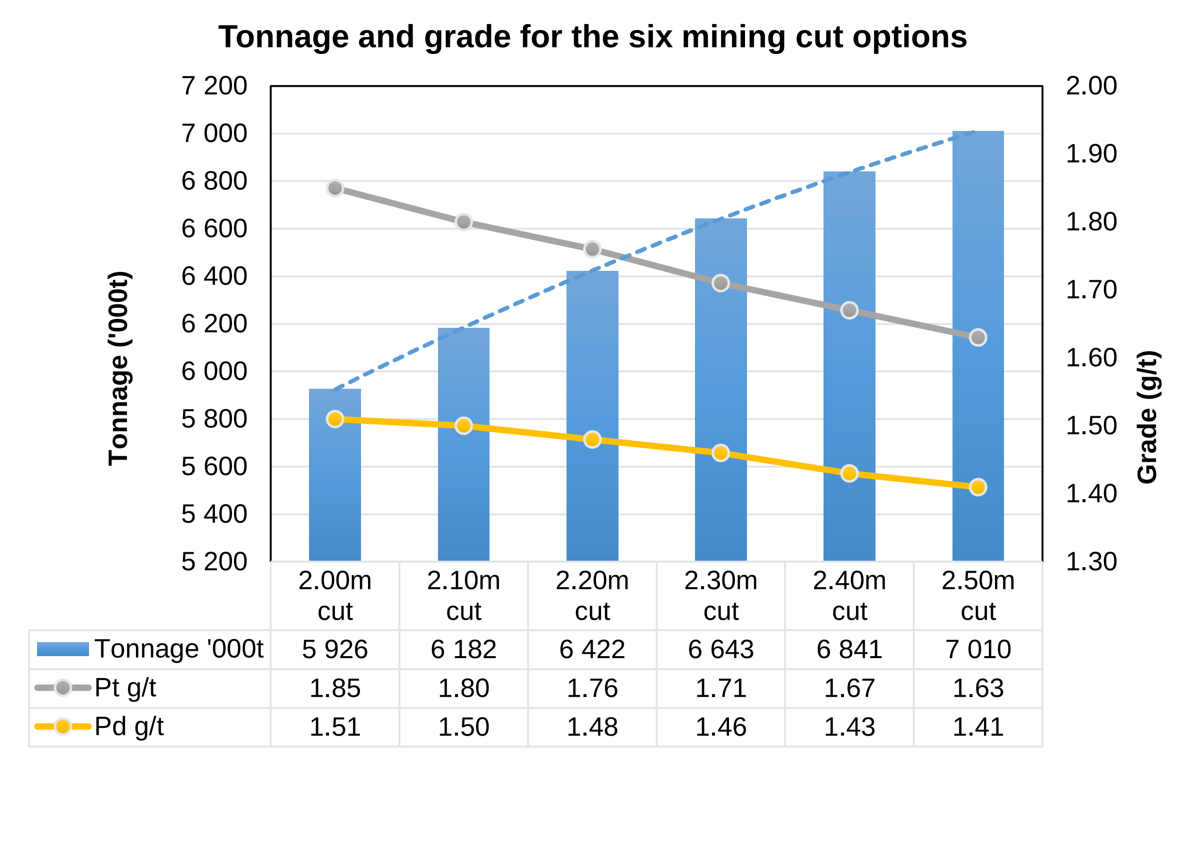


Figure 4.3: Selected mining cuts

The orebody at Zimplats is tabular and in the block model, the orebody is comprised of mineralisation layers (fq to hm) as shown in Appendix 7.1 with every layer representing a 0.25m seam. The 2.50m mining cut covers the seams hb to fg and the 2.00m mining cut covers seams hb to fe. The other mining cuts are distributed evenly between these two options with 0.1m variations between subsequent options. Currently, 'steeps' at the mine are being mined at 2.50m mining cut and this mining cut option has been included in this study. The following section discusses the Mineral Reserve calculations done for the six mining cut options.

4.4 Mineral Reserve calculations

The geological block model for P4 mine was used in the calculation of Mineral Reserves for the 'steeps' section for each mining cut option. Figure 4.4 shows the results of the Mineral Reserve calculations for the six mining cuts. The results consist of the PGM grades (Pt and Pd) and the ore tonnages for the selected mining blocks for the six mining cut options. Appendix 7.2 shows the detailed results of the Mineral Reserve calculations completed for each mining cut option.



	2.00m cut	2.10m cut	2.20m cut	2.30m cut	2.40m cut	2.50m cut
Tonnage '000t	5 926	6 182	6 422	6 643	6 841	7 010
Pt g/t	1.85	1.80	1.76	1.71	1.67	1.63
Pd g/t	1.51	1.50	1.48	1.46	1.43	1.41

Figure 4.4: Mineral Reserve calculation results

Rh and Au results are not shown in Figure 4.4 as they have marginal grades which largely remain constant across the mining profile as seen in Figure 4.2. Pt and Pd grades decrease gradually as the mining cut increase from 2.00m to 2.50m but, the tonnage increases. This is due to an increase in tonnage from the footwall (fg to fe seams) which according to the grade profile (Figure 4.2) have low PGM values. A grade reduction factor (GRF) is applied to determine the final grades. GRF is the overall percentage factor representing all factors that affect grade of mined ore. These factors include (Zimplats, 2012):

- Geological losses: 2 to 20%;
- Pillar factors: 15 to 25%;
- Grade dilution: 6 to 12%; and
- Tonnage dilution: 6 to 18%.

Based on the Mineral Reserve calculation, if the objective is to maximise grade, the 2.00m mining cut would be the optimum mining cut because it is the option with the highest grades of Pt and Pd of 1.85g/t and 1.51g/t respectively, which will result in highest average 4E grade. If the objective is to maximise Mineral Reserve tonnage, the 2.50m mining cut would be the optimum mining cut with 7 010 000t. An analysis of the production schedule for each mining cut was done.

4.5 Production schedules

Vulcan Gantt Scheduler was used to generate production schedules for the different mining cut designs using the Mineral Reserve calculation results and the mine design. Figure 4.5 shows the generated schedule for the 'steeps' blocks under study coloured by periods.

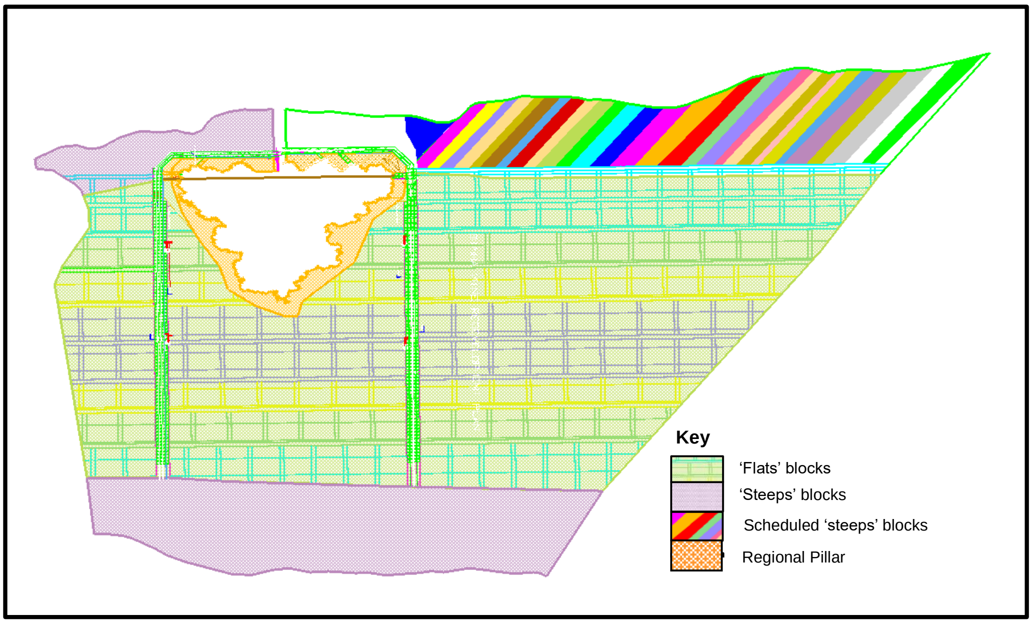


Figure 4.5: 'Steeps' mine design schedule

Different block colours indicate different production periods in the generated schedule. The blocks have different width dimensions indicating variations in average block advance for various block geometries.

4.5.1 Production schedule for the 2.00m mining cut

Figure 4.6 shows a graphical presentation of the 2.00m mining cut schedule report with the tonnage mined over the life of the scheduled blocks and the 4E grades achieved during that period. Appendix 7.3 shows the data used to draw Figures 4.6 to 4.11.

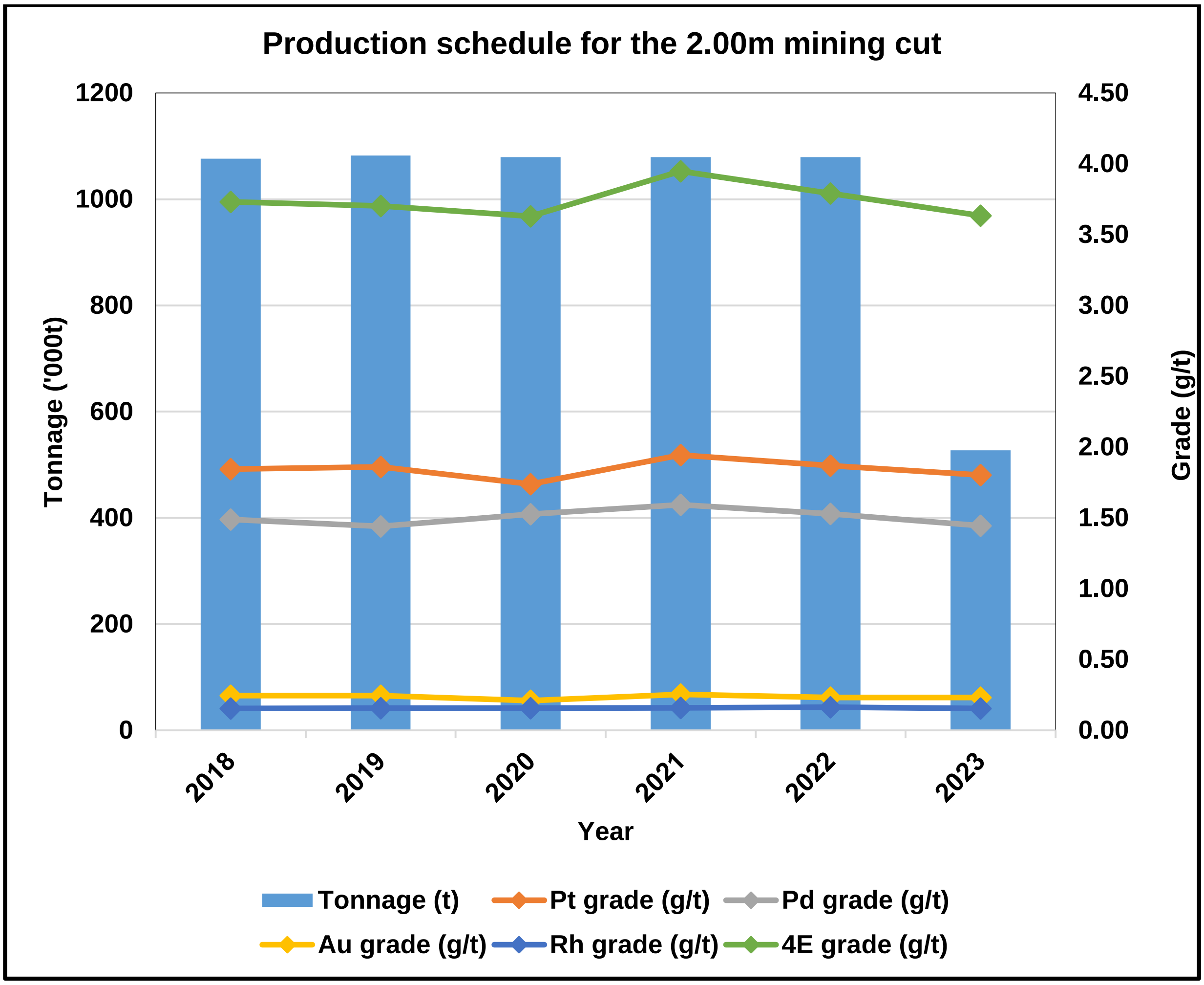


Figure 4.6: Production schedule results for the 2.00m mining cut

From Figure 4.6 it can be observed that the blocks will be depleted in six years at an average production rate of 1 080 000t per year. The average 4E grade is 3.75g/t and the 4E grade trend can be sustained for that production period. A marginal increase in grade is, however, observed after 2020 which is attributed to a patch of high in-situ grade that will be encountered during mining. The grade of individual metals (Pt, Au and Rh) remain fairly constant throughout the project life. Of the four metals, Au and Rh have the lowest grade values, that is below 0.50g/t. Pt and Pd contribute more to the combined 4E grade with Pt averaging 1.85g/t and Pd 1.51g/t.

4.5.2 Production schedule for the 2.10m mining cut

Figure 4.7 shows the production schedule results for the 2.10m mining cut.

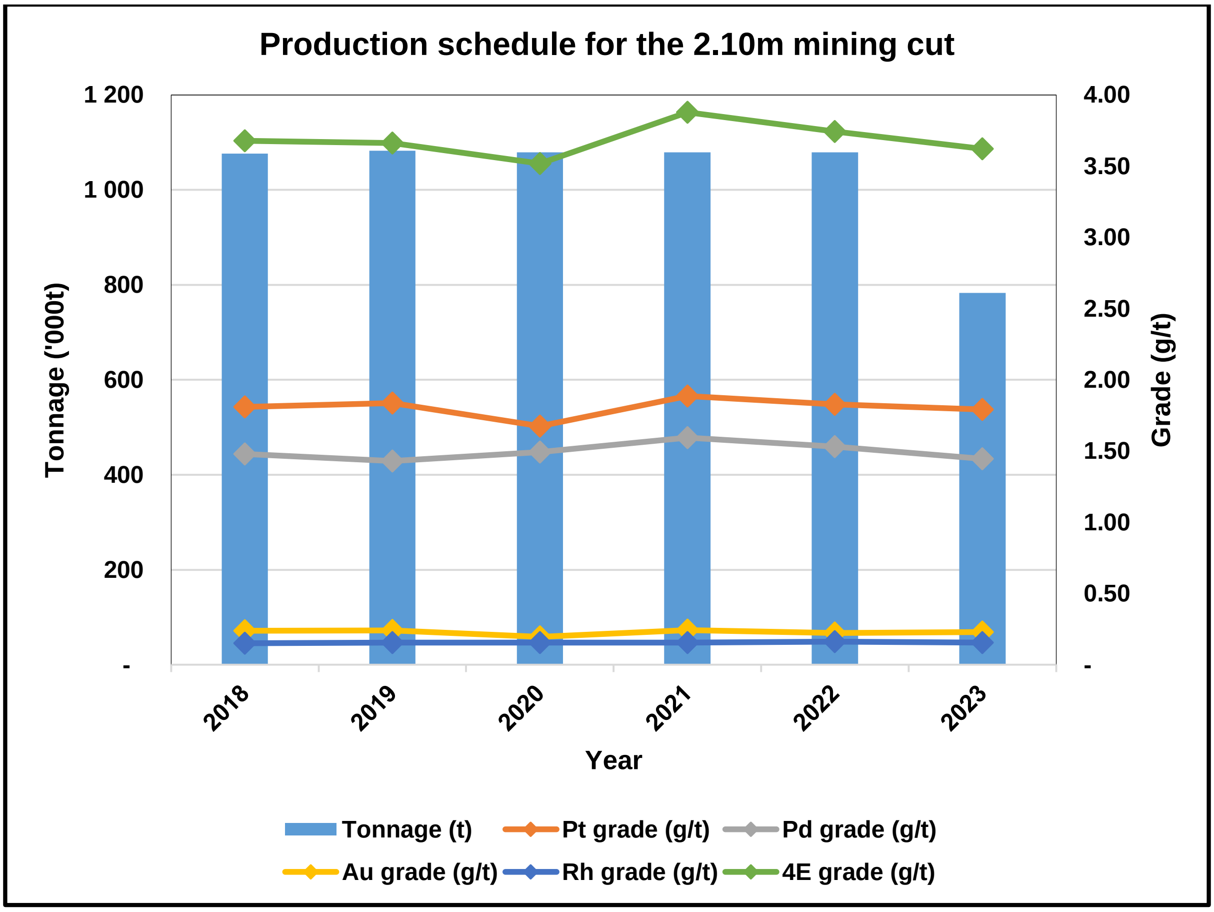


Figure 4.7: Production schedule results for the 2.10m mining cut

Figure 4.7 shows that the average annual production rate is the same as for the 2.00m mining cut but the life of project will be prolonged as the blocks will be depleted in the third quarter of 2023 compared to the second quarter of 2023 for the 2.00m mining cut schedule. This is due to the increased stope width, which has a higher Mineral Reserve tonnage. However, the grade profile shows that the grades are lower compared to the 2.00m mining cut even though the general trends largely remain the same. The average 4E grade is 3.69g/t largely due to a drop in Pt grade from 1.85g/t to an average of 1.80g/t. Pd dropped marginally from 1.51g/t to 1.50g/t and, Au and Rh remained below 0.50g/t. The drop in grades is due to an increase in tonnage contribution from the footwall of the mining profile which has low PGM grade values.

4.5.3 Production schedule for the 2.20m mining cut

Figure 4.8 shows the production schedule results for the 2.20m mining cut.

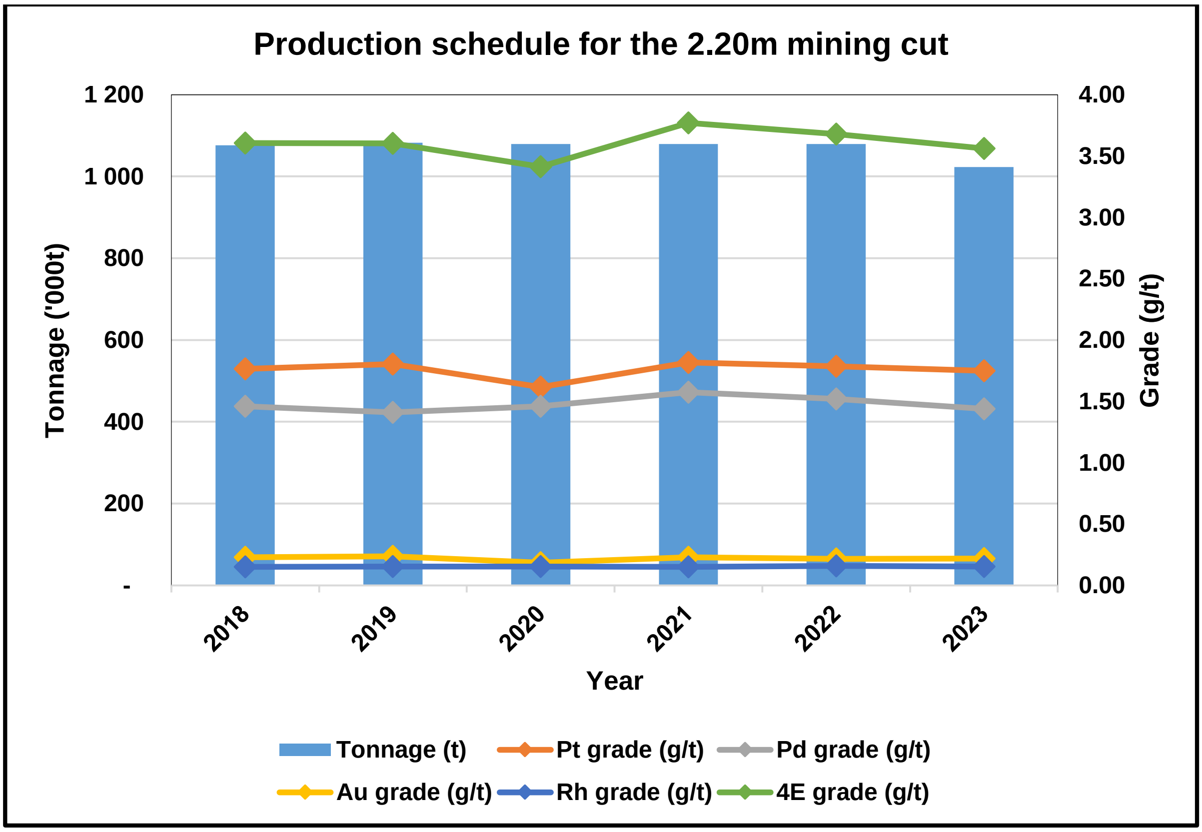


Figure 4.8: Production schedule results for the 2.20m mining cut

As shown in Figure 4.8, the 2.20m mining cut design results in a longer project life than the 2.00m and 2.10m mining cuts. With the 2.20m mining cut, the scheduled blocks will be depleted in the fourth quarter of 2023, which is three months more than the 2.10m mining cut design and six months more than the 2.00m mining cut design. However, the combined 4E PGM grade drops to an average of 3.61g/t as Pt grade drops to 1.76g/t and Pd drops to 1.48g/t as additional tonnage is mined from the footwall with low Pt and Pd grades. No major changes are observed on grades for the other two metals, Au and Rh.

4.5.4 Production schedule for the 2.30m mining cut

Figure 4.9 shows the production schedule results for the 2.30m mining cut.

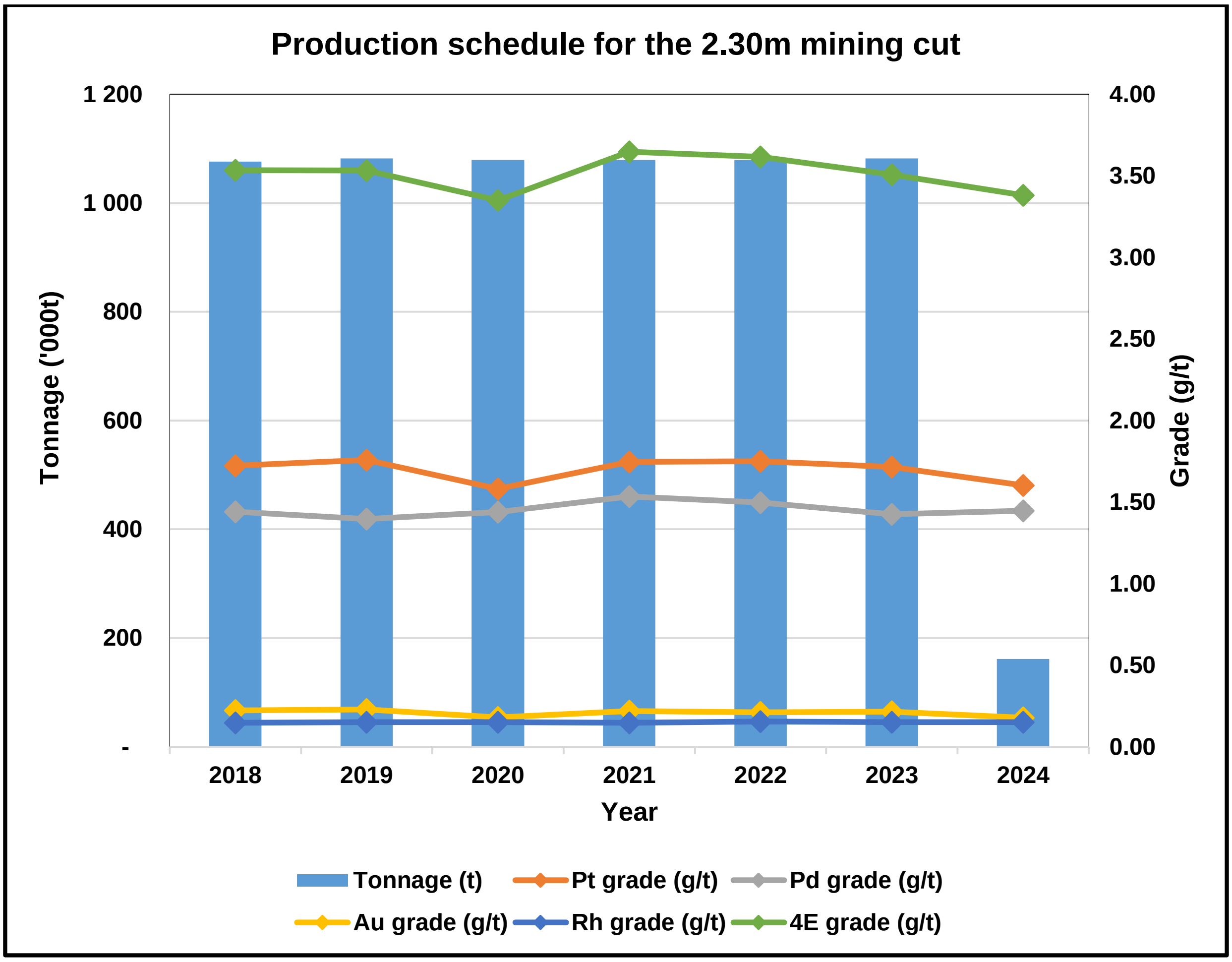


Figure 4.9: Production schedule results for the 2.30m mining cut

Figure 4.9 shows that the 2.30m mining cut results in a mine design with a longer project life than the other three mining cuts discussed. With the 2.30m mining cut, the scheduled blocks will be depleted in the first quarter of 2024, that is, three months later than the 2.20m mining cut design. The Pt grade however further drops to an average of 1.71g/t resulting in an average 4E grade of 3.53g/t. This is due to more tonnage being mined from the footwall of the 2.30m mining profile compared to the other mining cuts (2.00m, 2.10m and 2.30m) discussed.

4.5.5 Production schedule for the 2.40m mining cut

Figure 4.10 shows production schedule results for the 2.40m mining cut.

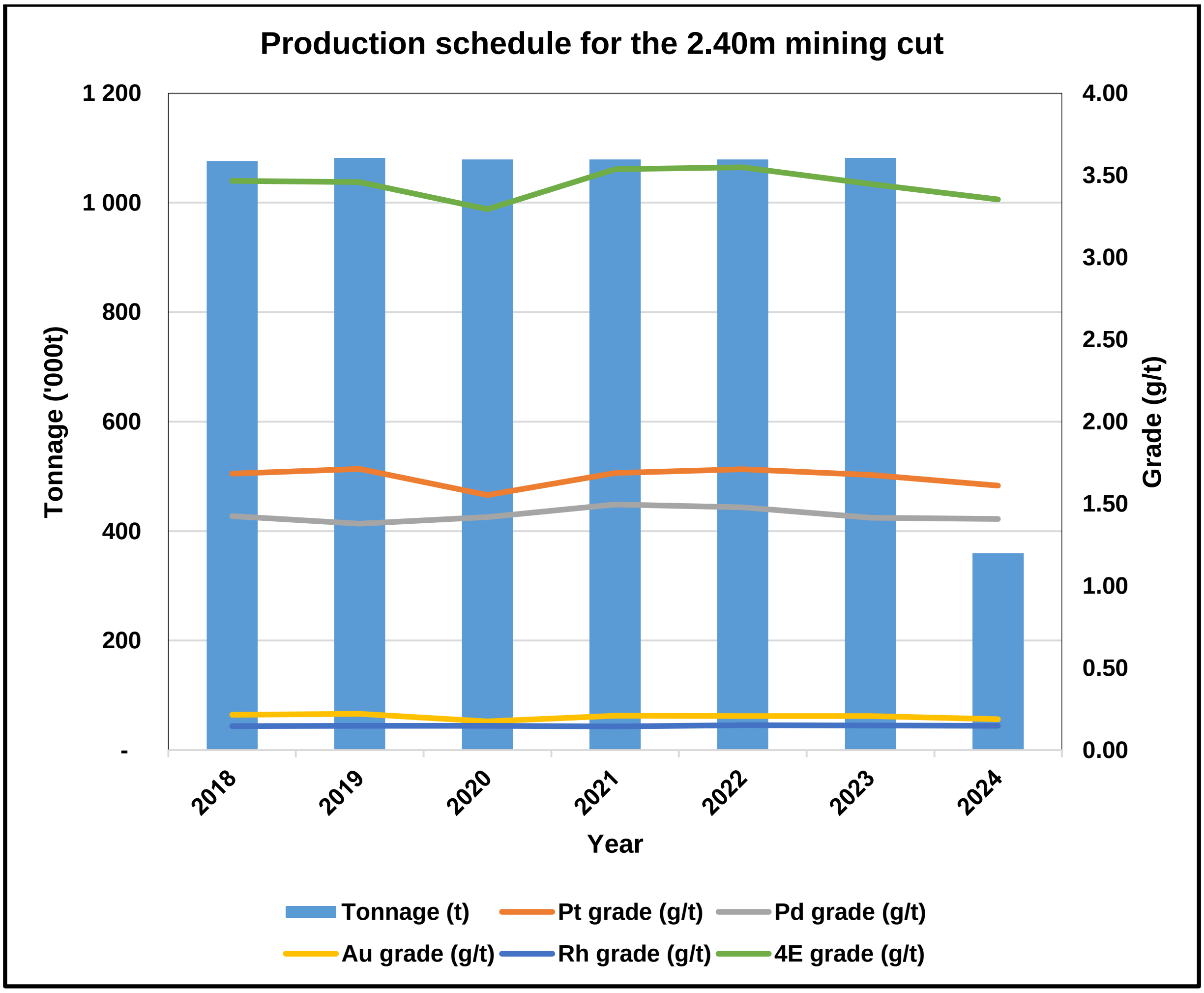


Figure 4.10: Production schedule results for the 2.40m mining cut

Figure 4.10 shows that the 2.40m mining cut results in a mine design with a longer project life than the other four mining cuts discussed. With the 2.40m mining cut, the scheduled blocks will be depleted in the second quarter of 2024, that is, three months later than the 2.30m mining cut design due to increased tonnage. The Pt grade however further drops to an average of 1.67g/t resulting in an average 4E grade of 3.45g/t due to more tonnage contribution with depressed grades from the footwall.

4.5.6 Production schedule for the 2.50m mining cut

Figure 4.11 shows the production schedule results for the 2.50m mining cut.

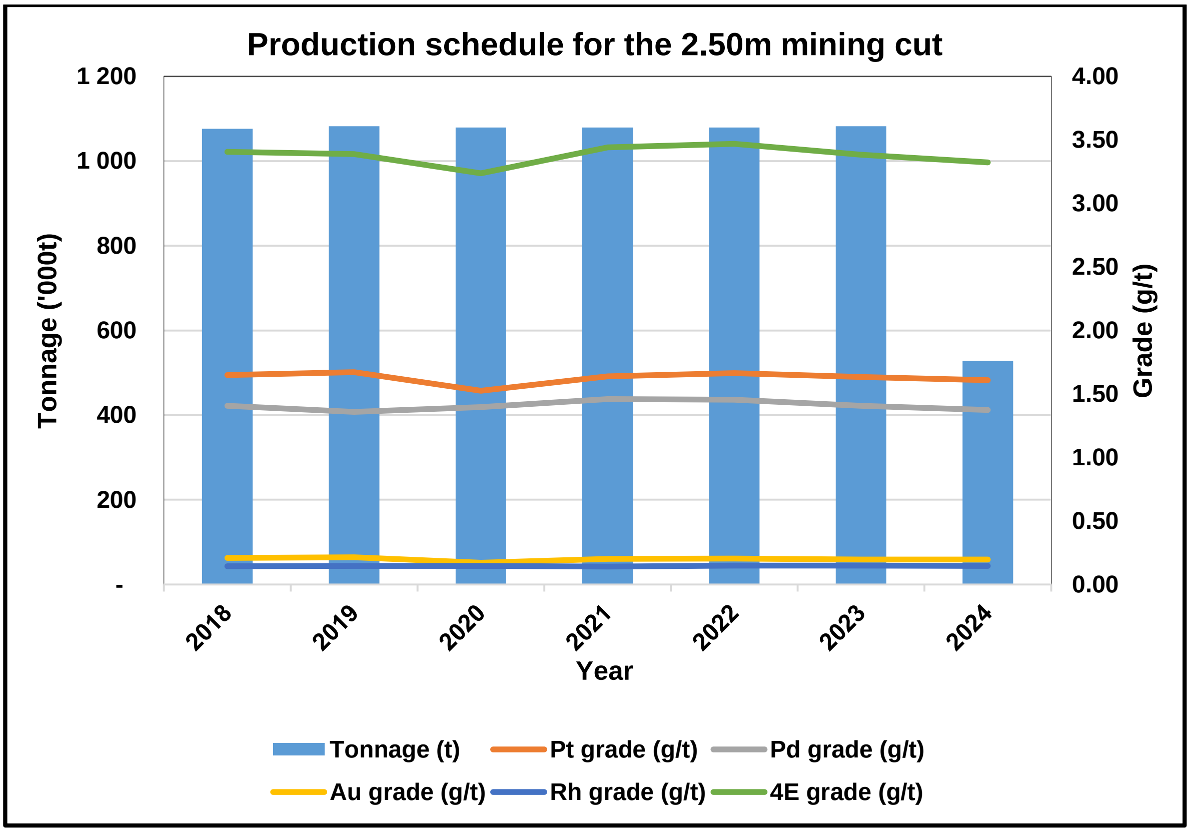


Figure 4.11: Production schedule results for the 2.10m mining cut

Figure 4.11 shows that the 2.50m mining cut results in a mine design with a longer project life than all the other mining cuts discussed in this study. With the 2.50m mining cut, the scheduled blocks will be depleted in the third quarter of 2024, that is, three months later than the 2.40m mining cut design due to increased tonnage. The Pt grade however further drops to an average of 1.63g/t resulting in an average 4E grade of 3.38g/t due to more footwall mining. All the results discussed in this section show a general trend of an increase in LOM and declining grades with an increase in mining cut from 2.00m to 2.50m. The production rate remains the same averaging 1 080 000t per year for each design option under consideration.

Based on production schedule results, if the objective is maximising LOM, the 2.50m cut will be the optimum mining cut because the schedule has the longest life of mine compared to the other five mining cut options. Though this option has lower PGM grades compared to the other mining cut options, it produces more metal content due to the increased tonnage. If the objective is to maximise grade, 2.00m is the optimum cut. A further analysis of each mining cut was done based on the implication of the mining cuts on tonnage.

4.5.7 Tonnage comparisons for different mining cuts

Figure 4.12 shows the graph of tonnage profiles for the six mining cut options for the period 2018 to 2024.

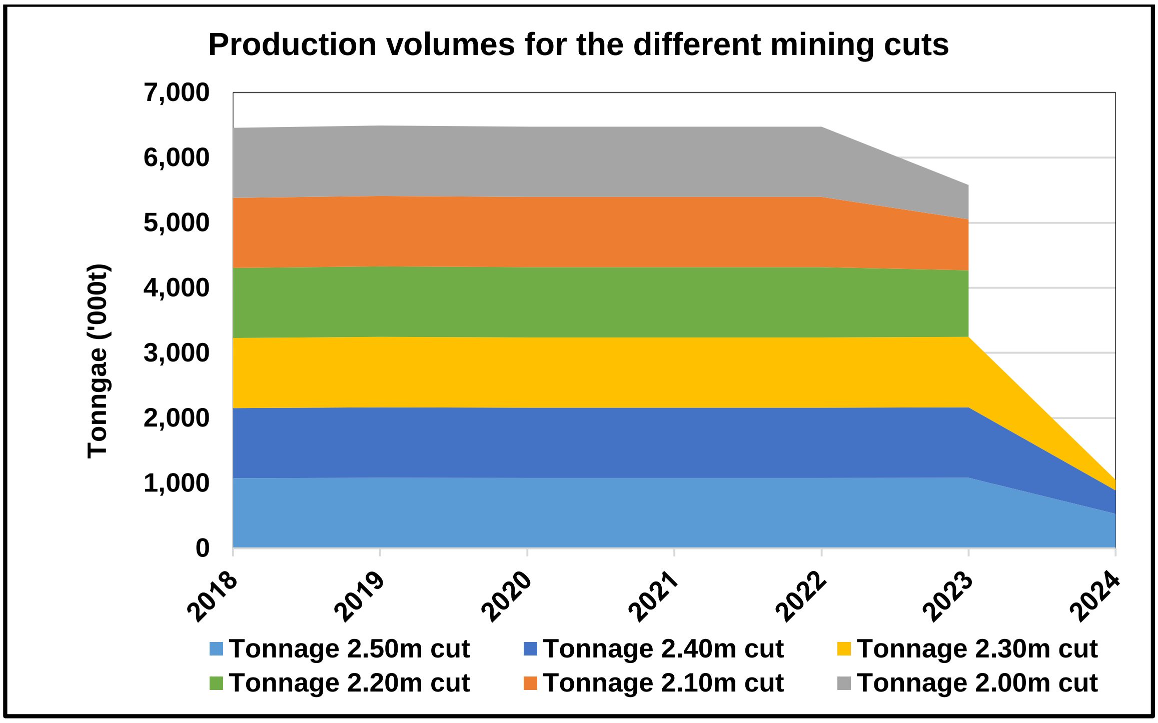


Figure 4.12: Tonnage comparisons for different mining cuts

As the mining cut is increased, the life of mine also increases if the production rate remains constant. A higher mining cut also ensures maximum Mineral Resource utilisation though this may not result in optimum project value. A longer LOM entails production of more tonnage which ultimately results in more metal content. Figure 4.12 shows that the mining blocks will be depleted in 2023 with a 2.00m mining cut and in 2024 with a 2.50m mining cut. The drawback of a longer LOM is risk of substitution as different mineral deposits are discovered and technology advances. If the optimisation criterion is production volume maximisation, then the 2.50m mining cut would be the optimum mining cut amongst the six mining cut options. However, mine optimisation usually require a number of factors to be considered to achieve true optimality, which creates value for the mining organisation. Therefore, this study considers other parameters such as grade, extraction ratio and ore tonnage in determining the optimum mining cut in the mining of the ‘steeps’ resources at Zimplats.

4.5.8 Grade comparisons for different mining cuts

Figure 4.13 shows the 4E grade trends for the different mining cuts.

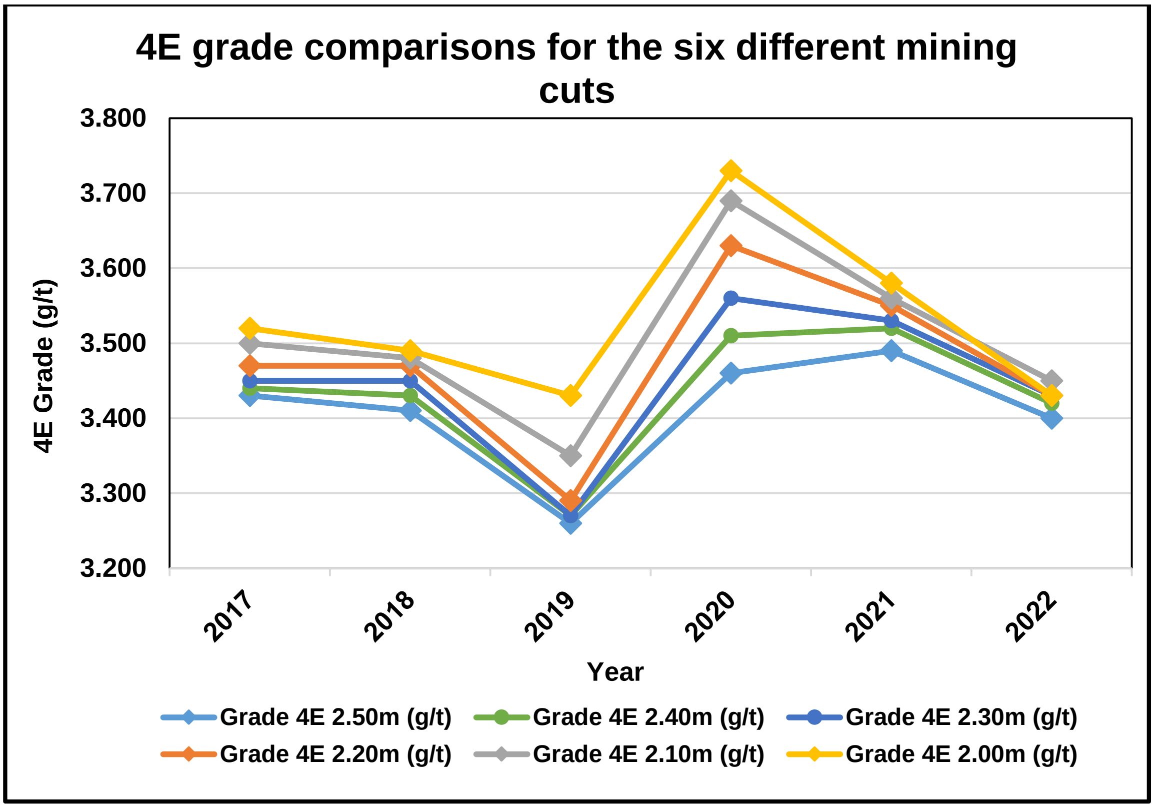


Figure 4.13: Grade comparisons for different mining cuts

The 2.00m mining cut has the highest 4E grade and the 2.50m mining cut has the lowest grade among the six different mining cuts under review. If the optimisation criteria is solely grade maximisation then the 2.00m mining cut would be the best option but however, this may not guarantee true optimality. True optimality cannot be achieved because, as the mining cut narrows and the grade increases, the tonnage will decrease and the metal content also decrease. For true optimality to be achieved there needs to be a balance in all the key value drivers. In addition to tonnage and grade, the different mining cut will have different implications to Mineral Reserve extraction rates.

4.6 Extraction ratio analysis

This section presents results for the analysis done on the extraction ratios for the six mining cut options. The design pillar width to height ratio for the 'steeps' design at Zimplats is fixed at 2.4 (Zimplats, 2017). With the in-stope pillars being 6m x 6m, the in-stope extraction ratio for this design is 75%. Figure 4.14 shows how the extraction ratio is calculated for a typical 'steeps' mining block.

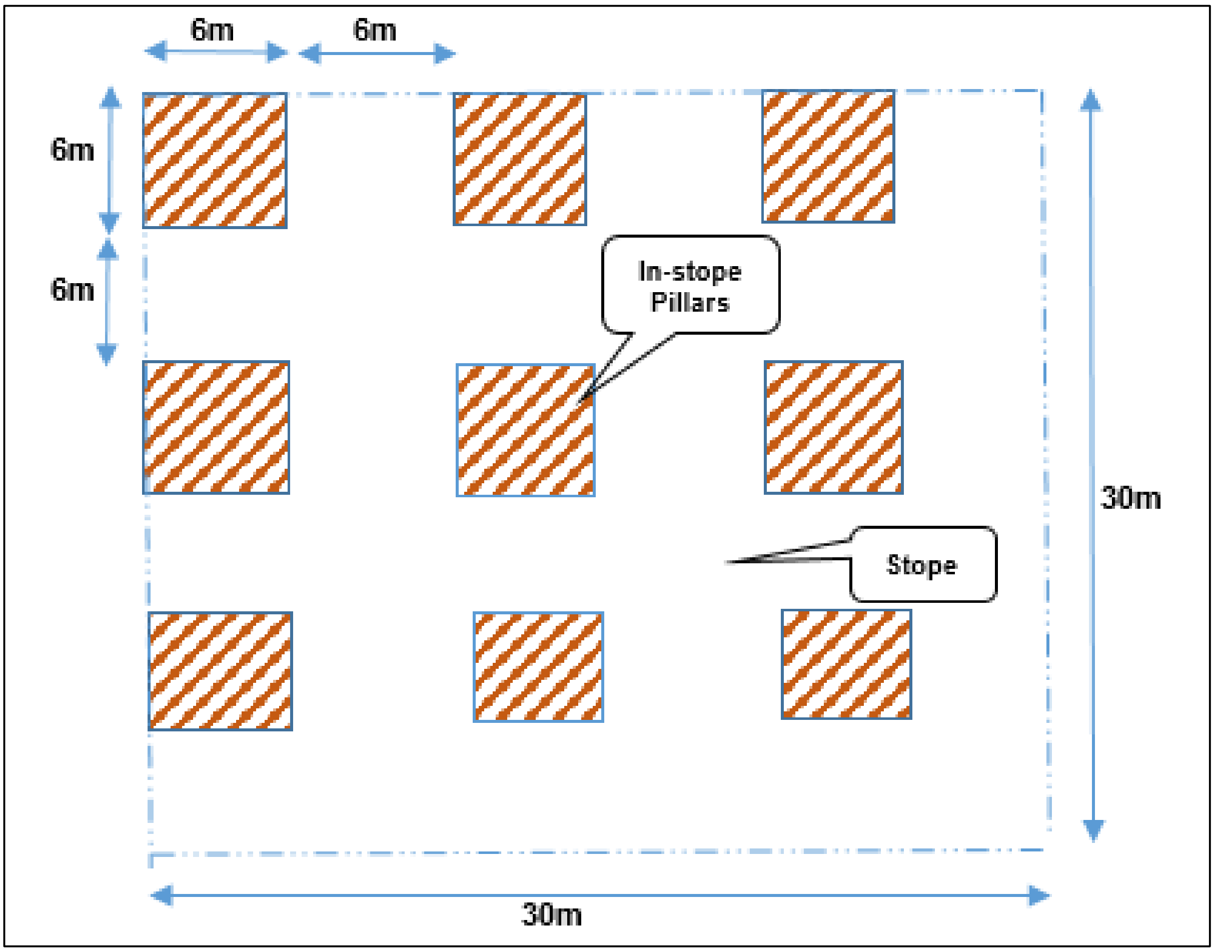


Figure 4.14: Extraction ratio calculation for 'steeps' stoping blocks

From Figure 4.14, total pillar area and total block area are given by Equations 4.1 and 4.2, respectively.

Total pillar area = Pillar length x Pillar width x No. of pillars (4.1)

= 6m x 6m x 9 = 324m^2

Total block area = Block width x Block length (4.2)

= 36m x 36m

= 1 296m^2

Therefore, the extraction ratio is given by Equation 4.3:

$$\text{Extraction ratio} = \frac{(Total\ block\ area - Total\ pillar\ area)}{Total\ block\ area} \quad (4.3)$$

$$= \frac{(1\ 296 - 324)m^2}{1\ 296m^2}$$

=75%

As the mining height decreases from 2.50m, the width to height ratio decreases if the pillar width is kept constant. Therefore, to maintain the width to height ratio, the pillar dimensions must be reduced as the mining cut decreases. Table 4.2 shows the pillar dimensions for the different mining cut options and the width to height ratios.

Table 4.2: 'Steeps' design pillar analysis

Mining cut (m)	Pillar dimensions (m)	Width to height ratio
2.50	6.00 x 6.00	2.4
2.40	5.76 x 5.76	2.4
2.30	5.52 x 5.52	2.4
2.20	5.28 x 5.28	2.4
2.10	5.04 x 5.04	2.4
2.00	4.80 x 4.80	2.4

As the mining cut and the pillar dimensions decrease, the extraction ratios for the 'steeps' design with 6.00m mining panel width for the six mining cut options increase. Using Equations 4.1 to 4.3, the extraction ratios calculated for the different mining cuts under study are shown in Figure 4.15.

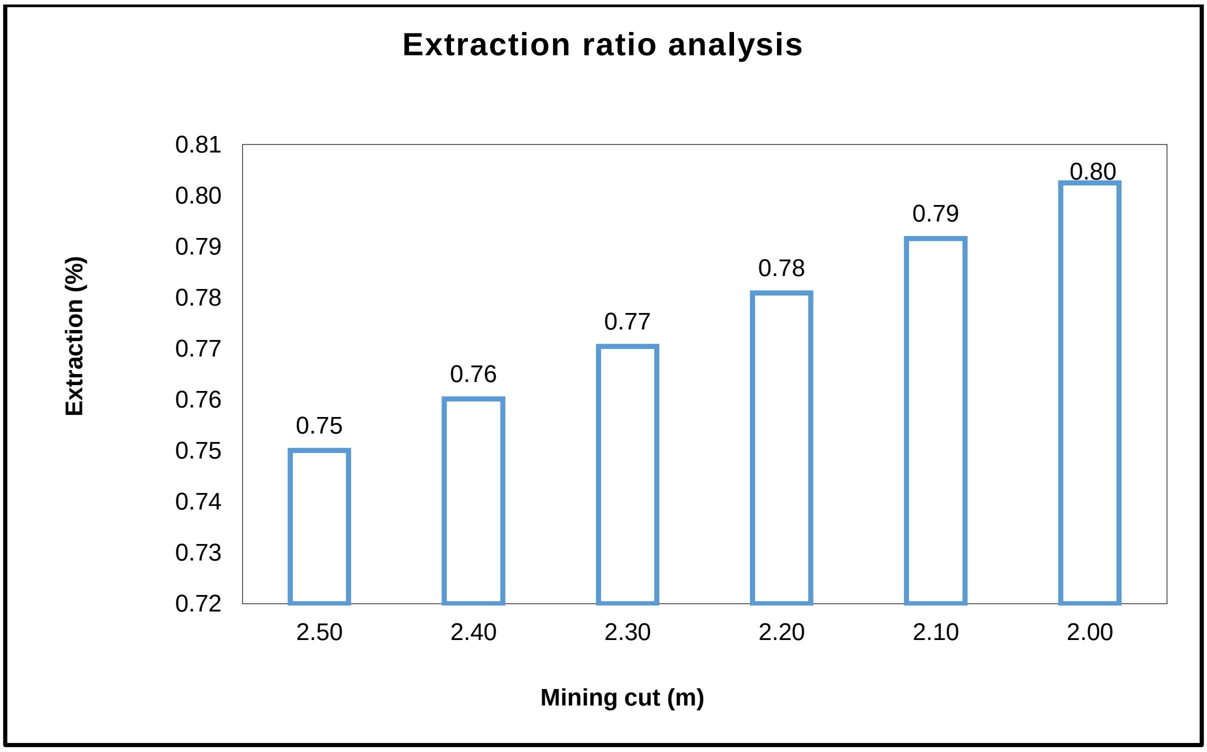


Figure 4.15: Extraction ratio analysis

Figure 4.15 shows that the extraction ratio increases with a reduction in mining cut. This is due to the reduction in pillar dimensions with reduction in height and a constant pillar width to height ratio of 2.4. The trend is also due to a constant stoping panel width of 6m. This is the minimum mining width for the Zimplats room and pillar mining method (Zimplats, 2017). Most of the equipment used at Zimplats require 6.00m as the minimum safe mining panel width. Based on extraction percentage, the 2.00m mining cut would be the optimum cut because it maximises Mineral Reserve recovery and increases metal content. The production schedule analysed in Section 4.5 will have different implications to other economic parameters including revenue and cost.

4.7 Mining revenue analysis

Table 4.3 shows the tonnage, the metal content and the revenue generated over the LOM from the different mining cuts. The grade is weighted average (weighted against Mineral Reserve tonnage) and the following prices were used to calculate revenue: Pt $926/oz, Pd $ 1 053/oz, Au $ 1 303/oz and Rh $ 1 580/oz. A smelter recovery of 98% was used for all the metals and concentrator recovery of 82%, 87%, 74% and 60% was used for Pt, Pd, Rh and Au, respectively. Payability was fixed at 90%.

Table 4.3: Revenue analysis for the six mining cut options

	Mined tonnes	Pt	Pd	Au	Rh	Total revenue
2.50m cut	7 006 733					
Grade (g/t)		1.628	1.411	0.145	0.198	
Metal content (Oz)		295 080	271 372	19 093	32 471	
Revenue ($m)		239.40	249.82	44.85	21.74	**555.82**
2.40m cut	6 837 848					
Grade (g/t)		1.667	1.434	0.148	0.205	
Metal content (Oz)		294 902	269 027	18 949	32 675	
Revenue ($m)		239.26	247.66	45.13	21.58	**553.63**
2.30m cut	6 639 978					
Grade (g/t)		1.710	1.456	0.150	0.212	
Metal content (Oz)		293 732	265 286	18 715	32 819	
Revenue ($m)		238.31	244.22	21.31	45.33	**549.17**
2.20m cut	6 419 150					
Grade (g/t)		1.756	1.477	0.153	0.220	
Metal content (Oz)		291 563	260 240	18 397	32 897	
Revenue ($m)		236.55	239.57	45.44	20.95	**542.51**
2.10m cut	6 179 196					
Grade (g/t)		1.805	1.498	0.155	0.228	
Metal content (Oz)		288 498	254 023	18 007	32 911	
Revenue ($m)		234.06	233.85	20.50	45.46	**533.88**
2.00m cut	5 923 216					
Grade (g/t)		1.847	1.509	0.157	0.236	
Metal content (Oz)		282 966	245 288	17 450	32 680	
Revenue ($m)		229.57	225.81	45.14	19.87	**520.39**

Table 4.3 shows that as the mining cut decreases from 2.50m to 2.00m, the tonnage mined decreases, average grade increases, metal content drops as seen in previous analyses. This result in the total revenue per mining cut decreasing. The drop in metal

content is due to the decrease in tonnage and this negatively affects the revenue generated. Therefore, based on revenue generated, 2.50m is the optimum mining cut.

4.8 Mining costs analysis

Mining costs were analysed for each mining cut option and Table 4.4 summarises the results of the analysis.

Table 4.4: Mining costs analysis for different mining cut options

	$/t	2.50m cut	2.40m cut	2.30m cut	2.20m cut	2.10m cut	2.00m cut
Mining - fixed costs ($m).		60.55	59.09	57.38	55.47	53.40	51.18
Mining - variable costs ($m)	19.16	134.23	131.00	127.21	122.97	118.38	113.47
Ngezi concentrator ($m)	10.00	12.59	12.59	12.59	10.79	10.79	10.79
Smelter ($m)	3.29	4.14	4.14	4.14	3.55	3.55	3.55
Transport costs - ore ($m)	1.89	2.38	2.38	2.38	2.04	2.04	2.04
Transport costs - smelter matte ($m)	152.83	0.51	0.51	0.51	0.43	0.43	0.43
Mining shared services ($m)	1.65	2.08	2.08	2.08	1.78	1.78	1.78
Processing shared services ($m)	1.72	2.17	2.17	2.17	1.86	1.86	1.86
Total operating costs ($m)		**218.64**	**213.95**	**208.45**	**198.90**	**192.23**	**185.11**

Table 4.4 shows fixed costs and variable costs for the different mining cuts. Total costs are the total costs for the project duration. The variable costs are tonnage driven and they decrease with the decrease in ore volume. The ore volume decreases as the mining cut decreases. The variable cost rates used in this analysis were derived from the Zimplats financial model. The rates were calculated or determined from Zimplats's past performance. Mining variable costs constitute the biggest component of the total operating costs. The major mining costs drivers which change as the mining cut

changes are, drilling, blasting, ore loading and hauling. The drilling costs increase because the number of holes drilled increase as the mining cut increase. Blasting costs increase when the mining cut increases, as more explosives are required to charge more drilled holes for the increased cut. Figures 4.16 to 4.21 shows drilling design for the 2.50m, 2.40m, 2.30m, 2.20m, 2.10m and 2.00m mining cuts. All the designs use the same type of explosives.

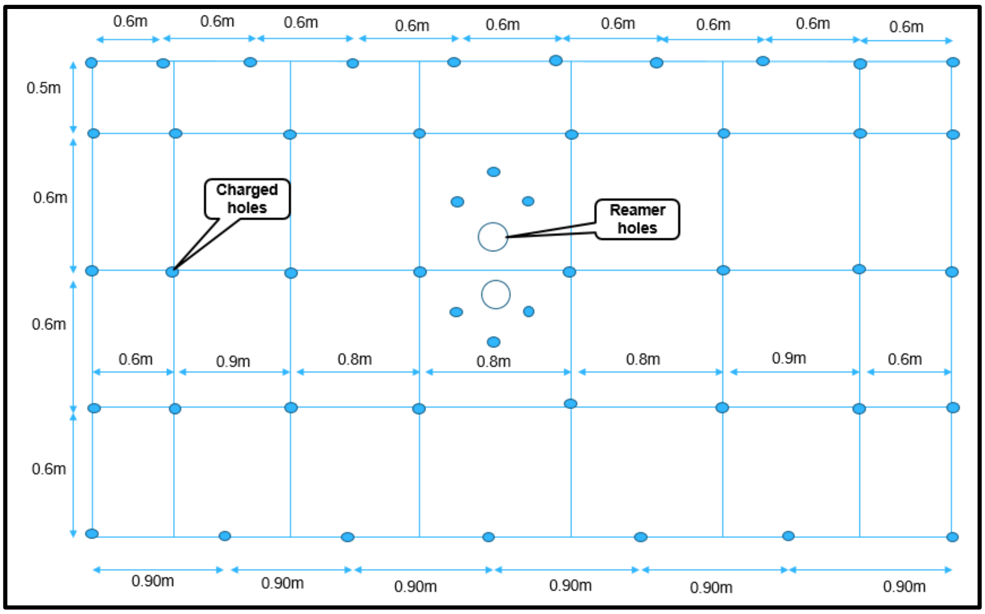


Figure 4.16: 2.50m mining cut drilling design

For the 2.50m mining cut, there are five rows of holes from the footwall to the hanging wall. The last row of holes in the hanging wall are the hanging wall perimeter holes, and the last row in the footwall are the footwall lifter holes. The rows are spaced vertically at 0.60m and the hanging wall perimeter holes are at 0.5m from the next row of drill holes. The total number of holes drilled in this design is 49 of which two holes are reamed to create an initial free face for the blast.

Drilling design for the 2.40m mining cut is similar to the drilling design for the 2.50m mining cut with a total of 49 holes. The only difference is the reduced burden (0.40m) between the hanging wall perimeter holes and the next row of drill holes. The impact of that difference is reduced tonnage for the 2.40m mining cut compared to the 2.50m mining cut.

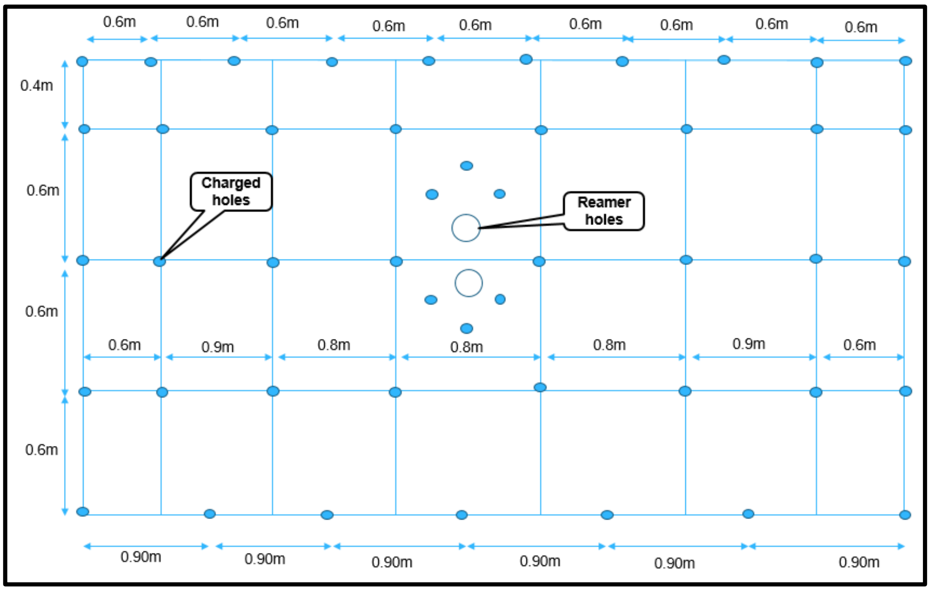


Figure 4.17: 2.40m mining cut drilling design

The 2.30m mining cut comes with a similar design to the 2.50m and 2.40m mining cuts. The drilling design however produces less tonnage than the 2.50 and 2.40m mining cuts due to further reduction in burden (0.30m) of the perimeter holes as shown in Figure 4.18.

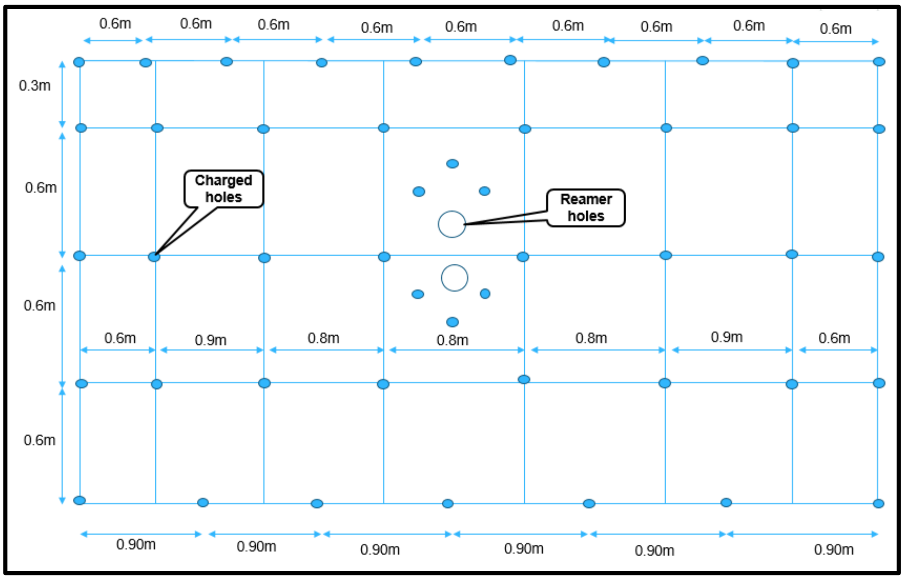


Figure 4.18: 2.30m mining cut drilling design

In the drilling design for the 2.20m mining cut the hanging wall perimeter holes burden is 0.20m. This design, though it produces smooth hanging wall conditions, it produces less tonnage than the 2.50m, 2.40m and the 2.30m. All the other drilling parameters remain the same.

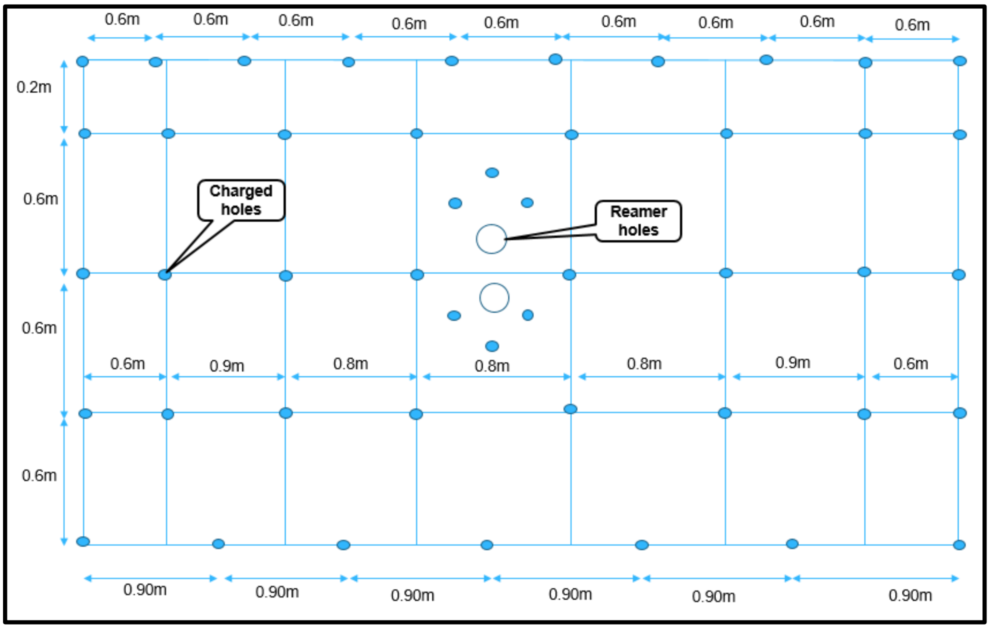


Figure 4.19: 2.20m mining cut drilling design

Figure 4.20 shows that the drilling design for the 2.10m mining cut is different from the designs of the 2.50m, 2.40, 2.30, and 2.20m mining cuts. The major difference is that the 2.10m mining cut has 4 rows of drill holes. The panel width remains 6.00m for all the mining cuts analysed in this study. This design has a total of 41 holes where two holes are reamed to create the initial free face for the blast. The reduced mining cut means it produces less tonnage than the 2.50m, 2.40m, 2.30m and 2.20m mining cut drilling designs.

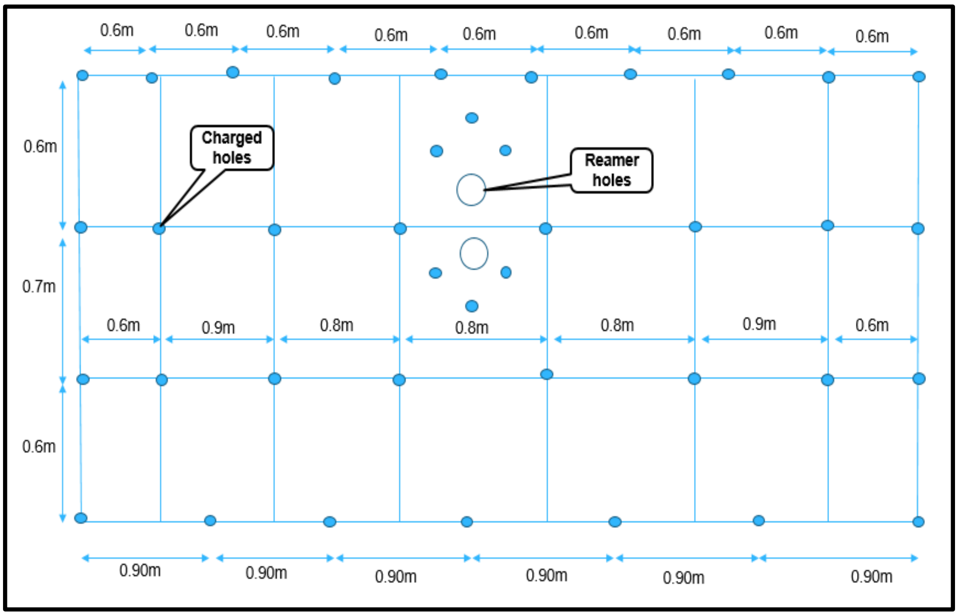


Figure 4.20: 2.10m mining cut drilling design

Drilling design for the 2.00m mining cut has the same number of holes as the 2.10m cut with the only difference between the two designs being the burden between the two middle rows of drill holes.

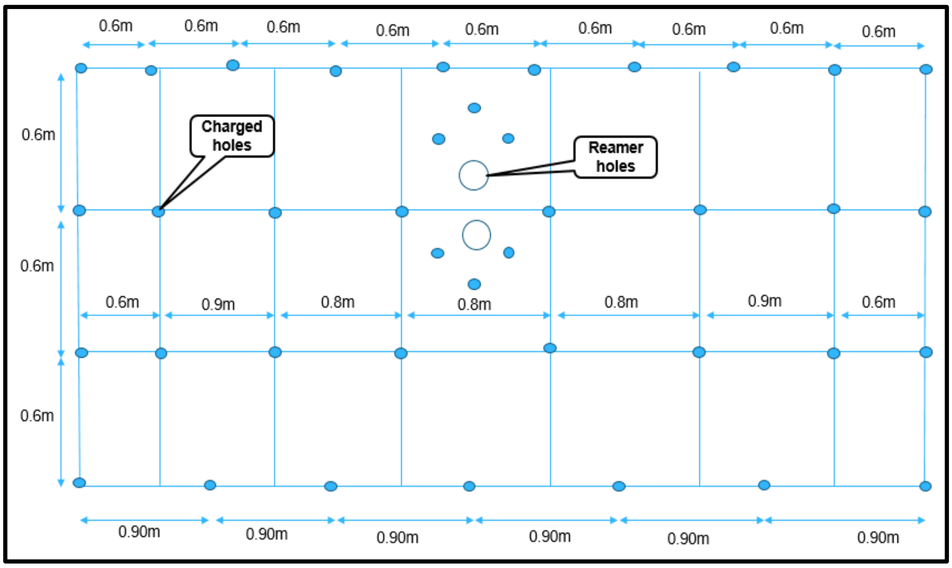


Figure 4.21: 2.00m mining cut drilling design

Table 4.5 shows the analyses of the drilling and blasting parameters for the six mining cut drilling designs. This analysis is followed by cost calculations for the six drilling and blasting designs.

Table 4.5: Drilling and blasting cost analysis for the selected mining cuts (Zimplats, 2017)

	Unit	2.50m cut	2.40m cut	2.30m cut	2.20m cut	2.10m cut	2.00m cut
Tunnel width (hole-hole)	m	6	6	6	6	6	6
Tunnel height (hole-hole)	m	2.5	2.5	2.5	2.5	2.5	2.5
Hole depth	m	3.2	3.2	3.2	3.2	3.2	3.2
Number of reamed holes		2	2	2	2	2	2
Total number of holes drilled		49	49	49	49	49	49
Number of perimeter holes		16	16	16	16	16	16
Number of charged holes		2	2	2	2	2	2
Number of uncharged holes		47	47	47	47	47	47
Design kg/hole emulsion	kg	4.5	4.5	4.5	4.5	4.5	4.5
Design kg/perimeter hole emulsion	kg	1.4	1.4	1.4	1.4	1.4	1.4
Quantity of emulsion/face	kg	161.9	161.9	161.9	161.9	132.1	132.1
Primer cartrudges/round		47	47	47	47	39	39
Mass of primer cartridge	kg	0.20	0.20	0.20	0.20	0.20	0.20
Total mass of cartridges per round	kg	9.31	9.31	9.31	9.31	7.72	7.72
Total mass of explosives per round	kg	171.21	171.21	171.21	171.21	139.82	139.82
Emulsion density	g/cc	1.13	1.13	1.13	1.13	1.13	1.13
Target advance	m	3	3	3	3	3	3
Tonnage blasted	t	145.8	139.968	134.136	128.304	122.472	116.64
Powder factor	kg/t	1.17	1.22	1.28	1.33	1.14	1.20
Costs	**Unit**	**2.50m cut**	**2.40m cut**	**2.30m cut**	**2.20m cut**	**2.10m cut**	**2.00m cut**
Magnum buster gel (32*200)	$0.23/cartridge	10.81	10.81	10.81	10.81	8.97	8.97
3.5m multi LPD ST	$0.86/unit	40.42	40.42	40.42	40.42	33.54	33.54
UG 100 emulsion	$0.69/kg	111.71	111.71	111.71	111.71	91.15	91.15
Total cost	$	162.94	162.94	162.94	162.94	133.66	133.66
Total cost per tonne blasted	$/t	1.12	1.16	1.21	1.27	1.09	1.15
Number of holes drilled		49	49	49	49	41	41
Meters drilled per end @3.2 hole length	m	156.8	156.8	156.8	156.8	131.2	131.2
Cost of drilling @ $1.43/m	$	224.224	224.224	224.224	224.224	187.616	187.616
Drilling cost per tonne	$/t	1.54	1.6	1.67	1.75	1.53	1.61
Total cost	$	387.17	387.17	387.17	387.17	321.28	321.28
Total cost per tonne drilled and blasted	**$/t**	**2.66**	**2.77**	**2.89**	**3.02**	**2.62**	**2.75**

Table 4.5 shows that in terms of absolute cost, the drilling design for the 2.50, 2.40m, 2.30m and 2.20m mining cuts have higher cost compared to the drilling design for the 2.10m and 2.00m mining cuts. This higher cost is due to the increase in number of holes drilled for the different designs, from 41 to 49 and number of holes charged from 39 to 47. However, in terms of cost per tonne mined, 2.10m is the optimum mining cut with the least cost of $2.62/tonne compared to the other five mining cuts in this analyses. Loading and hauling costs will increase with increasing mining cut as loading and hauling equipment handle more tonnage produced from the increased cut. Figure

4.22 shows the graphical presentation of the costs from Table 4.4 and the revenue generated for different mining cut options as shown in Table 4.3.

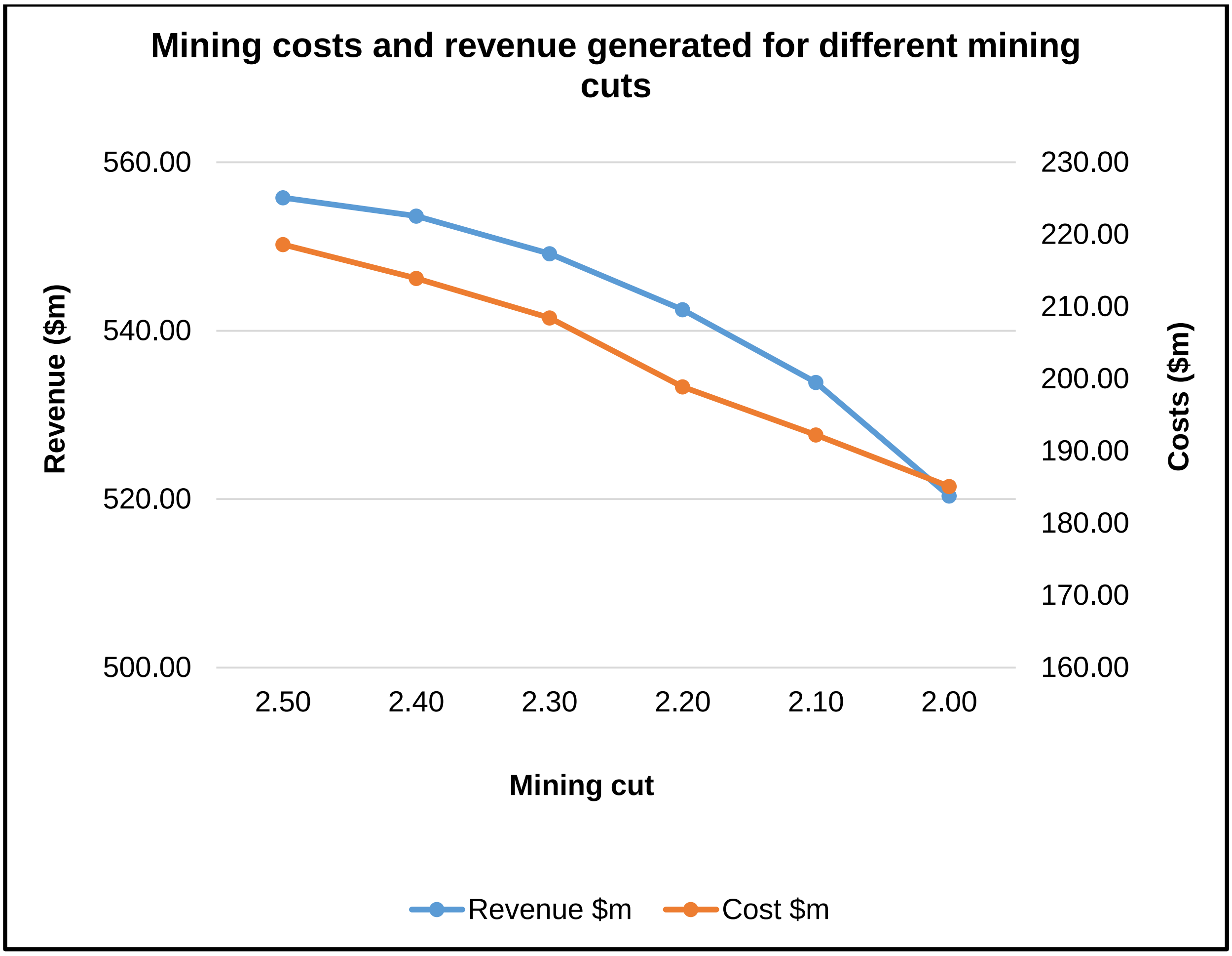


Figure 4.22: Mining costs and revenue generated for different mining cuts

Figure 4.22 shows that both the costs and revenue follow a similar decreasing trend as the mining cut decreases. Revenue decreases as less metal content will be produced and costs decrease due to handling of less ore volume as the tonnage decreases as the mining cut is decreased from 2.50m to 2.00m. The trends also show that the revenue generated is higher than the costs with the gap narrowing from 2.10m to 2.00m. This is due to the revenue which decrease at a higher rate than the decrease in costs as the mining cut decreases from 2.10m to 2.00m and the two graphs cross each other between 2.10m and 2.00m mining cuts, the point beyond which costs exceed revenue. This revenue trend is largely driven by the decreasing metal content as the mined tonnage decrease with the decreasing mining cut. The biggest difference between costs and revenue is at 2.20m mining cut. Based on the analysis of the two trends the 2.20m mining cut option is the optimum cut as it maximises profit. The previous analyses in this chapter resulted in different optimum mining cuts depending with the optimisation criteria. As mentioned in Chapter 2, NPV was selected as the optimisation criteria for this study. Therefore, to consolidate the previous techno-economic evaluation results, a DCF analysis was done for the mining cuts. The advantage of the DCF analysis is that it combines all the technical and economic inputs to enable a better evaluation of the optimum mining cut.

4.9 Discounted cash flow input parameters

In addition to the input parameter discussed in Section 4.7 for revenue and cost calculation, the input parameters in the next sub-section were used in the DCF analyses. Section 4.10 presents the results of the financial valuation for the six mining cut options under study. A financial model was constructed in Microsoft Excel 2016. The financial model incorporated production values for tonnage and grade from the mining schedules generated in Vulcan Gantt Scheduler shown in Appendix 7.3. It also incorporated input parameters briefly described in the next sub-sections and shown in detail in Appendix 7.5.

4.9.1 Metal recoveries (concentrator and smelter)

Concentrator recoveries applied in this analysis are different for each metal. Pt recovery is 82%, Pd is 87%, Rh is 74% and Au is 60%. Smelter recovery is 98% for all the four minerals (Zimplats, 2017).

4.9.2 Payability

It is assumed that all the minerals will not be sold for the full price and payability for all the PGMs is 90% in this financial model (Zimplats, 2017).

4.9.3 Metal prices

Metal prices used in the calculation of revenue in this financial model were obtained from the Zimplats' website. These are $928/oz for Pt, $1 053/oz for Pd, $1 580/oz for Rh and $1 303/oz for Au (Zimplats, 2017).

4.9.4 Operating costs

These costs are based on the average costs achieved by Zimplats during steady state production. These costs include mining, processing and services costs with the overall unit cost of $37.71/t (Zimplats, 2017).

4.9.5 The tax rates

The taxes considered in this financial model are income tax and royalty rates as stipulated by the government of Zimbabwe. Royalty paid for Pt, Pd, Au and Rh is 2.5% of the revenue generated from each mineral and income tax paid is 28% of the total taxable revenue amount (Zimplats, 2017).

4.9.6 Real discount rate

A real discount rate of 12.3% was used which is the rate Zimplats (2017) use for its projects valuation.

4.9.7 Production schedules for the four design options

Ore grades and tonnages in the production schedules from Vulcan Gantt Scheduler shown in Appendix 7.3 were used in the financial model to calculate the costs and revenue generated during the project life.

4.10 NPV versus mining cut

Figure 4.23 shows the summarised NPV results for the six mining cut options. Detailed cash flows for the six design options are shown in Appendix 7.6.

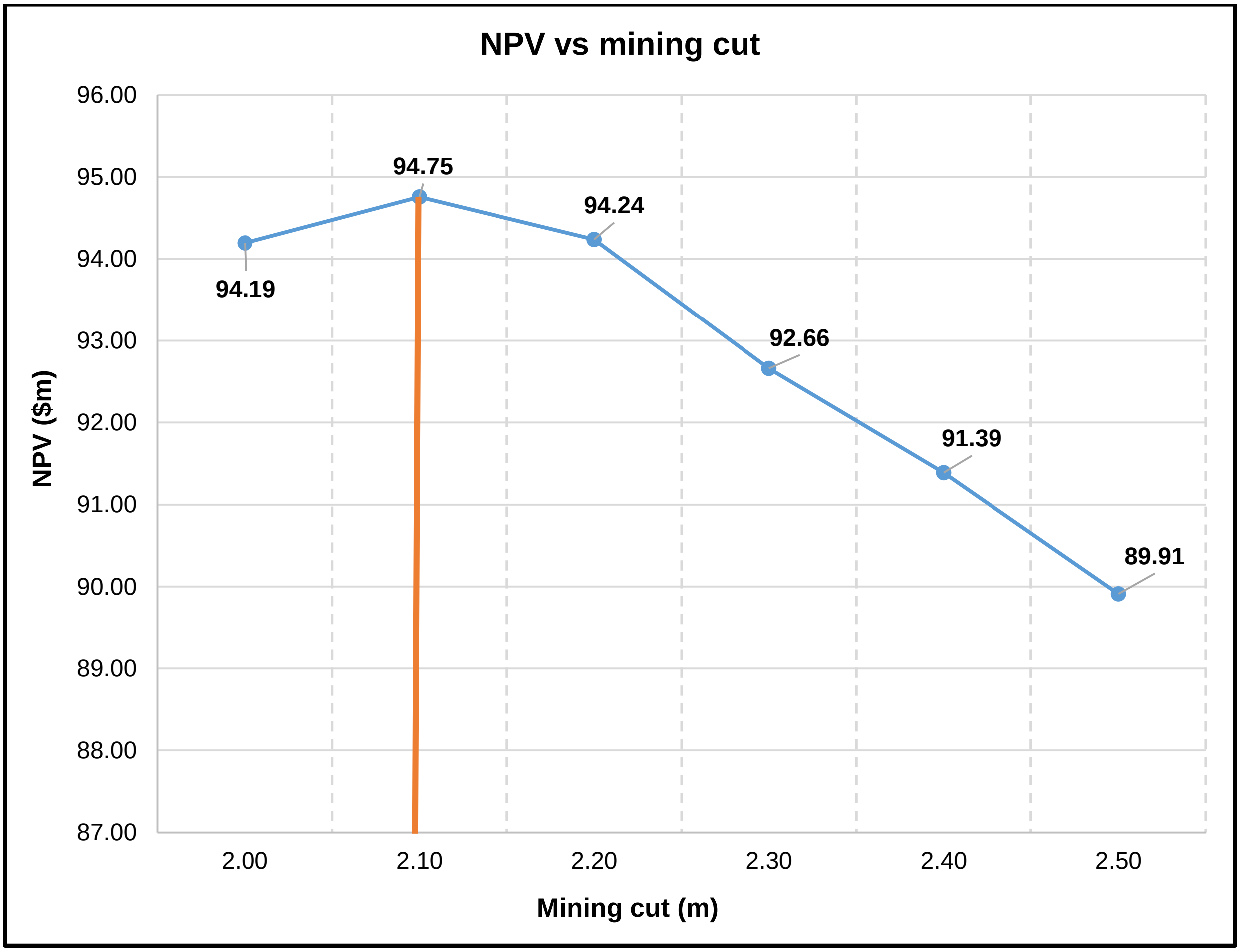


Figure 4.23: NPV versus mining cut

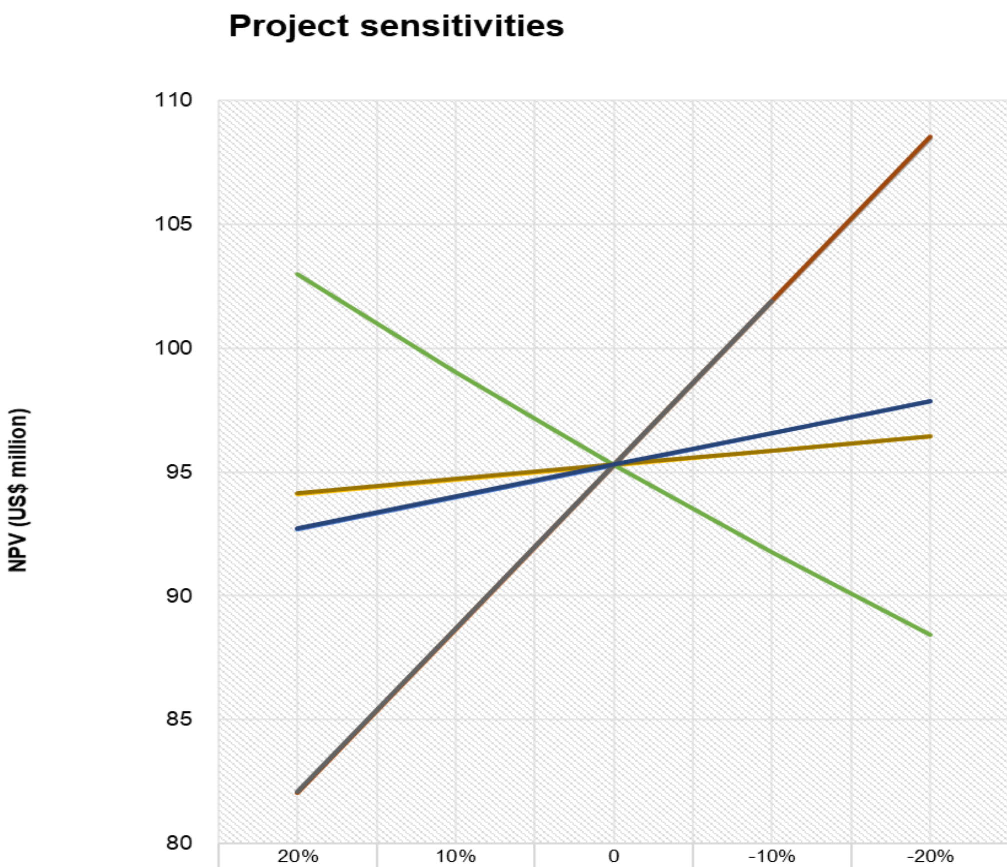
Figure 4.23 shows that NPV increases with increasing mining cut and reaches a maximum point at 2.10m then start to decrease. The initial increase in NPV as cut increases from 2.00m to 2.10m is due to the increase in metal content as the tonnage increases with increasing mining cut. However, the NPV does not continue to increase as the mining cut increases due to decreasing PGM grades as discussed in previous sections of this Chapter. The NPV starts to drop after the 2.10m mining cut option to 2.50m mining cut, and this is attributed to less revenue generation due to decreasing grades and increasing costs due to handling of more ore volumes with depressed grades. Results from the cash flow analysis show that the 2.10m mining cut has the highest NPV at $94.75m for the mining cut options in the range 2.00m to 2.50m. From these results, it shows that any movement along the mining cut scale from 2.10m either going up or down results in reduction in project value. The following section test the robustness of this design option by conducting a sensitivity analysis.

4.11 Sensitivity analysis of the 2.10m mining cut

The NPV for the 2.10 m mining cut option was evaluated for sensitivity to changes in a number of key variables. The following sensitivity analysis was conducted:

1. NPV at 12.3% real discount rate relative to changes in Pt price;
2. NPV at 12.3% real discount rate relative to changes in Pd price;
3. NPV at 12.3% real discount rate relative to changes in Au price;
4. NPV at 12.3% real discount rate relative to changes in Rh price;
5. NPV relative to changes in the real discount rate;
6. NPV at 12.3% real discount rate relative to changes in concentrator Pt recovery;
7. NPV at 12.3% real discount rate relative to changes in concentrator Pd recovery;
8. NPV at 12.3% real discount rate relative to changes in concentrator Au recovery; and
9. NPV at 12.3% real discount rate relative to changes in concentrator Rh recovery.

The key variables were changed by 20%, 10%, -10% and -20% and the resulting NPVs were recorded. The results of the sensitivity analysis are shown in Figure 4.24. The detailed results of the sensitivity analysis are shown in Appendix 7.7.



	20%	10%	0	-10%	-20%
Platinum Price	82	89	95	102	108
Paladium price	82	89	95	102	108
Gold price	94	95	95	96	96
Rhodium price	93	94	95	97	98
Discount Rate	103	99	95	92	88
Recovery Pt (concentrator)	82	89	95	102	109
Recovery Pd (concentrator)	82	89	95	102	
Recovery Au (concentrator)	94	95	95	96	96
Recovery Rh (concentrator)	93	94	95	97	98

Figure 4.24: Project sensitivities

Figure 4.24 shows that the project value is more sensitive to Pt and Pd concentrator recoveries and prices. Any change in these variables significantly affect project value. The project is also more sensitive to the real discount rate which means that a change in the discount rate will result in a significant change to the project NPV. The 2.10m mining cut option is the optimum cut because, though it is sensitive to Pt and Pd recoveries and prices, it is robust enough to withstand any changes (negative or positive) to these key variables as no negative NPV was recorded in this analysis.

4.12 Chapter summary

In this chapter, the room and pillar mining method employed at Zimplats and the modified layout for the extraction of steeper parts of the Zimplats tabular orebody were discussed. Six mining cut options were selected and their positions in relation to the mineralised horizon were determined. This was achieved by analysing the grade profiles for Pt, Pd, Au and Rh for P4 mine. Mineral Reserve calculations were done for the six mining cut options and results were imported into Vulcan software. The chapter also discussed the schedules that were generated by Vulcan Gantt Scheduler for the six mining cuts. Extraction ratios, costs and revenue analyses for all the six mining cut options were also done in this chapter.

Considering different parameters, different results in terms of the optimum cut were found depending with the optimisation criteria as shown in Table 4.6.

Table 4.6 Optimisation criteria and optimum cut

Optimisation criteria	Optimum cut
1. Mineral Reserve tonnage	2.50m
2. Grade	2.00m
3. Extraction ratio	2.00m
4. Revenue	2.50m
5. Cost	2.00m

If Mineral Reserve tonnage was the objective, 2.50m would be the optimum cut. If grade was the objective, 2.00m would be the optimum cut. If extraction ratio was the objective, 2.00m would be the optimum cut. If revenue was the objective, 2.50m would be the optimum cut. If costs was the objective 2.00m would be the optimum cut. However, a reliable mining cut is obtained after integrating all the economic and technical parameters. This was done using the DCF analysis. The DCF analysis was done for each mining cut option using the Zimplats financial model and results from the scheduling done in Vulcan. The 2.10m mining cut was determined to be the best mining cut, this option had the highest NPV compared to the NPVs for the other mining cut options. A sensitivity analysis was conducted to determine the stability of this option and it was found to be stable under changing key parameters such as metal prices and processing recoveries. The next chapter provides conclusions and recommendations formulated from the findings of the study.

5 CONCLUSIONS AND RECOMMENDATIONS

5.1 Chapter overview

This chapter concludes the research study and provide recommendations. Observations made in the results analysis are discussed, research limitations are highlighted and recommendations are drawn in this chapter.

5.2 Research observations

Operability of TMMs is affected by orebody dip angles at Zimplats room and pillar mine. The optimum mining angle for the TMMs range between 1^0 and 9^0 and Zimplats has categorised this type of the orebody as the ‘flats’ orebody. For the dip angles ranging between 9^0 and 14^0, the orebody is categorised as the ‘steeps’ orebody. For optimum TMMs operation in the ‘steeps’ orebody area, an apparent dip was introduced through the generation of the ‘steeps’ mining layout whose mining cut was found to be not optimum according to analysis carried out in Chapter 4.

Grade increases with a reduction in mining cut. For the mining cut options evaluated, the 2.00m mining cut had the highest average 4E grade and the 2.50m mining cut had the lowest average 4E grade. As was observed on the PGMs grade profile for P4 mine, the PGM grades increase gradually from the footwall towards the peak PGM horizon. Therefore, the varying of mining cuts by raising the footwall elevation results in the improvement of the overall grade achieved.

Tonnage increases with an increase in mining cut. As the mining cut increases from 2.00m to 2.50m the blast output also increases resulting in more production volume.
The extraction ratio increases with a reduction in mining cut. This is due to the reduction in pillar dimensions with reduction in height and a constant pillar width to height ratio of 2.4 and constant stoping panel width of 6m.

NPV increases with increasing mining cut and reaches a maximum point at 2.10m then starts to decrease. Results from the cash flow analysis shows that the 2.10m mining cut has the highest NPV (US$94.75m) for the mining cut options in the range 2.00m to 2.50m.

If the techno-economic parameters namely, grade, tonnage, extraction ratio, cost and revenue are evaluated separately, true optimality is not attained. The optimum mining cut achieved will be different depending on the optimisation criteria. Results from the sensitivity analysis showed that the project is more sensitive to Pt and Pd prices. Pt and Pd milling recoveries are also steep indicating the project is also highly sensitive to these parameters.

5.3 Recommendation

The recommendations from the research study are that:

- Zimplats should consider adopting the 2.10m mining cut for the 'steeps' sections. The 2.10m mining cut produces the highest NPV as compared to the 2.00m, 2.20m, 2.30m, 2.40m and 2.50m mining cuts; and
- Zimplats should consider designing a mining method for the 'steeps' resources with dip angles greater than 14º for maximum Mineral Resource utilisation as 30% of its Mineral Resource base is in this category.

5.4 Recommendations for future research

Six mining cut design options were evaluated to determine the optimum mining cut for the mining of 'steeps' resources at Zimplats. Discussion of results for the evaluation was done in chapter four and that established the basis for the formulation of the following recommendations:

- The PGM mineralisation include major concentrations of Pt, Pd, Rh and Au, and minor concentrations of iridium (Ir), ruthenium (Ru) and osmium (Os). This study only focused on the minerals with major concentrations, that is, Pt, Pd, Au and Rh. Minerals which are not found in higher concentration were not included in the evaluation. The inclusion of these minerals ought to be considered to identify their effects on the results of the evaluation of the different mining cut options;

- Studies must be carried out to determine TMM productivities in a 2.10m mining cut. In order to achieve overall optimisation of 'steeps' mining, productivities must be evaluated to determine TMM efficiencies in the new 'steeps' layout using the selected optimised mining cut. If there are any areas that need optimisation, they must also be addressed before the final design is commissioned;
- Other key parameters must also be reviewed and changes must be made where necessary to achieve optimum overall production performance. These key parameters include ventilation and safety systems. With a change in mining cut, there are changes in volumes of air flow circulating in the section, changes to loading bay designs, changes in services equipping in development ends. These parameters must be modified to conform to the mining cut design; and
- Studies can also be done to change stope profile configuration to minimise ore dilution and at the same time improving machine operability efficiencies.

6 REFERENCES

Abb Mining, 2017, Increased productivity with optimisation, Collaborative production management in mining. INTERNET. http://new.abb.com/mining/thought-leadership-topics/optimisation, [Accessed 28/10/2017].

Alford, C., Brazil, M. and David H. No date, Optimisation in underground mining, Bryan Mining Geology Research Centre, The University of Melbourne, Victoria, Australia.

Alford, C., Brazil, M. and Lee, D. 2007, Optimisation in underground mining, *Handbook of operations research in natural resources*. Vol. 99, pp. 561-577, Springer, USA.

Anglo American, 2018, Digging deeper: mining methods explained. INTERNET. https://www.angloamerican.com/futuresmart/our-industry/mining-explained/digging-deeper-mining-methods-explained, [Accessed 21/06/2018].

Anani, A. 2016, Applications of simulation and optimisation techniques in optimising room and pillar mining systems. PhD Dissertation, Missouri University of Science and Technology, United States of America. INTERNET. http://scholarsmine.mst.edu/doctoral_dissertations/2467, [Accessed 17/09/2017].

Beichl, I. and Sullivan, F. 2000, The Metropolis Algorithm, Computing in Science and Engineering Vol 2, Issue 65. INTERNET. http://hua-zhou.github.io/teaching/biostatm280-2017spring/readings/metropolis.pdf, [Accessed 30/05/2018].

Bhatti, M. A. 2000, Graphical optimisation*. In: Practical Optimisation Methods*. Springer, New York, USA.

Brazil, M., Lee, D., Rubinstein, J., Thomas, D., Weng, J. and Wormald, N., 2000, Network optimisation of underground mine design, *The Australasian Institute for Mining and Metallurgy,* Vol 305, Issue 1, pp. 57-65.

Burke, C., 2017, What is optimisation? INTERNET. https://sites.math.washington.edu/~burke/crs/515/notes/nt_1.pdf, [Accessed 28/10/2017].

Caccetta, L. 2007, Application of optimisation techniques in open pit mining. *Handbook of Operations Research in Natural Resources*, Vol 99, 2007, Springer, Boston, MA.

Chadwick, J. 2009, Mine optimisation*, International Mining Magazine, pp 18-28*. INTERNET. http://www.infomine.com/library/publications/docs/InternationalMining/Chadwick2009bb.pdf, [Accessed 06/02/2018].

Canadian Institute of Mining, 2003, Estimation of Mineral Resources and Mineral Reserves best practice Guidelines. INTERNET. http://web.cim.org/UserFiles/File/Estimation-Mineral-Resources-Mineral-Reserves-11-23-2003.pdf. [Accessed 10/12/2018].

David, 2017, Ore dilution & recovery in mining. INTERNET. https://www.911metallurgist.com/ore-dilution-recovery-mining/, [Accessed 01/10/2017].

Diering, D., Andersen, D., Langwieder, G. and Smith, G. 2012, The basic resource equations (BRE & BR2RE) – A new approach to the definition and reconciliation of mineral resources and reserves at Anglo American Platinum LTD, Journal of the *Southern African Institute of Mining and Metallurgy.*

Ding, B., Pelley, C. and Ruiter, J. 2004, Determining underground stope mineability using dynamic block value assignment approach, Taylor and Francis Group, London.

Don, S., Sameera, S., Topal, E., Asad, M. and Ali, W. 2015, Designing an optimal stope layout for underground mining based on a heuristic algorithm, *International Journal of Mining Science and Technology*, Vol 25, Issue 5, pp. 767-772.

Dowgielewicz, T., Ainia, B., Debicki, T. and Kawecki, T. 2006, Overview on optimisation criteria for production activities, *Institute of Logistics and Warehousing eBusiness Centre*, Poznan, Valencia.

Ebbels, A. 2016, Optimising underground mining strategies, SRK Consulting, Australia. INTERNET. https://www.srk.com/sites/default/files/file/AEbbels_OptimisingUndergroundMiningStrategies_2016_0.pdf, [Accessed 28/01/2018].

Ebrahimi, A. 2013, The importance of dilution factor for open pit mining projects, SRK Consulting, Vancouver, Canada. INTERNET. http://www.srk.com/files/File/papers/dilution_factor_openpit_a_ebrahimi.pdf, [Accessed 16/10/2017].

Ercikdi, B., Kesimal, A., Yilmaz, E. and Kaya, R. 2018, Parameters influencing ore dilution in underground mines, Department of Mining Engineering, Karadeniz Technical University, Trabzon, Turkey. INTERNET. https://www.scribd.com/document/240769491/Ore-dilution, [Accessed 28/01/2018].

Gelman, A., Carli, J. B., Stem, H. S., Dunson, D. B. and Vehtari, A. 2013, Bayesian data analysis, Third Edition, New York: Chapman and Hall/CRC.

George, M. and Richard, D. (No date), Design and optimisation in open pit mines, Kalulushi, Zambia.

Goodfellow, R. and Dimitrakopoulos, R. 2016, Global optimisation of open pit mining complexes with uncertainty, *Science Direct*, Vol 40, pp. 292-304.

Hem, P. 2012, Underground Mine Planning, Keevil Institute of Mining Engineering - University of British Columbia, INTERNET. http://technology.infomine.com/reviews/ugmineplanning/welcome.asp?view=full, [Accessed 02/02/2018].

Jorge, 2012, How to calculate apparent dip real dip. INTERNET. http://www.structuralgeology.org/2012/05/how-calculate-apparent-dip-real-dip.html, [Accessed 15/09/2017].

Kaapanda, L. 2011, An evaluation of factors determining the selection of mobile telecommunications service providers in the northern region of Namibia, MCom Dissertation, Regent Business School, South Africa.

Little, J., Knights, P. and Topal, E. 2012, Integrated optimization of underground mine design and scheduling, *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol 113, pp. 775-785.

Martinez, L. 2010, Strategic project evaluation for open pit mining ventures using real options and allied econometric techniques, MSc Dissertation, Queensland University of Technology Brisbane, Australia.

Minnit, R. 2004, Cut-off grade determination for the maximum value of a small Wits-type gold mining operation, *The Journal of the South African Institute of Mining and Metallurgy*, 2004.

Moharaj, M. C. 2014, Ore body modelling and comparison of different reserve estimation techniques, BTech Dissertation, Department of Mining Engineering National Institute of Technology, Rourkela, India. INTERNET. http://ethesis.nitrkl.ac.in/6193/1/E-31.pdf, [Accessed 26/01/2018].

Morin, M. A. 2001, Underground hardrock mine design and planning, PhD Thesis, Queen’s University, Kingston, Ontario.

Musingwini, C. 2016, Optimisation in underground mine planning–developments and opportunities, Presidential Address: *The Journal of the Southern African Institute of Mining and Metallurgy*, Vol 116, pp. 809-820.

Myburgh C., Deb K. and Craig S. 2014, Applying modern heuristics to maximising NPV through cut-off grage optimisation. *Orebody Modelling and Strategic Planning Conference*, Perth, Western Australia. INTERNET, http://www.maptek.com/pdf/insight/AUSIMM_Applying_Modern_Heuristics.pdf, [Accessed 20/02/2018].

Nelis, G., Gamache, M., Marcotte, D. and Bai, X. 2016, Stope optimisation with vertical convexity constraints, Springer Science Business Media, New York. INTERNET. http://repositorio.uchile.cl/bitstream/handle/2250/146492/Stope-optimisation.pdf?sequence=1, [Accessed 02/02/2018].

Nhleko, A., Tholana, T. and Neingo, P. 2018, A review of underground stope boundary optimisation algorithms, Resources Policy. Vol 56. Pp. 59-69. INTERNET. https://doi.org/10.1016/j.resourpol.2017.12.004, [Accessed12/12/2017].

Nieto, A. and Zhang, K. 2013, A cut-off grade economic strategy for a by-product mineral commodity operation: *Institute of Materials, Minerals and Mining and The AusIMM*, Vol 122(3), pp. 166-171.

Okubo, S. and Yamatomi, J. 2002, Underground mining methods and equipment*, Civil Engineering*, University of Tokyo, Japan.

Orelogy, 2017, Cut-off grade optimisation. INTERNET. http://www.orelogy.com/our-solutions/open-pit-scheduling-experts-2/cutoff-grade-optimisation/, [Accessed 29/09/2017].

Ovanic, J. and Young, D.S. 1995, Economic optimisation of stope geometry using seperable programming with special Branch and Bound techniques. *The Canadian Conference on Computer Applications in the Mineral Industry*: 129-135. Rotterdam: Balkema.

Pandey, S. 2018, Ore body modelling concepts and techniques. INTERNET. http://www.academia.edu/9018457/Ore_Body_Modelling-_Concepts_and_Techniques [Accessed 19/01/2018].

Pickering, R.G.B. 2004, The optimisation of mining method and equipment, Sandvik Mining and Construction, International Platinum Conference 'Platinum Adding Value', *The Journal of the South African Institute of Mining and Metallurgy*.

Pourrahimian, Y., Askar-Nasab, H., Tannant, D. 2018, Production scheduling with minimum mining width constraints using mathematical programming.

RBS Guide, 2010, A Guide to Dissertation Writing, Regent Business School, Durban, South Africa.

RWE Power International, 2008, Overcoming production deadlock at Zhungeer Coal Mine, Swindon Wiltshire, United Kingdom. INTERNET. http://www.rwe.com/web/cms/mediablob/en/217624/data/216542/1/rwe-technology-international/mining/mining-engineering/optimisation-of-mining-operations/Overcoming-production-deadlock-at-Zhungeer-coal-mine.pdf, [Accessed 30/01/2018].

SGS, 2018, Orebody modelling and resource estimation. INTERNET. http://www.sgs.com/en/mining/exploration-services/orebody-modeling-and-resource-estimation, [Accessed 28/01/2018].

Shahriar, K., Oraee, K. and Bakhtavar, E. 2007, A study on the optimisation algorithms for determining open pit and underground mining limits, Department of Mining and Metallurgy, Amirkabir University of Technology, Iran.

Sharma, A. 2009, *Operations Research*, Himalaya Publishing House.

Smit, A. and Lane, G. 2010, Mine optimisation and its application using the Anglo Platinum Mine Optimisation Tool (APMOT). *The Journal of the Southern African Institute of Mining and Metallurgy*. South Africa.

Smith, M. 2015, Mine plan optimisation. INTERNET. http://www.miningmath.com/download/cursos/Mine_Plan_Optimisation.pdf, [Accessed 02/02/2018].

Soyer, N. 2006, An approach on dilution and ore recovery/loss calculations in mineral reserve estimations at the Cayelli Mine, Turkey. PhD Thesis, The Graduate School of Natural and Applied Science of Middle East Technology University. INTERNET. http://citeseerx.ist.psu.edu/viewdoc/download?doi=10.1.1.633.5060&rep=rep1&type=pdf, [Accessed 19/10/2017].

Swain, B. 2016, Application of linear programming in mine systems optimisation, *National Institute of Technology*, Rourkela.

Taylor, B. No date, Integer programming: The branch and bound method. INTERNET. http://web.tecnico.ulisboa.pt/mcasquilho/compute/_linpro/TaylorB_module_c.pdf. [Accessed 01/06/2018].

The SAMREC Code, 2016, The South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves, 2016 Edition. South Africa.

Topal, E. and Sens, J. 2010, A new algorithm for stope boundary optimisation, *Journal of Coal Science and Engineering,* China, Vol 16, Issue 2, pp. 113–119.

Valicek, P., Fourie, F., Krafft, G. and Sevenoaks, J. 2012, Optimisation of mechanized mining layout within Anglo American Platinum, *The Journal of the Southern African Institute of Mining and Metallurgy,* South Africa.

Van Der Merwe, J. 1990, The extraction safety factor concept in high-extraction coal mining, *Journal of the South African Institute of Mining and Metallurgy*, Vol 90, Issue 11, 1990, pp. 57-65.

Wellmer, W., Dalheimer, M. and Wagner, M. 2008, Economic evaluations in exploration, Hardcover ISBN; 978-3-540-73557-1.

Zimplats, 2017, Operations review report, Internal document, Ngezi, Zimbabwe.

Zimplats, 2016, ‘Steeps’ trial report, Internal document, Ngezi, Zimbabwe.

Zimplats, 2012, Geology of the Great Dyke, Internal document, Ngezi, Zimbabwe.

7 APPENDICES

Appendix 7.1: Grade profile analysis for P4 mine (Zimplats, 2017)

Layer	Meters rom base	Pt ppm	Pd ppm	Au ppm	Rh ppm	4E
HM	3.25	0.01	0.01	0.05	0.00	0.072
HL	3.00	0.02	0.01	0.06	0.00	0.091
HK	2.75	0.03	0.01	0.08	0.00	0.129
HJ	2.50	0.05	0.02	0.09	0.00	0.161
HI	2.25	0.10	0.05	0.15	0.01	0.299
HH	2.00	0.11	0.06	0.20	0.01	0.378
HG	1.75	0.08	0.05	0.19	0.01	0.333
HF	1.50	0.10	0.06	0.30	0.01	0.464
HE	1.25	0.12	0.06	0.29	0.01	0.480
HD	1.00	0.12	0.05	0.17	0.01	0.346
HC	0.75	0.22	0.10	0.21	0.01	0.535
HB	0.50	0.63	0.30	0.38	0.03	1.338
HA	0.25	2.20	0.99	0.55	0.11	3.855
PK	**0.00**	**3.83**	**1.66**	**0.49**	**0.22**	**6.201**
FA	-0.25	2.80	1.43	0.27	0.19	4.686
FB	-0.50	2.22	1.51	0.18	0.18	4.087
FC	-0.75	2.17	1.82	0.19	0.19	4.382
FD	-1.00	2.00	2.19	0.13	0.21	4.540
FE	-1.25	1.37	2.04	0.07	0.18	3.656
FF	-1.50	1.00	1.70	0.04	0.15	2.895
FG	-1.75	0.87	1.52	0.02	0.14	2.551
FH	-2.00	0.85	1.42	0.02	0.15	2.442
FI	-2.25	0.73	1.17	0.02	0.15	2.061
FJ	-2.50	0.56	0.87	0.01	0.13	1.569
FK	-2.75	0.45	0.77	0.01	0.11	1.348
FL	-3.00	0.37	0.59	0.01	0.10	1.071
FM	-3.25	0.25	0.43	0.01	0.07	0.757
FN	-3.50	0.13	0.22	0.01	0.04	0.398
FO	-3.75	0.07	0.12	0.01	0.02	0.216
FP	-4.00	0.05	0.09	0.00	0.02	0.159
FQ	-4.25	0.03	0.07	0.00	0.01	0.113

Appendix 7.2: Mineral Reserve calculation results (Zimplats, 2017)

Appendix 7.2.1, 2.00m mining cut Mineral Reserves

SOURCE	REGION	PL	PD	AU	RH	4E	ROM
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_1.00t	1.84	1.82	0.19	0.15	4.01	194 784
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_10.00t	1.76	1.46	0.22	0.16	3.60	343 788
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_11.00t	1.91	1.36	0.29	0.15	3.71	413 204
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_12.00t	1.87	1.47	0.24	0.16	3.75	536 092
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_13.00t	1.94	1.39	0.28	0.15	3.77	456 018
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_14.00t	1.65	1.49	0.19	0.16	3.49	371 378
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_15.00t	1.60	1.53	0.17	0.16	3.46	386 242
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_16.00t	1.81	1.44	0.24	0.15	3.65	311 517
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_2.00t	1.94	1.66	0.25	0.15	4.00	403 442
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_3.00t	1.95	1.59	0.26	0.16	3.95	432 189
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_4.00t	1.94	1.57	0.25	0.16	3.93	436 206
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_5.00t	1.90	1.56	0.23	0.17	3.86	426 565
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_6.00t	1.82	1.54	0.22	0.16	3.74	378 016
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_7.00t	1.89	1.47	0.25	0.16	3.77	340 887
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_8.00t	1.89	1.39	0.27	0.15	3.70	249 959
DEC15_p4_isatis_reserves.bmf	wits.tri\2.00m.tri\block_9.00t	1.70	1.50	0.19	0.16	3.55	245 884
DEC15_p4_isatis_reserves.bmf	**Grand Total**	**1.85**	**1.51**	**0.24**	**0.16**	**3.75**	**5 923 216**

Appendix 7.2.2, 2.10m mining cut Mineral Reserves

SOURCE	REGION	PL	PD	AU	RH	4E	ROM
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_1.00t	1.80	1.81	0.18	0.15	3.94	201 710
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_10.00t	1.72	1.45	0.22	0.15	3.54	357 050
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_11.00t	1.87	1.35	0.28	0.15	3.65	440 669
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_12.00t	1.83	1.46	0.23	0.16	3.67	557 292
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_13.00t	1.89	1.38	0.27	0.15	3.69	483 384
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_14.00t	1.62	1.48	0.18	0.15	3.44	382 934
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_15.00t	1.57	1.51	0.17	0.16	3.41	396 655
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_16.00t	1.77	1.43	0.23	0.15	3.59	328 089
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_2.00t	1.88	1.65	0.23	0.15	3.91	422 200
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_3.00t	1.89	1.57	0.25	0.15	3.86	452 259
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_4.00t	1.88	1.56	0.24	0.16	3.84	455 313
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_5.00t	1.85	1.54	0.22	0.16	3.77	442 317
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_6.00t	1.77	1.52	0.21	0.16	3.66	390 749
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_7.00t	1.84	1.46	0.24	0.16	3.69	355 506
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_8.00t	1.85	1.38	0.26	0.15	3.65	263 157
DEC15_p4_isatis_reserves.bmf	wits.tri\2.10m.tri\block_9.00t	1.67	1.49	0.19	0.16	3.50	252 870
DEC15_p4_isatis_reserves.bmf	**Grand Total**	**1.80**	**1.50**	**0.23**	**0.16**	**3.69**	**6 179 196**

Appendix 7.2.3, 2.20m mining Mineral Reserves

SOURCE	REGION	PL	PD	AU	RH	4E	ROM
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_1.00t	1.75	1.79	0.17	0.15	3.86	208 112
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_10.00t	1.69	1.43	0.21	0.15	3.49	369 451
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_11.00t	1.83	1.34	0.27	0.15	3.59	467 700
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_12.00t	1.78	1.45	0.22	0.16	3.61	576 362
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_13.00t	1.84	1.37	0.26	0.15	3.62	509 623
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_14.00t	1.60	1.46	0.18	0.15	3.39	393 429
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_15.00t	1.55	1.50	0.16	0.15	3.36	406 108
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_16.00t	1.74	1.41	0.23	0.15	3.53	343 906
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_2.00t	1.82	1.62	0.22	0.15	3.81	440 449
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_3.00t	1.83	1.56	0.24	0.15	3.77	471 834
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_4.00t	1.83	1.54	0.23	0.16	3.75	473 111
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_5.00t	1.80	1.52	0.21	0.16	3.69	455 178
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_6.00t	1.73	1.50	0.21	0.16	3.59	402 289
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_7.00t	1.79	1.45	0.23	0.15	3.62	369 446
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_8.00t	1.82	1.38	0.25	0.15	3.59	275 956
DEC15_p4_isatis_reserves.bmf	wits.tri\2.20m.tri\block_9.00t	1.64	1.47	0.18	0.15	3.45	259 151
DEC15_p4_isatis_reserves.bmf	**Grand Total**	**1.76**	**1.48**	**0.22**	**0.15**	**3.61**	**6 419 150**

Appendix 7.2.4, 2.30m mining cut Mineral Reserves

SOURCE	REGION	PL	PD	AU	RH	4E	ROM
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_1.00t	1.69	1.74	0.17	0.15	3.75	214 023
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_10.00t	1.65	1.40	0.20	0.15	3.41	380 756
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_11.00t	1.77	1.32	0.26	0.14	3.49	493 878
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_12.00t	1.72	1.42	0.21	0.15	3.51	593 204
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_13.00t	1.78	1.35	0.25	0.15	3.52	534 299
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_14.00t	1.55	1.43	0.17	0.15	3.30	402 715
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_15.00t	1.51	1.47	0.16	0.15	3.28	414 679
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_16.00t	1.69	1.39	0.22	0.15	3.44	358 907
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_2.00t	1.74	1.58	0.21	0.14	3.68	457 659
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_3.00t	1.75	1.52	0.22	0.15	3.64	490 668
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_4.00t	1.76	1.50	0.22	0.15	3.63	488 819
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_5.00t	1.74	1.49	0.20	0.16	3.59	464 975
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_6.00t	1.67	1.47	0.20	0.15	3.49	412 784
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_7.00t	1.72	1.42	0.22	0.15	3.51	382 553
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_8.00t	1.76	1.36	0.24	0.15	3.50	288 314
DEC15_p4_isatis_reserves.bmf	wits.tri\2.30m.tri\block_9.00t	1.59	1.44	0.18	0.15	3.36	264 703
DEC15_p4_isatis_reserves.bmf	**Grand Total**	**1.71**	**1.46**	**0.21**	**0.15**	**3.53**	**6 639 978**

Appendix 7.2.5, 2.40m Mineral Reserves

SOURCE	REGION	PL	PD	AU	RH	4E	ROM
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_1.00t	1.65	1.72	0.16	0.14	3.67	219 410
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_10.00t	1.62	1.39	0.20	0.15	3.35	390 825
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_11.00t	1.74	1.31	0.25	0.14	3.43	518 222
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_12.00t	1.68	1.40	0.20	0.15	3.44	607 653
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_13.00t	1.73	1.34	0.24	0.15	3.45	557 198
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_14.00t	1.52	1.41	0.17	0.15	3.25	410 725
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_15.00t	1.48	1.45	0.15	0.15	3.23	422 409
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_16.00t	1.65	1.37	0.21	0.15	3.38	373 095
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_2.00t	1.69	1.56	0.20	0.14	3.59	473 188
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_3.00t	1.69	1.49	0.21	0.14	3.54	508 278
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_4.00t	1.71	1.47	0.21	0.15	3.54	502 114
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_5.00t	1.70	1.46	0.20	0.16	3.52	471 989
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_6.00t	1.63	1.44	0.19	0.15	3.42	422 027
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_7.00t	1.68	1.40	0.21	0.15	3.43	394 098
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_8.00t	1.72	1.35	0.23	0.14	3.44	300 088
DEC15_p4_isatis_reserves.bmf	wits.tri\2.40m.tri\block_9.00t	1.56	1.42	0.17	0.15	3.31	269 487
DEC15_p4_isatis_reserves.bmf	Grand Total	**1.67**	**1.43**	**0.20**	**0.15**	**3.45**	**6 837 848**

Appendix 7.2.6, 2.50m Mineral Reserves

SOURCE	REGION	PL	PD	AU	RH	4E	ROM
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_1.00t	1.59	1.67	0.15	0.14	3.56	224 193
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_10.00t	1.58	1.36	0.19	0.14	3.27	399 557
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_11.00t	1.69	1.28	0.24	0.14	3.35	538 191
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_12.00t	1.63	1.37	0.20	0.15	3.35	619 741
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_13.00t	1.68	1.31	0.23	0.14	3.36	577 964
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_14.00t	1.48	1.38	0.16	0.14	3.17	417 522
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_15.00t	1.44	1.42	0.15	0.15	3.15	429 198
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_16.00t	1.61	1.34	0.21	0.14	3.29	386 516
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_2.00t	1.62	1.51	0.19	0.14	3.47	486 106
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_3.00t	1.63	1.45	0.20	0.14	3.42	523 245
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_4.00t	1.65	1.44	0.20	0.15	3.43	513 054
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_5.00t	1.66	1.43	0.19	0.15	3.43	476 653
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_6.00t	1.58	1.41	0.18	0.15	3.32	429 377
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_7.00t	1.62	1.36	0.20	0.14	3.33	403 838
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_8.00t	1.66	1.32	0.22	0.14	3.35	311 111
DEC15_p4_isatis_reserves.bmf	wits.tri\2.50m.tri\block_9.00t	1.52	1.39	0.17	0.15	3.22	273 422
DEC15_p4_isatis_reserves.bmf	**Grand Total**	**1.63**	**1.41**	**0.20**	**0.15**	**3.38**	**7 006 733**

Appendix 7.3: Mining schedules
(Zimplats, 2017)

Appendix 7.3.1, 2.00m mining cut schedule

Title	Unit	2018	2019	2020	2021	2022	2023
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	527
Pl Grade	**g/t**	1.85	1.86	1.74	1.94	1.87	1.80
Pd Grade	**g/t**	1.49	1.44	1.53	1.59	1.53	1.44
Au Grade	**g/t**	0.24	0.25	0.21	0.25	0.23	0.23
Rh Grade	**g/t**	0.15	0.16	0.16	0.16	0.16	0.16
4e Grade	**g/t**	3.73	3.70	3.63	3.95	3.79	3.63

Appendix 7.3.2, 2.10m mining cut schedule

Title	Unit	2018	2019	2020	2021	2022	2023
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	783
Pl Grade	**g/t**	1.81	1.84	1.67	1.89	1.83	1.79
Pd Grade	**g/t**	1.48	1.43	1.49	1.59	1.53	1.45
Au Grade	**g/t**	0.24	0.24	0.20	0.24	0.22	0.23
Rh Grade	**g/t**	0.15	0.16	0.15	0.15	0.16	0.15
4e Grade	**g/t**	3.68	3.66	3.52	3.88	3.74	3.62

Appendix 7.3.3, 2.20m mining cut schedule

Title	Unit	2018	2019	2020	2021	2022	2023
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	1 023
Pl Grade	**g/t**	1.77	1.80	1.62	1.82	1.79	1.75
Pd Grade	**g/t**	1.46	1.41	1.46	1.57	1.52	1.44
Au Grade	**g/t**	0.23	0.24	0.19	0.23	0.22	0.22
Rh Grade	**g/t**	0.15	0.15	0.15	0.15	0.16	0.15
4e Grade	**g/t**	3.60	3.60	3.41	3.77	3.68	3.56

Appendix 7.3.4, 2.30m mining cut schedule

Title	Unit	2018	2019	2020	2021	2022	2023	2024
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	1 082	161
Pl Grade	**g/t**	1.72	1.76	1.58	1.75	1.75	1.72	1.60
Pd Grade	**g/t**	1.44	1.40	1.44	1.54	1.50	1.43	1.45
Au Grade	**g/t**	0.22	0.23	0.18	0.22	0.21	0.21	0.18
Rh Grade	**g/t**	0.15	0.15	0.15	0.15	0.16	0.15	0.15
4e Grade	**g/t**	3.53	3.53	3.35	3.65	3.62	3.51	3.38

Appendix 7.3.5, 2.40m mining cut schedule

Title	Unit	2018	2019	2020	2021	2022	2023	2024
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	1 082	359
Pl Grade	**g/t**	1.68	1.71	1.55	1.69	1.71	1.68	1.61
Pd Grade	**g/t**	1.42	1.38	1.42	1.50	1.48	1.42	1.41
Au Grade	**g/t**	0.21	0.22	0.18	0.21	0.21	0.21	0.19
Rh Grade	**g/t**	0.14	0.15	0.15	0.14	0.15	0.15	0.15
4e Grade	**g/t**	3.47	3.46	3.29	3.54	3.55	3.45	3.35

Appendix 7.3.6, 2.50m mining cut schedule

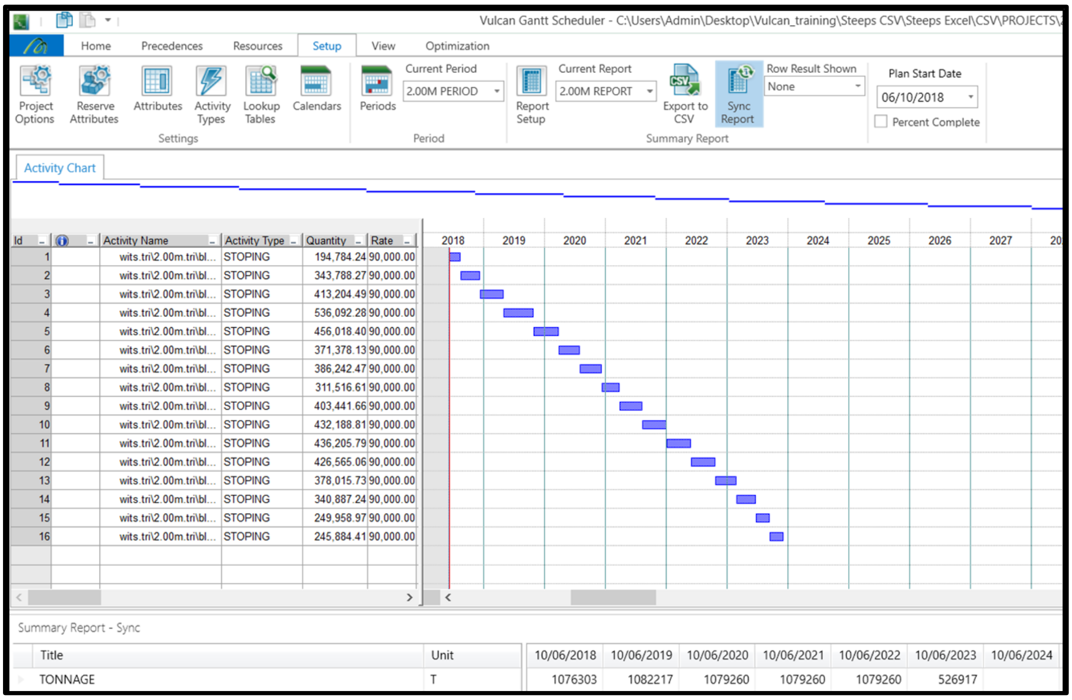
Title	Unit	2018	2019	2020	2021	2022	2023	2024
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	1 082	528
Pl Grade	**g/t**	1.65	1.67	1.52	1.64	1.66	1.63	1.61
Pd Grade	**g/t**	1.41	1.36	1.40	1.46	1.46	1.41	1.37
Au Grade	**g/t**	0.21	0.21	0.17	0.20	0.20	0.20	0.20
Rh Grade	**g/t**	0.14	0.15	0.15	0.14	0.15	0.15	0.15
4e Grade	**g/t**	3.41	3.39	3.24	3.44	3.47	3.38	3.32

Appendix 7.3.7, 2.08m mining cut schedule

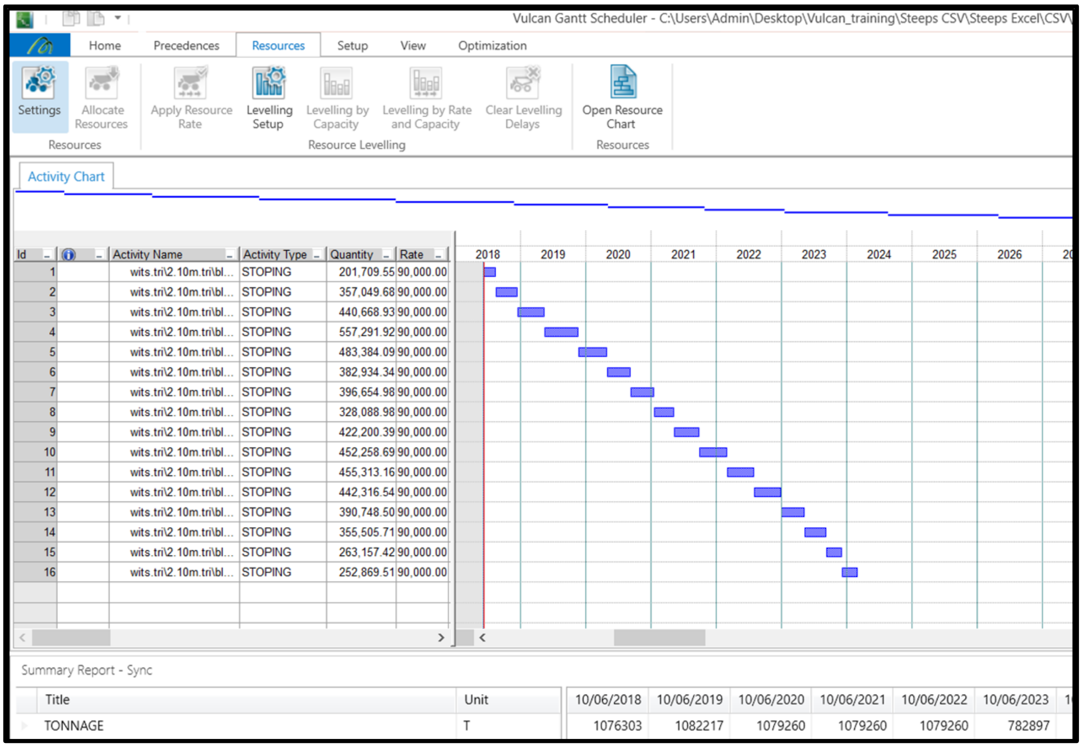
Title	Unit	2018	2019	2020	2021	2022	2023
Tonnage	**t '000'**	1 076	1 082	1 079	1 079	1 079	775
Pl Grade	**g/t**	1.82	1.84	1.68	1.89	1.83	1.80
Pd Grade	**g/t**	1.48	1.44	1.50	1.60	1.54	1.45
Au Grade	**g/t**	0.24	0.24	0.20	0.24	0.22	0.23
Rh Grade	**g/t**	0.15	0.16	0.16	0.16	0.16	0.16
4e Grade	**g/t**	3.69	3.68	3.53	3.89	3.76	3.64

Appendix 7.4: VGS mining schedule results

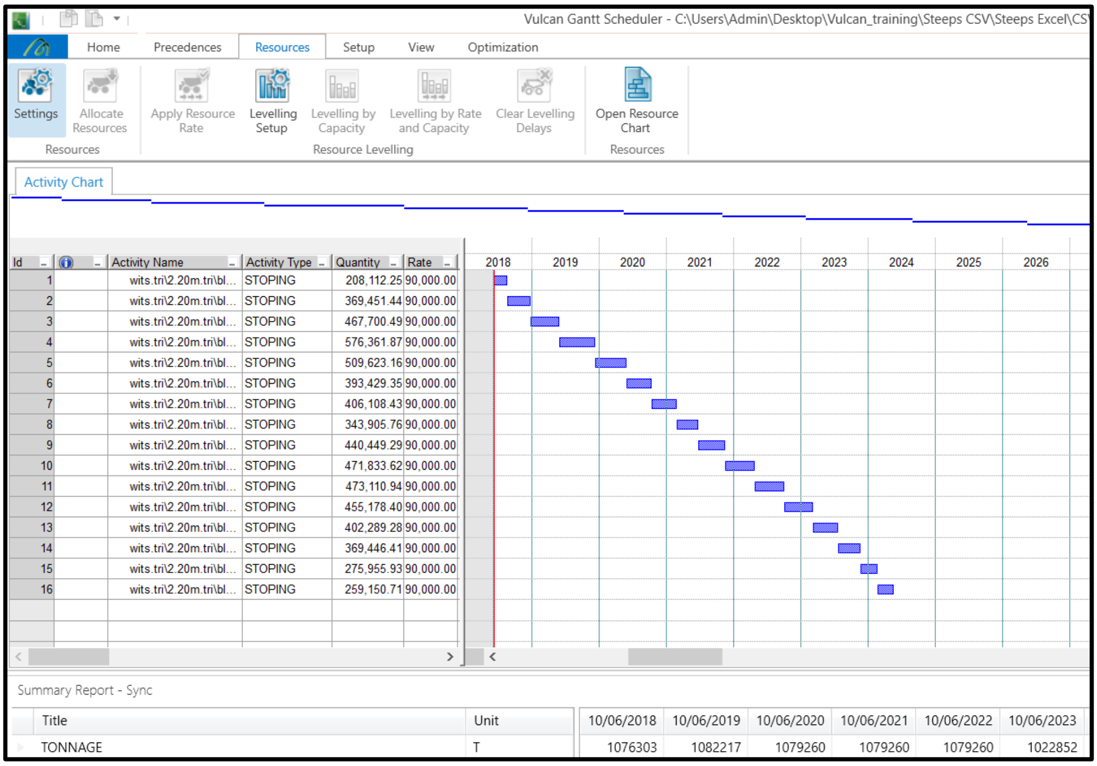
Appendix 7.4.1, 2.00m mining cut schedule



Appendix 7.4.2, 2.10m mining schedule



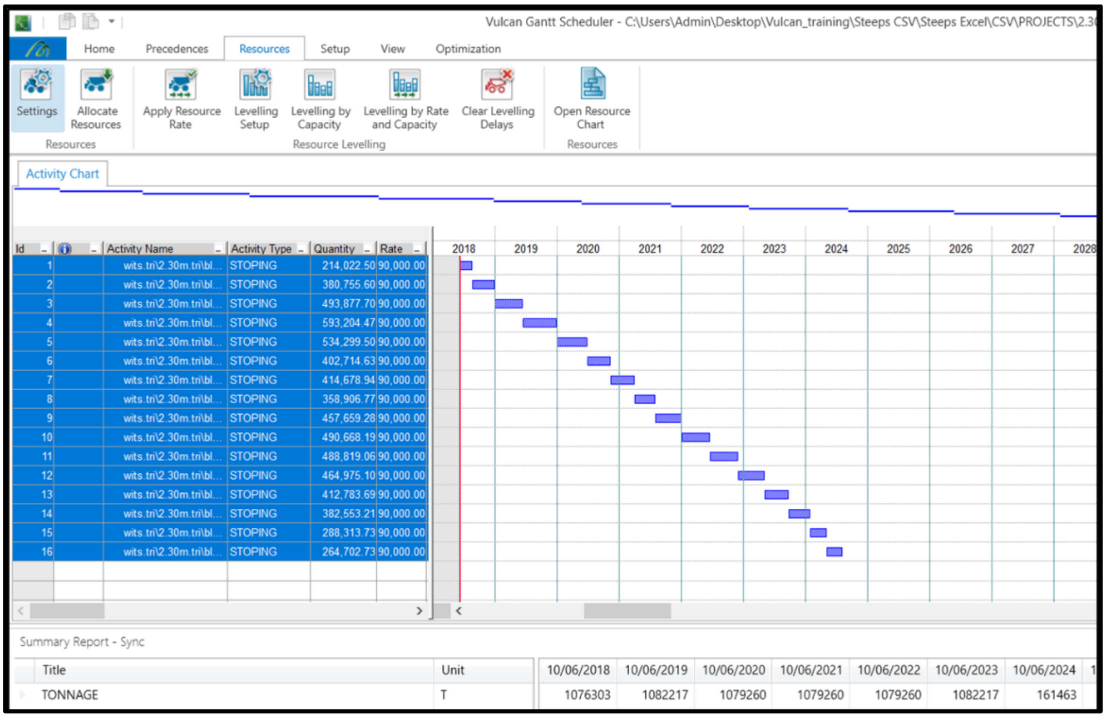
Appendix 7.4.3, 2.20m mining schedule



Id	Activity Name	Activity Type	Quantity	Rate
1	wits.tri\2.20m.tri\bl...	STOPING	208,112.25	90,000.00
2	wits.tri\2.20m.tri\bl...	STOPING	369,451.44	90,000.00
3	wits.tri\2.20m.tri\bl...	STOPING	467,700.49	90,000.00
4	wits.tri\2.20m.tri\bl...	STOPING	576,361.87	90,000.00
5	wits.tri\2.20m.tri\bl...	STOPING	509,623.16	90,000.00
6	wits.tri\2.20m.tri\bl...	STOPING	393,429.35	90,000.00
7	wits.tri\2.20m.tri\bl...	STOPING	406,108.43	90,000.00
8	wits.tri\2.20m.tri\bl...	STOPING	343,905.76	90,000.00
9	wits.tri\2.20m.tri\bl...	STOPING	440,449.29	90,000.00
10	wits.tri\2.20m.tri\bl...	STOPING	471,833.62	90,000.00
11	wits.tri\2.20m.tri\bl...	STOPING	473,110.94	90,000.00
12	wits.tri\2.20m.tri\bl...	STOPING	455,178.40	90,000.00
13	wits.tri\2.20m.tri\bl...	STOPING	402,289.28	90,000.00
14	wits.tri\2.20m.tri\bl...	STOPING	369,446.41	90,000.00
15	wits.tri\2.20m.tri\bl...	STOPING	275,955.93	90,000.00
16	wits.tri\2.20m.tri\bl...	STOPING	259,150.71	90,000.00

Title	Unit	10/06/2018	10/06/2019	10/06/2020	10/06/2021	10/06/2022	10/06/2023
TONNAGE	T	1076303	1082217	1079260	1079260	1079260	1022852

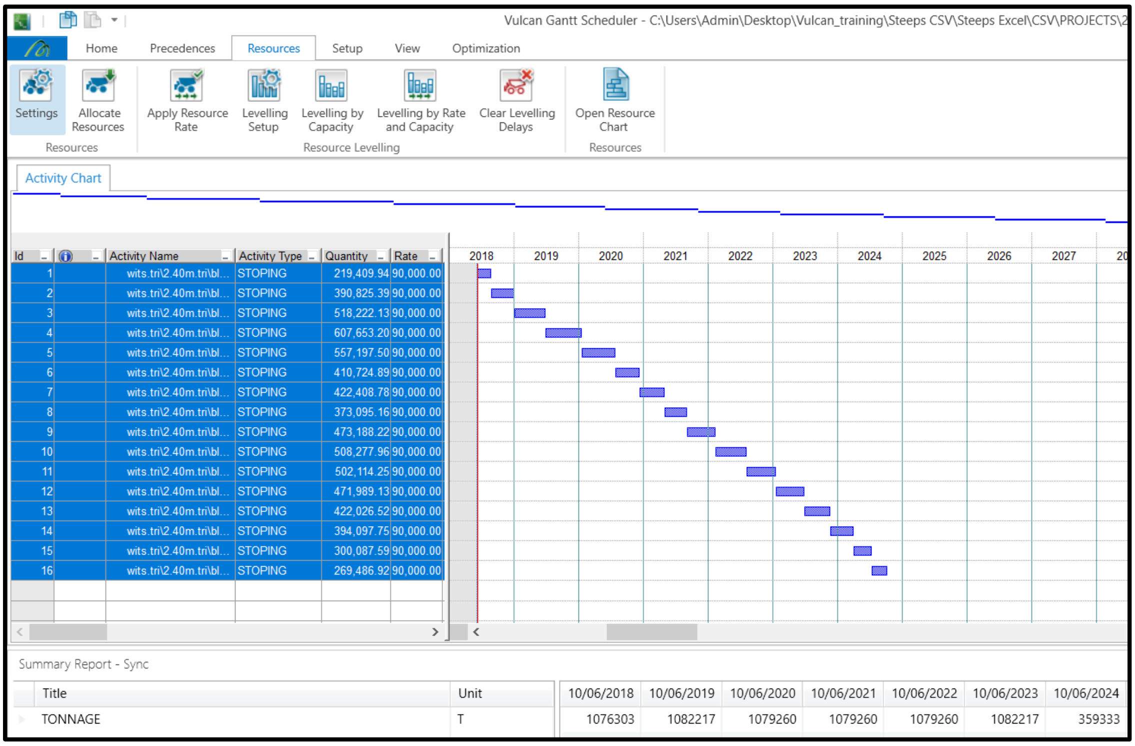
Appendix 7.4.4, 2.30m mining schedule



Id	Activity Name	Activity Type	Quantity	Rate
1	wits.tri\2.30m.tri\bl...	STOPING	214,022.50	90,000.00
2	wits.tri\2.30m.tri\bl...	STOPING	380,755.60	90,000.00
3	wits.tri\2.30m.tri\bl...	STOPING	493,877.70	90,000.00
4	wits.tri\2.30m.tri\bl...	STOPING	593,204.47	90,000.00
5	wits.tri\2.30m.tri\bl...	STOPING	534,299.50	90,000.00
6	wits.tri\2.30m.tri\bl...	STOPING	402,714.63	90,000.00
7	wits.tri\2.30m.tri\bl...	STOPING	414,678.94	90,000.00
8	wits.tri\2.30m.tri\bl...	STOPING	358,906.77	90,000.00
9	wits.tri\2.30m.tri\bl...	STOPING	457,659.28	90,000.00
10	wits.tri\2.30m.tri\bl...	STOPING	490,668.19	90,000.00
11	wits.tri\2.30m.tri\bl...	STOPING	488,819.06	90,000.00
12	wits.tri\2.30m.tri\bl...	STOPING	464,975.10	90,000.00
13	wits.tri\2.30m.tri\bl...	STOPING	412,783.69	90,000.00
14	wits.tri\2.30m.tri\bl...	STOPING	382,553.21	90,000.00
15	wits.tri\2.30m.tri\bl...	STOPING	288,313.73	90,000.00
16	wits.tri\2.30m.tri\bl...	STOPING	264,702.73	90,000.00

Title	Unit	10/06/2018	10/06/2019	10/06/2020	10/06/2021	10/06/2022	10/06/2023	10/06/2024
TONNAGE	T	1076303	1082217	1079260	1079260	1079260	1082217	161463

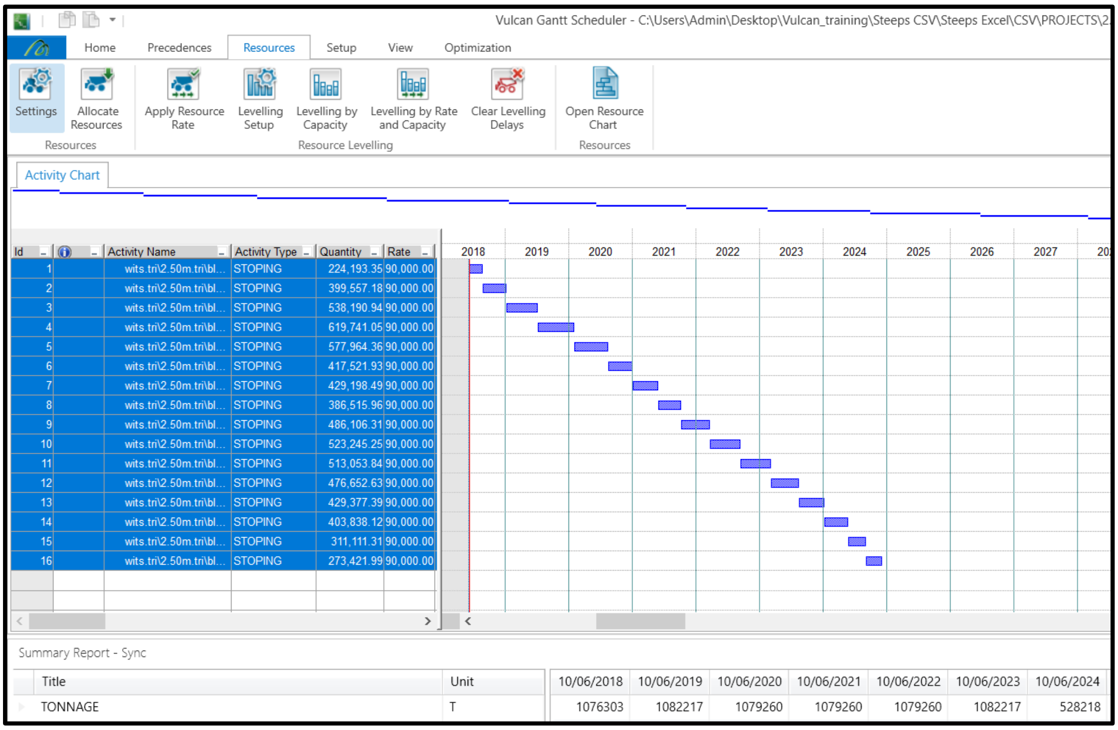
Appendix 7.4.5, 2.40m mining schedule



Id	Activity Name	Activity Type	Quantity	Rate
1	wits.tri\2.40m.tri\bl...	STOPING	219,409.94	90,000.00
2	wits.tri\2.40m.tri\bl...	STOPING	390,825.39	90,000.00
3	wits.tri\2.40m.tri\bl...	STOPING	518,222.13	90,000.00
4	wits.tri\2.40m.tri\bl...	STOPING	607,653.20	90,000.00
5	wits.tri\2.40m.tri\bl...	STOPING	557,197.50	90,000.00
6	wits.tri\2.40m.tri\bl...	STOPING	410,724.89	90,000.00
7	wits.tri\2.40m.tri\bl...	STOPING	422,408.78	90,000.00
8	wits.tri\2.40m.tri\bl...	STOPING	373,095.16	90,000.00
9	wits.tri\2.40m.tri\bl...	STOPING	473,188.22	90,000.00
10	wits.tri\2.40m.tri\bl...	STOPING	508,277.96	90,000.00
11	wits.tri\2.40m.tri\bl...	STOPING	502,114.25	90,000.00
12	wits.tri\2.40m.tri\bl...	STOPING	471,989.13	90,000.00
13	wits.tri\2.40m.tri\bl...	STOPING	422,026.52	90,000.00
14	wits.tri\2.40m.tri\bl...	STOPING	394,097.75	90,000.00
15	wits.tri\2.40m.tri\bl...	STOPING	300,087.59	90,000.00
16	wits.tri\2.40m.tri\bl...	STOPING	269,486.92	90,000.00

Title	Unit	10/06/2018	10/06/2019	10/06/2020	10/06/2021	10/06/2022	10/06/2023	10/06/2024
TONNAGE	T	1076303	1082217	1079260	1079260	1079260	1082217	359333

Appendix 7.4.6, 2.50m mining schedule



Id	Activity Name	Activity Type	Quantity	Rate
1	wits.tri\2.50m.tri\bl...	STOPING	224,193.35	90,000.00
2	wits.tri\2.50m.tri\bl...	STOPING	399,557.18	90,000.00
3	wits.tri\2.50m.tri\bl...	STOPING	538,190.94	90,000.00
4	wits.tri\2.50m.tri\bl...	STOPING	619,741.05	90,000.00
5	wits.tri\2.50m.tri\bl...	STOPING	577,964.36	90,000.00
6	wits.tri\2.50m.tri\bl...	STOPING	417,521.93	90,000.00
7	wits.tri\2.50m.tri\bl...	STOPING	429,198.49	90,000.00
8	wits.tri\2.50m.tri\bl...	STOPING	386,515.96	90,000.00
9	wits.tri\2.50m.tri\bl...	STOPING	486,106.31	90,000.00
10	wits.tri\2.50m.tri\bl...	STOPING	523,245.25	90,000.00
11	wits.tri\2.50m.tri\bl...	STOPING	513,053.84	90,000.00
12	wits.tri\2.50m.tri\bl...	STOPING	476,652.63	90,000.00
13	wits.tri\2.50m.tri\bl...	STOPING	429,377.39	90,000.00
14	wits.tri\2.50m.tri\bl...	STOPING	403,838.12	90,000.00
15	wits.tri\2.50m.tri\bl...	STOPING	311,111.31	90,000.00
16	wits.tri\2.50m.tri\bl...	STOPING	273,421.99	90,000.00

Title	Unit	10/06/2018	10/06/2019	10/06/2020	10/06/2021	10/06/2022	10/06/2023	10/06/2024
TONNAGE	T	1076303	1082217	1079260	1079260	1079260	1082217	528218

Appendix 7.5: Financial model input data
(Zimplats, 2017)

Appendix 7.5.1, Metal recoveries

Metal	Concentrator (%)	Smelter (%)
Pt	82	98
Pd	87	98
Au	74	98
Rh	60	98

Appendix 7.5.2, Payability information

Metal	% Paid
Pt	90
Pd	90
Au	90
Rh	90

Appendix 7.5.3, Metal prices

Metal	Unit	Price
Pt	USD/Oz	928
Pd	USD/Oz	1 053
Au	USD/Oz	1 303
Rh	USD/Oz	1 580

Appendix 7.5.4, Operating costs

Metal	Unit	Operating costs ($/t)
Mining – fixed costs	USD (million)	10.50
Mining – variable costs	USD/t ROM ore	19.29
Processing - concentrator	USD/t feed	10.00
Processing - smelter	USD/t matte	3.29
Transport costs - ore	USD/t ROM ore	1.89
Transport costs – smelter matte	USD/t matte	152.83
Mining overheads	USD/t ROM ore	1.62
Processing overheads	USD/t feed	1.75

Appendix 7.5.5, Royalty and tax rates

Metal	Royalty (%)
Pt	2.50
Pd	2.50
Au	2.50
Rh	2.50
	Tax (%)
Income	28

Appendix 7.6: Full cash flows

Appendix 7.6.1, 2.00 m mining cut

Financial Year				2018	2019	2020	2021	2022	2023
Ore Tonnes			5 923 216	1 076 303	1 082 217	1 079 260	1 079 260	1 079 260	526 917
Ore Grade		Weighted Avg							
Pt (g/t)		1.85		1.85	1.86	1.74	1.94	1.87	1.80
Pd (g/t)		1.51		1.49	1.44	1.53	1.59	1.53	1.44
Rh (g/t)		0.24		0.24	0.25	0.21	0.25	0.23	0.23
Au (g/t)		0.16		0.15	0.16	0.16	0.16	0.16	0.16
		3.75							
Production	koz								
Platinum	352	g	10 938 914	1 986 006.82	2 012 135.27	1 877 115.20	2 098 039.34	2 015 902.07	949 715.14
Palladium	287	g	8 938 009	1 601 498.58	1 559 323	1 647 384	1 719 417	1 649 042	761 345
Rhodium	45	g	1 399 722	263 192.26	265 564	225 372	274 049	249 617	121 928
Gold	30	g	930 813	165 316.13	169 391	168 177	170 445	175 787	81 698
	Recovery		304 864						
	Smelter	Concentrator	oz						
Platinum (oz)	98%	82%	282 966	51 374	52 050	48 557	54 272	52 147	24 567
Palladium (oz)	98%	87%	245 288	43 950	42 793	45 210	47 186	45 255	20 894
Rhodium (oz)	98%	74%	32 680	6 145	6 200	5 262	6 398	5 828	2 847
Gold (oz)	98%	60%	17 450	3 099	3 176	3 153	3 195	3 295	1 532
Concentrator mass pull	t		3.2%	34 872	35 064	34 968	34 968	34 968	17 072
Smelter Matte	t		8.1%	2 825	2 840	2 832	2 832	2 832	1 383
Revenue	LTP Price	Payability	USD million						
Platinum	928	90%	230	41 679 923	42 228 276	39 394 637	44 031 127	42 307 329	19 931 479
Palladium	1 053	90%	226	40 460 209	39 394 678	41 619 455	43 439 296	41 661 334	19 234 599
Rhodium	1 580	90%	45	8 487 962	8 564 448	7 268 255	8 838 077	8 050 157	3 932 190
Gold	1 303	90%	20	3 529 108	3 616 089	3 590 175	3 638 591	3 752 632	1 744 069
USD Revenue				94 157 203	93 803 491	91 872 522	99 947 091	95 771 451	44 842 337
Discount for Payment Terms	3	months	1.029						
Revenue Factor				1.00	1.00	1.00	1.00	1.00	1.00

US$ Revenue		$ million		94.16	93.80	91.87	99.95	95.77	44.84
Inflation		2.0%		1.02	1.04	1.06	1.08	1.10	1.13
		$/t	Sum						
Mining - Fixed Costs		8.64	51	9.30	9.35	9.33	9.33	9.33	4.55
Mining - Variable Costs		19.16	113	20.62	20.73	20.68	20.68	20.68	10.09
Ngezi Concentrator		10.00	11	1.79	1.80	1.80	1.80	1.80	1.80
Smelter		3.29	4	0.59	0.59	0.59	0.59	0.59	0.59
Transport Costs - Ore		1.89	2	0.34	0.34	0.34	0.34	0.34	0.34
Transport Costs - Smelter Matte		152.83	0	0.07	0.07	0.07	0.07	0.07	0.07
Mining Shared Services		1.65	2	0.30	0.30	0.30	0.30	0.30	0.30
Processing Shared Services		1.72	2	0.31	0.31	0.31	0.31	0.31	0.31
Total Operating Costs		$ million	185	33	33	33	33	33	18
USD per ore tonne		31.25	31	30.96	30.95	30.97	30.96	30.96	34.27
EBIT (pre-royalty)		$ million	335	61	60	58	67	62	27
Revenue		$ million	520	94	94	92	100	96	45
Pt Royalty		2.5%	6	1.04	1.06	0.98	1.10	1.06	0.50
Pd Royalty		2.5%	6	1.01	0.98	1.04	1.09	1.04	0.48
Rh Royalty		2.5%	1	0.21	0.21	0.18	0.22	0.20	0.10
Au Royalty		2.5%	0	0.09	0.09	0.09	0.09	0.09	0.04
Royalty - total			13	2.35	2.35	2.30	2.50	2.39	1.12
EBIT		$ million	335	60.84	60.31	58.45	66.54	62.36	26.79
Capital allowance		$ million	6	-	-	-	1.00	2.00	3.00
EBIT - Capital Allowance		$ million	329	60.84	60.31	58.45	65.54	60.36	23.79
EBIT - Capital Allowance		$ million	329	60.84	60.31	58.45	65.54	60.36	23.79
Inflation	2.0%								
Royalty %				2.5%	2.5%	2.5%	2.5%	2.5%	2.5%
Royalty		$ million	13	2.35	2.35	2.30	2.50	2.39	1.12
Tax									
Taxable income		$ million	316	58.48	57.97	56.16	63.04	57.97	22.66
Cumulative taxable income		$ million	300	58.48	115.30	169.20	228.92	282.39	299.52
Tax Rate	28%		89	16.38	16.23	15.72	17.65	16.23	6.35

Profit/Loss		$ million	228	42.11	41.73	40.43	45.39	41.74	16.32
Tax rate from Income Statement	58%			58%	58%	58%	58%	58%	58%
Tax payable (on EBIT - Royalty)		$ million	187	33.92	33.62	32.57	37.14	34.78	14.89
Revenue		$ million	520	94.16	93.80	91.87	99.95	95.77	44.84
Operating Costs		$ million	185	33.32	33.49	33.42	33.41	33.41	18.06
EBIT		$ million	335	60.84	60.31	58.45	66.54	62.36	26.79
Royalty		$ million	13	2.35	2.35	2.30	2.50	2.39	1.12
Tax		$ million	187	33.92	33.62	32.57	37.14	34.78	14.89
FCF		$ million	135	**24.56**	**24.35**	**23.59**	**26.90**	**25.19**	**10.78**
Cumulative FCF		$ million		24.56	48.91	72.49	99.39	124.58	135.35
Real Discount Rate	**12.3 %**			1.12	1.26	1.42	1.59	1.79	2.01
Discounted FCF			94	21.87	19.30	16.65	16.90	14.09	5.37
NPV	**94.19**		Cumulative	21.87	41.17	57.82	74.73	88.82	94.19

Appendix 7.6.2, 2.10m mining cut option

Financial Year				2018	2019	2020	2021	2022	2023
Ore Tonnes			6 179 196	1 076 303	1 082 217	1 079 260	1 079 260	1 079 260	782 897
Ore Grade		Weighted Avg							
Pt (g/t)		1.80		1.81	1.84	1.67	1.89	1.83	1.79
Pd (g/t)		1.50		1.48	1.43	1.49	1.59	1.53	1.45
Rh (g/t)		0.23		0.24	0.24	0.20	0.24	0.22	0.23
Au (g/t)		0.16		0.15	0.16	0.15	0.15	0.16	0.15
		3.69							
Production	koz								
Platinum	359	g	11 152 759	1 947 110.22	1 987 290.91	1 807 617.38	2 035 968.70	1 971 614.86	##########
Palladium	298	g	9 256 296	1 591 448.24	1 547 310	1 611 368	1 720 400	1 653 556	1 132 214
Rhodium	45	g	1 409 614	255 715.41	261 556	212 007	261 421	239 573	179 342
Gold	31	g	960 504	163 512.22	167 829	166 831	166 838	174 318	121 175
	Recovery		315 721						
	Smelter	Concentrator	oz						
Platinum (oz)	98%	82%	288 498	50 368	51 407	46 759	52 666	51 001	36 297
Palladium (oz)	98%	87%	254 023	43 674	42 463	44 221	47 213	45 379	31 072
Rhodium (oz)	98%	74%	32 911	5 970	6 107	4 950	6 103	5 593	4 187
Gold (oz)	98%	60%	18 007	3 065	3 146	3 128	3 128	3 268	2 272
Concentrator mass pull	t		3.2%	34 872	35 064	34 968	34 968	34 968	25 366
Smelter Matte	t		8.1%	2 825	2 840	2 832	2 832	2 832	2 055
Revenue	LTP Price	Payability	USD million						
Platinum	928	90%	234	40 863 608	41 706 872	37 936 100	42 728 463	41 377 882	29 447 772
Palladium	1 053	90%	234	40 206 298	39 091 190	40 709 542	43 464 124	41 775 377	28 604 227
Rhodium	1 580	90%	45	8 246 834	8 435 189	6 837 239	8 430 829	7 726 243	5 783 779
Gold	1 303	90%	21	3 490 599	3 582 762	3 561 438	3 561 604	3 721 283	2 586 794
USD Revenue				92 807 338	92 816 014	89 044 319	98 185 020	94 600 785	66 422 573
Discount for Payment Terms	3	months	1.029						
Revenue Factor				1.00	1.00	1.00	1.00	1.00	1.00
US$ Revenue		$ million		92.81	92.82	89.04	98.19	94.60	66.42
Inflation		2.0%		1.02	1.04	1.06	1.08	1.10	1.13
		$/t	Sum						
Mining - Fixed Costs		8.64	53	9.30	9.35	9.33	9.33	9.33	6.77
Mining - Variable Costs		19.16	118	20.62	20.73	20.68	20.68	20.68	15.00

Ngezi Concentrator		10.00	11	1.79	1.80	1.80	1.80	1.80	1.80
Smelter		3.29	4	0.59	0.59	0.59	0.59	0.59	0.59
Transport Costs - Ore		1.89	2	0.34	0.34	0.34	0.34	0.34	0.34
Transport Costs - Smelter Matte		152.83	0	0.07	0.07	0.07	0.07	0.07	0.07
Mining Shared Services		1.65	2	0.30	0.30	0.30	0.30	0.30	0.30
Processing Shared Services		1.72	2	0.31	0.31	0.31	0.31	0.31	0.31
Total Operating Costs		$ million	192	33	33	33	33	33	25
USD per ore tonne		31.11	31	30.96	30.95	30.97	30.96	30.96	32.15
EBIT (pre-royalty)		$ million	342	59	59	56	65	61	41
Revenue		$ million	534	93	93	89	98	95	66
Pt Royalty		2.5%	6	1.02	1.04	0.95	1.07	1.03	0.74
Pd Royalty		2.5%	6	1.01	0.98	1.02	1.09	1.04	0.72
Rh Royalty		2.5%	1	0.21	0.21	0.17	0.21	0.19	0.14
Au Royalty		2.5%	1	0.09	0.09	0.09	0.09	0.09	0.06
Royalty - total			13	2.32	2.32	2.23	2.45	2.37	1.66
EBIT		$ million	342	59.49	59.32	55.62	64.77	61.19	41.25
Capital allowance		$ million	6	-	-	-	1.00	2.00	3.00
EBIT - Capital Allowance		$ million	336	59.49	59.32	55.62	63.77	59.19	38.25
EBIT - Capital Allowance		$ million	336	59.49	59.32	55.62	63.77	59.19	38.25
Inflation	2.0%								
Royalty %				2.5%	2.5%	2.5%	2.5%	2.5%	2.5%
Royalty		$ million	13	2.32	2.32	2.23	2.45	2.37	1.66
Tax									
Taxable income		$ million	322	57.17	57.00	53.40	61.32	56.82	36.59
Cumulative taxable income		$ million	306	57.17	113.05	164.23	222.33	274.79	306.00
Tax Rate	28%		90	16.01	15.96	14.95	17.17	15.91	10.25
Profit/Loss		$ million	232	41.16	41.04	38.45	44.15	40.91	26.34
Tax rate from Income Statement	58%			58%	58%	58%	58%	58%	58%
Tax payable (on EBIT - Royalty)		$ million	190	33.16	33.06	30.97	36.15	34.12	22.96
Revenue		$ million	534	92.81	92.82	89.04	98.19	94.60	66.42
Operating Costs		$ million	192	33.32	33.49	33.42	33.41	33.41	25.17
EBIT		$ million	342	59.49	59.32	55.62	64.77	61.19	41.25
Royalty		$ million	13	2.32	2.32	2.23	2.45	2.37	1.66

Tax		$ million	190	33.16	33.06	30.97	36.15	34.12	22.96
FCF		$ million	138	**24.01**	**23.94**	**22.43**	**26.17**	**24.71**	**16.63**
Cumulative FCF		$ million		24.01	47.95	70.38	96.55	121.26	137.89
Real Discount Rate	**12.3%**			1.12	1.26	1.42	1.59	1.79	2.01
Discounted FCF			95	21.38	18.98	15.83	16.45	13.83	8.29
NPV	**94.75**		Cumulati ve	21.38	40.36	56.19	72.64	86.47	94.75

Appendix 7.6.3, 2.20m mining cut option

Financial Year				2018	2019	2020	2021	2022	2023
Ore Tonnes			6 419 150	1 076 303	1 082 217	1 079 260	1 079 260	1 079 260	1 022 852
Ore Grade		Weighted Avg							
Pt (g/t)		1.76		1.77	1.80	1.62	1.82	1.79	1.75
Pd (g/t)		1.48		1.46	1.41	1.46	1.57	1.52	1.44
Rh (g/t)		0.22		0.23	0.24	0.19	0.23	0.22	0.22
Au (g/t)		0.15		0.15	0.15	0.15	0.15	0.16	0.15
		3.61							
Production	koz								
Platinum	362	g	11 271 269	1 900 029.92	1 951 198.77	1 743 486.01	1 960 589.94	1 927 251.26	1 788 712.96
Palladium	305	g	9 482 843	1 571 130.40	1 526 998	1 575 386	1 698 826	1 638 619	1 471 884
Rhodium	45	g	1 409 011	247 559.59	256 045	200 044	247 660	232 926	224 776
Gold	32	g	981 345	160 839.88	165 376	164 642	162 540	171 184	156 762
	Recovery		323 448						
	Smelter	Concentrator	oz						
Platinum (oz)	98%	82%	291 563	49 150	50 473	45 100	50 716	49 854	46 270
Palladium (oz)	98%	87%	260 240	43 117	41 906	43 234	46 621	44 969	40 393
Rhodium (oz)	98%	74%	32 897	5 780	5 978	4 670	5 782	5 438	5 248
Gold (oz)	98%	60%	18 397	3 015	3 100	3 087	3 047	3 209	2 939
Concentrator mass pull	t		3.2%	34 872	35 064	34 968	34 968	34 968	33 140
Smelter Matte	t		8.1%	2 825	2 840	2 832	2 832	2 832	2 684
Revenue	LTP Price	Payability	USD million						
Platinum	928	90%	237	39 875 543	40 949 414	36 590 188	41 146 504	40 446 832	37 539 357
Palladium	1 053	90%	240	39 692 989	38 578 020	39 800 509	42 919 082	41 398 023	37 185 622
Rhodium	1 580	90%	45	7 983 808	8 257 470	6 451 423	7 987 046	7 511 876	7 249 047
Gold	1 303	90%	21	3 433 551	3 530 381	3 514 726	3 469 843	3 654 382	3 346 505
USD Revenue				90 985 891	91 315 285	86 356 846	95 522 476	93 011 113	85 320 531
Discount for Payment Terms	3	months	1.029						

Revenue Factor				1.00	1.00	1.00	1.00	1.00	1.00
US$ Revenue		$ million		90.99	91.32	86.36	95.52	93.01	85.32
Inflation		2.0%		1.02	1.04	1.06	1.08	1.10	1.13
		$/t	Sum						
Mining - Fixed Costs		8.64	55	9.30	9.35	9.33	9.33	9.33	8.84
Mining - Variable Costs		19.16	123	20.62	20.73	20.68	20.68	20.68	19.60
Ngezi Concentrator		10.00	11	1.79	1.80	1.80	1.80	1.80	1.80
Smelter		3.29	4	0.59	0.59	0.59	0.59	0.59	0.59
Transport Costs - Ore		1.89	2	0.34	0.34	0.34	0.34	0.34	0.34
Transport Costs - Smelter Matte		152.83	0	0.07	0.07	0.07	0.07	0.07	0.07
Mining Shared Services		1.65	2	0.30	0.30	0.30	0.30	0.30	0.30
Processing Shared Services		1.72	2	0.31	0.31	0.31	0.31	0.31	0.31
Total Operating Costs		$ million	199	33	33	33	33	33	32
USD per ore tonne		30.99	31	30.96	30.95	30.97	30.96	30.96	31.13
EBIT (pre-royalty)		$ million	344	58	58	53	62	60	53
Revenue		$ million	543	91	91	86	96	93	85
Pt Royalty		2.5%	6	1.00	1.02	0.91	1.03	1.01	0.94
Pd Royalty		2.5%	6	0.99	0.96	1.00	1.07	1.03	0.93
Rh Royalty		2.5%	1	0.20	0.21	0.16	0.20	0.19	0.18
Au Royalty		2.5%	1	0.09	0.09	0.09	0.09	0.09	0.08
Royalty - total			14	2.27	2.28	2.16	2.39	2.33	2.13
EBIT		$ million	344	57.67	57.82	52.94	62.11	59.60	53.48
Capital allowance		$ million	6	-	-	-	1.00	2.00	3.00
EBIT - Capital Allowance		$ million	338	57.67	57.82	52.94	61.11	57.60	50.48
EBIT - Capital Allowance		$ million	338	57.67	57.82	52.94	61.11	57.60	50.48
Inflation	2.0%								
Royalty %				2.5%	2.5%	2.5%	2.5%	2.5%	2.5%
Royalty		$ million	14	2.27	2.28	2.16	2.39	2.33	2.13
Tax									
Taxable income		$ million	324	55.39	55.54	50.78	58.72	55.27	48.34

Cumulative taxable income		$ million	308	55.39	109.84	158.47	214.08	265.16	308.31
Tax Rate	28%		91	15.51	15.55	14.22	16.44	15.48	13.54
Profit/Loss		$ million	233	39.88	39.99	36.56	42.28	39.80	34.81
Tax rate from Income Statement	58%			58%	58%	58%	58%	58%	58%
Tax payable (on EBIT - Royalty)		$ million	191	32.13	32.21	29.45	34.64	33.22	29.78
Revenue		$ million	543	90.99	91.32	86.36	95.52	93.01	85.32
Operating Costs		$ million	199	33.32	33.49	33.42	33.41	33.41	31.84
EBIT		$ million	344	57.67	57.82	52.94	62.11	59.60	53.48
Royalty		$ million	14	2.27	2.28	2.16	2.39	2.33	2.13
Tax		$ million	191	32.13	32.21	29.45	34.64	33.22	29.78
FCF		$ million	139	**23.26**	**23.33**	**21.33**	**25.08**	**24.06**	**21.56**
Cumulative FCF		$ million		23.26	46.59	67.92	93.00	117.06	138.62
Real Discount Rate	**12.3%**			1.12	1.26	1.42	1.59	1.79	2.01
Discounted FCF			94	20.71	18.49	15.05	15.77	13.46	10.75
NPV	**94.24**		Cumulative	20.71	39.21	54.26	70.03	83.49	94.24

Appendix 7.6.4, 2.30m mining cut option

Financial Year				2018	2019	2020	2021	2022	2023	2024
Ore Tonnes			6 639 978	1 076 303	1 082 217	1 079 260	1 079 260	1 079 260	1 082 217	161 463
Ore Grade		Weighted Avg								
Pt (g/t)		1.71		1.72	1.76	1.58	1.75	1.75	1.72	1.60
Pd (g/t)		1.46		1.44	1.40	1.44	1.54	1.50	1.43	1.45
Rh (g/t)		0.21		0.22	0.23	0.18	0.22	0.21	0.21	0.18
Au (g/t)		0.15		0.15	0.15	0.15	0.15	0.16	0.15	0.15
		3.53								
Production	koz									
Platinum	365	g	11 355 097	1 855 065.85	1 903 468.65	1 706 515.12	1 885 545.67	1 889 074.56	1 856 493.19	258 933.61
Palladium	311	g	9 666 727	1 551 197.55	1 510 291	1 554 454	1 657 004	1 617 401	1 542 786	233 593
Rhodium	45	g	1 405 671	239 551.94	246 936	193 617	235 893	228 666	232 095	28 912
Gold	32	g	998 306	158 313.51	163 005	162 221	158 716	167 690	163 890	24 472
	Recovery		329 720							
	Smelter	Concentrator	oz							
Platinum (oz)	98%	82%	293 732	47 987	49 239	44 144	48 775	48 866	48 023	6 698
Palladium (oz)	98%	87%	265 286	42 570	41 447	42 659	45 474	44 387	42 339	6 411
Rhodium (oz)	98%	74%	32 819	5 593	5 765	4 520	5 507	5 339	5 419	675
Gold (oz)	98%	60%	18 715	2 968	3 056	3 041	2 975	3 144	3 072	459
Concentrator mass pull	t		3.2%	34 872	35 064	34 968	34 968	34 968	35 064	5 231
Smelter Matte	t		8.1%	2 825	2 840	2 832	2 832	2 832	2 840	424
Revenue	LTP Price	Payability	USD million							
Platinum	928	90%	238	38 931 892	39 947 711	35 814 287	39 571 565	39 645 625	38 961 847	5 434 187
Palladium	1 053	90%	244	39 189 406	38 155 946	39 271 675	41 862 495	40 861 979	38 976 902	5 901 485
Rhodium	1 580	90%	45	7 725 561	7 963 694	6 244 170	7 607 574	7 374 500	7 485 061	932 410
Gold	1 303	90%	21	3 379 619	3 479 764	3 463 034	3 388 204	3 579 776	3 498 668	522 413
USD Revenue				89 226 477	89 547 115	84 793 166	92 429 838	91 461 881	88 922 477	12 790 496
Discount for Payment Terms	3	months	1.029							
Revenue Factor				1.00	1.00	1.00	1.00	1.00	1.00	1.00
US$ Revenue		$ million		89.23	89.55	84.79	92.43	91.46	88.92	12.79
Inflation		2.0%		1.02	1.04	1.06	1.08	1.10	1.13	1.15

		$/t	Sum							
Mining - Fixed Costs		8.64	57	9.30	9.35	9.33	9.33	9.33	9.35	1.40
Mining - Variable Costs		19.16	127	20.62	20.73	20.68	20.68	20.68	20.73	3.09
Ngezi Concentrator		10.00	13	1.79	1.80	1.80	1.80	1.80	1.80	1.80
Smelter		3.29	4	0.59	0.59	0.59	0.59	0.59	0.59	0.59
Transport Costs - Ore		1.89	2	0.34	0.34	0.34	0.34	0.34	0.34	0.34
Transport Costs - Smelter Matte		152.83	1	0.07	0.07	0.07	0.07	0.07	0.07	0.07
Mining Shared Services		1.65	2	0.30	0.30	0.30	0.30	0.30	0.30	0.30
Processing Shared Services		1.72	2	0.31	0.31	0.31	0.31	0.31	0.31	0.31
Total Operating Costs		$ million	208	33	33	33	33	33	33	8
USD per ore tonne		31.39	31	30.96	30.95	30.97	30.96	30.96	30.95	48.91
EBIT (pre-royalty)		$ million	341	56	56	51	59	58	55	5
Revenue		$ million	549	89	90	85	92	91	89	13
Pt Royalty		2.5%	6	0.97	1.00	0.90	0.99	0.99	0.97	0.14
Pd Royalty		2.5%	6	0.98	0.95	0.98	1.05	1.02	0.97	0.15
Rh Royalty		2.5%	1	0.19	0.20	0.16	0.19	0.18	0.19	0.02
Au Royalty		2.5%	1	0.08	0.09	0.09	0.08	0.09	0.09	0.01
Royalty - total			14	2.23	2.24	2.12	2.31	2.29	2.22	0.32
EBIT		$ million	341	55.91	56.05	51.37	59.02	58.05	55.43	4.89
Capital allowance		$ million	10	-	-	-	1.00	2.00	3.00	4.00
EBIT - Capital Allowance		$ million	331	55.91	56.05	51.37	58.02	56.05	52.43	0.89
EBIT - Capital Allowance		$ million	331	55.91	56.05	51.37	58.02	56.05	52.43	0.89
Inflation	2.0%									
Royalty %				2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%
Royalty		$ million	14	2.23	2.24	2.12	2.31	2.29	2.22	0.32
Tax										
Taxable income		$ million	317	53.68	53.82	49.25	55.71	53.76	50.21	0.57
Cumulative taxable income		$ million	301	53.68	106.44	153.60	206.30	256.02	301.21	295.87
Tax Rate	28%		89	15.03	15.07	13.79	15.60	15.05	14.06	0.16
Profit/Loss		$ million	228	38.65	38.75	35.46	40.11	38.71	36.15	0.41
Tax rate from Income Statement	58%			58%	58%	58%	58%	58%	58%	58%
Tax payable (on EBIT - Royalty)		$ million	190	31.13	31.21	28.57	32.89	32.34	30.86	2.65
Revenue		$ million	549	89.23	89.55	84.79	92.43	91.46	88.92	12.79
Operating Costs		$ million	208	33.32	33.49	33.42	33.41	33.41	33.49	7.90

EBIT		$ million	341	55.91	56.05	51.37	59.02	58.05	55.43	4.89
Royalty		$ million	14	2.23	2.24	2.12	2.31	2.29	2.22	0.32
Tax		$ million	190	31.13	31.21	28.57	32.89	32.34	30.86	2.65
FCF		$ million	137	**22.54**	**22.60**	**20.69**	**23.82**	**23.42**	**22.35**	**1.92**
Cumulative FCF		$ million		22.54	45.15	65.83	89.65	113.07	135.42	137.34
Real Discount Rate	**12.3%**			1.12	1.26	1.42	1.59	1.79	2.01	2.25
Discounted FCF			93	20.07	17.92	14.60	14.97	13.11	11.14	0.85
NPV	**92.66**		Cumulative	20.07	37.99	52.59	67.56	80.67	91.81	92.66

Appendix 7.6.5, 2.40m mining cut option

Financial Year				2018	2019	2020	2021	2022	2023	2024
Ore Tonnes			6 837 848	1 076 303	1 082 217	1 079 260	1 079 260	1 079 260	1 082 217	359 333
Ore Grade		Weighted Avg								
Pt (g/t)		1.67		1.68	1.71	1.55	1.69	1.71	1.68	1.61
Pd (g/t)		1.43		1.42	1.38	1.42	1.50	1.48	1.42	1.41
Rh (g/t)		0.20		0.21	0.22	0.18	0.21	0.21	0.21	0.19
Au (g/t)		0.15		0.14	0.15	0.15	0.14	0.15	0.15	0.15
		3.45								
Production	koz									
Platinum	367	g	11 400 326	1 811 492.34	1 853 837.89	1 675 867.15	1 821 355.09	1 846 115.28	1 813 203.67	578 454.61
Palladium	315	g	9 803 045	1 533 200.01	1 491 097	1 531 059	1 614 835	1 595 770	1 531 406	505 678
Rhodium	45	g	1 399 509	231 058.41	238 138	189 347	225 831	224 202	223 264	67 669
Gold	32	g	1 010 779	156 018.66	160 399	159 715	155 737	163 450	162 184	53 276
	Recovery		334 370							
	Smelter	Concentrator	oz							
Platinum (oz)	98%	82%	294 902	46 859	47 955	43 351	47 115	47 755	46 904	14 963
Palladium (oz)	98%	87%	269 027	42 076	40 921	42 017	44 316	43 793	42 027	13 877
Rhodium (oz)	98%	74%	32 675	5 395	5 560	4 421	5 273	5 234	5 213	1 580
Gold (oz)	98%	60%	18 949	2 925	3 007	2 994	2 920	3 064	3 040	999
Concentrator mass pull	t		3.2%	34 872	35 064	34 968	34 968	34 968	35 064	11 642
Smelter Matte	t		8.1%	2 825	2 840	2 832	2 832	2 832	2 840	943
Revenue	LTP Price	Payability	USD million							
Platinum	928	90%	239	38 017 423	38 906 121	35 171 085	38 224 411	38 744 048	38 053 338	12 139 910
Palladium	1 053	90%	248	38 734 716	37 671 017	38 680 622	40 797 143	40 315 482	38 689 396	12 775 430
Rhodium	1 580	90%	45	7 451 644	7 679 957	6 106 462	7 283 046	7 230 522	7 200 276	2 182 313
Gold	1 303	90%	22	3 330 629	3 424 144	3 409 528	3 324 607	3 489 270	3 462 251	1 137 308
USD Revenue				87 534 413	87 681 239	83 367 697	89 629 206	89 779 322	87 405 262	28 234 962
Discount for Payment Terms	3	months	1.029							
Revenue Factor				1.00	1.00	1.00	1.00	1.00	1.00	1.00
US$ Revenue		$ million		87.53	87.68	83.37	89.63	89.78	87.41	28.23
Inflation		2.0%		1.02	1.04	1.06	1.08	1.10	1.13	1.15
		$/t	Sum							

Mining - Fixed Costs		8.64	59	9.30	9.35	9.33	9.33	9.33	9.35	3.11
Mining - Variable Costs		19.16	131	20.62	20.73	20.68	20.68	20.68	20.73	6.88
Ngezi Concentrator		10.00	13	1.79	1.80	1.80	1.80	1.80	1.80	1.80
Smelter		3.29	4	0.59	0.59	0.59	0.59	0.59	0.59	0.59
Transport Costs - Ore		1.89	2	0.34	0.34	0.34	0.34	0.34	0.34	0.34
Transport Costs - Smelter Matte		152.83	1	0.07	0.07	0.07	0.07	0.07	0.07	0.07
Mining Shared Services		1.65	2	0.30	0.30	0.30	0.30	0.30	0.30	0.30
Processing Shared Services		1.72	2	0.31	0.31	0.31	0.31	0.31	0.31	0.31
Total Operating Costs		$ million	214	33	33	33	33	33	33	13
USD per ore tonne		31.29	31	30.96	30.95	30.97	30.96	30.96	30.95	37.29
EBIT (pre-royalty)		$ million	340	54	54	50	56	56	54	15
Revenue		$ million	554	88	88	83	90	90	87	28
Pt Royalty		2.5%	6	0.95	0.97	0.88	0.96	0.97	0.95	0.30
Pd Royalty		2.5%	6	0.97	0.94	0.97	1.02	1.01	0.97	0.32
Rh Royalty		2.5%	1	0.19	0.19	0.15	0.18	0.18	0.18	0.05
Au Royalty		2.5%	1	0.08	0.09	0.09	0.08	0.09	0.09	0.03
Royalty - total			14	2.19	2.19	2.08	2.24	2.24	2.19	0.71
EBIT		$ million	340	54.21	54.19	49.95	56.22	56.37	53.91	14.84
Capital allowance		$ million	10	-	-	-	1.00	2.00	3.00	4.00
EBIT - Capital Allowance		$ million	330	54.21	54.19	49.95	55.22	54.37	50.91	10.84
EBIT - Capital Allowance		$ million	330	54.21	54.19	49.95	55.22	54.37	50.91	10.84
Inflation	2.0%									
Royalty %				2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%
Royalty		$ million	14	2.19	2.19	2.08	2.24	2.24	2.19	0.71
Tax										
Taxable income		$ million	316	52.03	52.00	47.86	52.98	52.12	48.73	10.13
Cumulative taxable income		$ million	295	52.03	103.00	148.85	198.90	247.13	291.01	295.43
Tax Rate	28%		88	14.57	14.56	13.40	14.83	14.59	13.64	2.84
Profit/Loss		$ million	227	37.46	37.44	34.46	38.14	37.53	35.08	7.29
Tax rate from Income Statement	58%			58%	58%	58%	58%	58%	58%	58%
Tax payable (on EBIT - Royalty)		$ million	189	30.18	30.16	27.76	31.31	31.39	30.00	8.20
Revenue		$ million	554	87.53	87.68	83.37	89.63	89.78	87.41	28.23
Operating Costs		$ million	214	33.32	33.49	33.42	33.41	33.41	33.49	13.40

EBIT		$ million	340	54.21	54.19	49.95	56.22	56.37	53.91	14.84
Royalty		$ million	14	2.19	2.19	2.08	2.24	2.24	2.19	0.71
Tax		$ million	189	30.18	30.16	27.76	31.31	31.39	30.00	8.20
FCF		$ million	137	**21.85**	**21.84**	**20.10**	**22.67**	**22.73**	**21.73**	**5.94**
Cumulative FCF		$ million		21.85	43.69	63.79	86.46	109.19	130.92	136.85
Real Discount Rate	**12.3%**			1.12	1.26	1.42	1.59	1.79	2.01	2.25
Discounted FCF			91	19.46	17.31	14.19	14.25	12.72	10.83	2.63
NPV	**91.39**		Cumulative	19.46	36.77	50.96	65.21	77.93	88.76	91.39

Appendix 7.6.6, 2.50m mining cut option

Financial Year			865028.7936	2018	2019	2020	2021	2022	2023	2024
Ore Tonnes			7 006 733	1 076 303	1 082 217	1 079 260	1 079 260	1 079 260	1 082 217	528 218
Ore Grade		Weighted Avg								
Pt (g/t)		1.63		1.65	1.67	1.52	1.64	1.66	1.63	1.61
Pd (g/t)		1.41		1.41	1.36	1.40	1.46	1.46	1.41	1.37
Rh (g/t)		0.20		0.21	0.21	0.17	0.20	0.20	0.20	0.20
Au (g/t)		0.15		0.14	0.15	0.15	0.14	0.15	0.15	0.15
		3.38								
Production	koz									
Platinum	367	g	11 407 219	1 773 233.49	1 808 547.98	1 644 652.91	1 767 849.40	1 796 384.55	1 766 481.05	850 069.70
Palladium	318	g	9 888 486	1 514 477.65	1 470 920	1 507 942	1 576 137	1 570 635	1 523 408	724 966
Rhodium	45	g	1 390 797	224 031.86	230 399	184 905	217 817	218 764	211 850	103 030
Gold	33	g	1 018 431	153 786.64	157 821	157 209	152 992	159 085	160 836	76 701
	Recovery		337 284							
	Smelter	Concentrator	oz							
Platinum (oz)	98%	82%	295 080	45 870	46 783	42 544	45 730	46 469	45 695	21 989
Palladium (oz)	98%	87%	271 372	41 562	40 367	41 383	43 254	43 103	41 807	19 895
Rhodium (oz)	98%	74%	32 471	5 231	5 379	4 317	5 085	5 108	4 946	2 405
Gold (oz)	98%	60%	19 093	2 883	2 959	2 947	2 868	2 982	3 015	1 438
Concentrator mass pull	t		3.2%	34 872	35 064	34 968	34 968	34 968	35 064	17 114
Smelter Matte	t		8.1%	2 825	2 840	2 832	2 832	2 832	2 840	1 386
Revenue	LTP Price	Payability	USD million							
Platinum	928	90%	239	37 214 492	37 955 630	34 515 998	37 101 498	37 700 359	37 072 780	17 840 241
Palladium	1 053	90%	250	38 261 715	37 161 287	38 096 589	39 819 466	39 680 484	38 487 334	18 315 510
Rhodium	1 580	90%	45	7 225 038	7 430 382	5 963 199	7 024 600	7 055 133	6 832 184	3 322 719
Gold	1 303	90%	22	3 282 981	3 369 107	3 356 050	3 266 011	3 396 083	3 433 474	1 637 378
USD Revenue				85 984 226	85 916 406	81 931 835	87 211 574	87 832 059	85 825 771	41 115 848
Discount for Payment Terms	3	months	1.029							
Revenue Factor				1.00	1.00	1.00	1.00	1.00	1.00	1.00
US$ Revenue		$ million		85.98	85.92	81.93	87.21	87.83	85.83	41.12
Inflation		2.0%		1.02	1.04	1.06	1.08	1.10	1.13	1.15

		$/t	Sum							
Mining - Fixed Costs		8.64	61	9.30	9.35	9.33	9.33	9.33	9.35	4.56
Mining - Variable Costs		19.16	134	20.62	20.73	20.68	20.68	20.68	20.73	10.12
Ngezi Concentrator		10.00	13	1.79	1.80	1.80	1.80	1.80	1.80	1.80
Smelter		3.29	4	0.59	0.59	0.59	0.59	0.59	0.59	0.59
Transport Costs - Ore		1.89	2	0.34	0.34	0.34	0.34	0.34	0.34	0.34
Transport Costs - Smelter Matte		152.83	1	0.07	0.07	0.07	0.07	0.07	0.07	0.07
Mining Shared Services		1.65	2	0.30	0.30	0.30	0.30	0.30	0.30	0.30
Processing Shared Services		1.72	2	0.31	0.31	0.31	0.31	0.31	0.31	0.31
Total Operating Costs		$ million	219	33	33	33	33	33	33	18
USD per ore tonne		31.20	31	30.96	30.95	30.97	30.96	30.96	30.95	34.25
EBIT (pre-royalty)		$ million	337	53	52	49	54	54	52	23
Revenue		$ million	556	86	86	82	87	88	86	41
Pt Royalty		2.5%	6	0.93	0.95	0.86	0.93	0.94	0.93	0.45
Pd Royalty		2.5%	6	0.96	0.93	0.95	1.00	0.99	0.96	0.46
Rh Royalty		2.5%	1	0.18	0.19	0.15	0.18	0.18	0.17	0.08
Au Royalty		2.5%	1	0.08	0.08	0.08	0.08	0.08	0.09	0.04
Royalty - total			14	2.15	2.15	2.05	2.18	2.20	2.15	1.03
EBIT		$ million	337	52.66	52.42	48.51	53.80	54.42	52.33	23.02
Capital allowance		$ million	10	-	-	-	1.00	2.00	3.00	4.00
EBIT - Capital Allowance		$ million	327	52.66	52.42	48.51	52.80	52.42	49.33	19.02
EBIT - Capital Allowance		$ million	327	52.66	52.42	48.51	52.80	52.42	49.33	19.02
Inflation	2.0%									
Royalty %				2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%
Royalty		$ million	14	2.15	2.15	2.05	2.18	2.20	2.15	1.03
Tax										
Taxable income		$ million	313	50.52	50.28	46.46	50.62	50.23	47.19	18.00
Cumulative taxable income		$ million	294	50.52	99.80	144.31	192.10	238.56	281.06	293.55
Tax Rate	28%		88	14.14	14.08	13.01	14.17	14.06	13.21	5.04
Profit/Loss		$ million	226	36.37	36.20	33.45	36.45	36.16	33.97	12.96
Tax rate from Income Statement	58%			58%	58%	58%	58%	58%	58%	58%
Tax payable (on EBIT - Royalty)		$ million	188	29.30	29.16	26.95	29.94	30.29	29.11	12.76
Revenue		$ million	556	85.98	85.92	81.93	87.21	87.83	85.83	41.12
Operating Costs		$ million	219	33.32	33.49	33.42	33.41	33.41	33.49	18.09

EBIT		$ million	337	52.66	52.42	48.51	53.80	54.42	52.33	23.02
Royalty		$ million	14	2.15	2.15	2.05	2.18	2.20	2.15	1.03
Tax		$ million	188	29.30	29.16	26.95	29.94	30.29	29.11	12.76
FCF		$ million	136	**21.22**	**21.12**	**19.51**	**21.68**	**21.93**	**21.08**	**9.24**
Cumulative FCF		$ million		21.22	42.33	61.85	83.53	105.46	126.54	135.78
Real Discount Rate	**12.3%**			1.12	1.26	1.42	1.59	1.79	2.01	2.25
Discounted FCF			90	18.89	16.74	13.78	13.63	12.28	10.50	4.10
NPV	**89.91**		Cumulative	18.89	35.63	49.41	63.03	75.31	85.81	89.91

Appendix 7.7: Sensitivity analysis for the 2.10m mining cut option

Appendix 7.7.1, Sensitivity to Pt prices

Platinum Price	742	835	928	1 021	1 114
% change	0.80	0.90	1.00	1.10	1.20
NPV	82	89	95	102	108

An increase in Pt price results in increase in project value at 12.3% discount rate.

Appendix 7.7.2, Sensitivity to Pd prices

Palladium price	842	948	1 053	1158	1 264
% change	0.80	0.90	1.00	1.10	1.20
NPV	82	89	95	102	108

An increase in Pd price results in improvement in project value at 12.3% discount rate.

Appendix 7.7.3, Sensitivity to Au prices

Gold price	1 042	1 172	1 303	1 433	1 563
% change	0.80	0.90	1.00	1.10	1.20
NPV	94	95	95	96	96

An increase in Au prices results in a marginal increase in project value at 12.3% discount rate.

Appendix 7.7.4, Sensitivity to Rh prices

Rhodium price	1 264	1422	1 580	1 738	1 896
% change	0.80	0.90	1.00	1.10	1.20
NPV	93	94	95	97	98

An increase in Rh prices results in a marginal increase in project value at 12.3% discount rate

Appendix 7.7.5, Sensitivity to real discount rate

Discount Rate	9.8%	11.1%	12.3%	13.5%	14.8%
% change	0.80	0.90	1.00	1.10	1.20
NPV	103	99	95	92	88

Changes to discount rate significantly changes project value.

Appendix 7.7.6, Sensitivity Pt recovery

Recovery Pt (concentrator)	66%	74%	82%	90%	99%
% change	0.80	0.90	1.00	1.10	1.20
NPV	82	89	95	102	109

A 20% improvement in Pt recovery result in an increase in NPV of $14 million.

Appendix 7.7.7, Sensitivity Pd recovery

Recovery Pd (concentrator)	70%	78%	87%	96%	105%
% change	0.8	0.9	1	1.1	1.2
NPV	82	89	95	102	

A 10% improvement in Pd recovery result in an increase in NPV of $7 million.

Appendix 7.7.8, Sensitivity Au recovery

Recovery Au (concentrator)	48%	54%	60%	65%	71%
% change	0.80	0.90	1.00	1.10	1.20
NPV	94	95	95	96	96

A 20% improvement in Au recovery result in an increase in NPV of $1 million.

Appendix 7.7.9, Sensitivity Rh recovery

Recovery Rh (concentrator)	59%	67%	74%	82%	89%
% change	0.80	0.90	1.00	1.10	1.20
NPV	93	94	95	97	98

A 20% improvement in Rh recovery result in an increase in NPV of $3 million.