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Ore reserves generation at variable development and stoping rates for a UG2 Bushveld Complex platinum reef conventional mining layout

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Ore reserves generation at variable development and stoping rates for a UG2 Bushveld Complex platinum reef conventional mining layout

Lefu Justinus Mohloki

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, Mining Engineering.

Johannesburg, 2009

DECLARATION

I declare that this research report is my own, unaided work. It is being submitted for the degree of Master of Science in Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.

_____

Lefu Justinus Mohloki

Dated this 21st Day of December 2009

ABSTRACT

Ore reserves generation at a rate lower than the depletion or mining rate will lead to a shortage of places to be mined usually referred to as 'lack of flexibility'.

The purpose of the research was to critically evaluate one major capital project that was about to be implemented with a view to establish if there exists a balance between depletion and replacement of ore reserves in that particular project. The project was planned to deliver 100 000 tonnes per month.

Numerical modelling was carried out in Mine2-4D and Earthworks Production Scheduler, EPS, the results, of which, were imported into Microsoft Excel for further analysis.

The analysis revealed that:-

1 Planned production level (100 000 tonnes per month) cannot be achieved at the planned development rates.
2 Production levels above 33 000 tonnes per month (15m/month face advance) takes the system out of balance.
3 Optimal Production equals 33 000 tonnes per month achieved in 4 years and maintained over 9 years.

Project flexibility should be assessed before any project can be approved, and monitored throughout the life of the project.

DEDICATION

In memory of my brother-in-law

Patrick Modisaotsile (Isaac) Ramasimong

and my sister

Mamodiehi (Mahlopho) Catherine Mohloki-Seng

ACKNOWLEDGEMENTS

I am indebted to my wife Grace, my children Lerato, Lefu-Junior and Christian for the precious time that they sacrificed for me to get the research done.

I would like to express my sincere appreciation to Mr Cuthbert Musingwini, my supervisor, for the time and resources he sacrificed in affording me to work on this research.

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1. INTRODUCTION

Ore reserves are a key pillar to the continued survival of a mine. A sustainable generation of ore reserves ensures the continued existence of a mine. Most companies can expend more time and resources in developing a mine to a steady state capacity but pay little consideration to the requirements to maintain steady state once achieved. In most instances it is taken for granted that once steady-state is achieved, everything will fall into place and planned production tempo would be maintained. This can be deduced from the amount of time and resources that are expended to define and accurately cost the capital footprint in any capital project as compared to the amount of time and resources expended in planning the working cost phase. Musingwini, Minnitt and Woodhall (2006) tabulated 5 recent analyses that were done on mine plans and noted that some of the analyses take no account of flexibility, while some that refer to flexibility in evaluating mine plans do not consider it in the final analysis. As a consequence, most previous studies have underplayed the importance of flexibility in mine plans.

It is important for organisations to measure projects against their Investment Proposal Reports (IPRs), but even most important, is for the organisations to get their IPRs correct so that they can measure themselves against realistic and achievable goals. Management concentrate on the profitability of a company as compared to previous measurement periods with little or no regard to the original investment proposal and, as a result, a lot of value may be destroyed in the process. Observations on some companies revealed that, the life of mine plan is adjusted every time planning is done and there were also yearly plans that were based on previous achievements and there is rarely mention of or reference to the Investment Proposal Reports. This practice

encourages project teams to hasten projects approval with knowledge that the IPRs will never be revisited.

Smith *et al* (2006) stated that the mineral resource forms the fundamental asset of any mining company. In order to optimize economic return(viz. to maximize net present value 'NPV'), clarity is required on, amongst other things, the exploitation strategy, mining duration, capital and operating costs and the optimal scale of operations. The optimal scale of operations is directly linked to the match between the development and stoping rates. An incorrect match between development and stoping rates could result in the value of this important asset not being realized.

If reserves are generated at a rate slower than the stoping rate, a mine will be forced to close down because there will simply be no reserves to mine. It is imperative therefore that development is kept well ahead of stoping. Musingwini, Ali and Dikgale (2009) defined mining replacement rate as the rate at which development generates new stopes to replace depleting ones thus, sustaining production. However, there is no standard way of determining the balance between development and stoping as practitioners rely on rules of thumb. Musingwini, Minnitt and Woodhall (2006) noted that mining layouts and schedules or, alternatively, mine plans have often been analysed, compared and optimized on the basis of their performance against an array of directly measurable and quantifiable parameters, such as cost per tonne and net present value (NPV). One could therefore argue that the analyses were fruitless exercises since they were based on tonnage profiles that would or would not be realized. They further noted the differences between strategic and tactical flexibility which they defined as follows:

- Strategic Flexibility – Flexibility obtained by structuring the company's portfolio of operations to be able to divest from unprofitable operations, take on board new operations able to generate the minimum required levels of profitability and schedule when Greenfield projects can be brought on stream.
- Tactical Flexibility - The ability to swiftly move the mining operations to different production faces when issues of grade control or unpredicted geological structures require it.

Some sort of assessment needs to be done on the tactical level which informs the decisions made on the strategic level for a company to be able to invest or divest from a given project.

Tamlyn (1994) observed, for gold mines, that working costs were increasing as a result of declining grades, increasing mining depths, wage increases and increases in administered prices such as stores, water and electricity. He further presented a working cost breakdown graph that indicated labour costs to vary from ± R 60 to ± R 82 per ton in comparison to the stores cost ranging from ± R 30 to ± R 56 per ton , however the independent axis of the graph was not labeled. It can be assumed that these costs were measured over the same time period from 1979 to 1992 as in his Figure 1 in the same document. The picture has not changed much because a benchmarked figure of labour costs as a percentage of operating costs at eight platinum mines using conventional mining methods are as presented in Table 1.1.

Table 1 : Labour costs percentage of operating costs

Shaft	Labour Cost % of Total Costs
1	38%
2	67%
3	57%
4	67%
5	69%
6	65%
7	58%
8	73%
Average	62%
Min	38%
Max	73%

The table above shows how important and big a component labour is, of the total operating costs. Producing more tonnes beyond the breakeven point ensures a higher recovery of the labour costs. Labour costs will not increase with extra tonnes produced within the labour force's capability while producing less than the labour force capacity will erode the profitability of the organisation. An over-recovery is more desirable than an under-recovery which backs the argument that labour should be productively deployed at all times. There should never be a case where labour is underutilised because of a lack of places to be mined.

Mosenthal (1992) presented productivity figures for Category 1 to 8 employees for the gold mining industry covering the period 1945 to 1990. Productivity figures have increased steadily from 25 to 34 tonnes per man per month. At that time he had already observed that labour was the biggest component of the operating costs as observed later by Tamlyn (1994). Platinum producers are producing in the range of 25 to 35 centares per man per month which equates to about 100 to

150 tonnes per man per month. The cost of paying for idle labour is huge in that it is not only the cost paid to labour in wages and benefits; there is also a loss of revenue and profitability that should be brought into account.

Smith (1992) defined planning as the specification of a future course of action to achieve defined objectives within given constraints. He further stated that in the free market environment the measure of success of a company is generally considered to be its ability to generate profits. The inability of a project to generate profits during unavailability of reserves due to bad planning also has a direct impact on the shareholders' equity and return on investment. Capital spent on the particular project could have been better spent on some project that could have brought some positive returns. The concept of planning by Smith (1992) therefore implies that mining replacement rate shall be part of the planning process.

1.1. REPLACEMENT RATE

Storrar (1977) indicated that most tabular reef gold mines on the Witwatersrand Basin considered a figure of two years as being a safe value for 'apparent ore reserve life'. McCarthy (2002) noted that it is usual to keep primary access development two years ahead of production in longhole stoping operations of narrow reefs. Lanham (2004) cited two years as an ideal amount of ore reserves that must be kept between development and stoping but noted that some platinum mines on the Bushveld Complex are comfortable with a figure of 12 months because of the associated simpler geology. It would appear therefore that the balance between development and stoping should be kept around two years for most narrow tabular reef deposits. However, development rates and stoping rates vary from time to time depending on such factors as logistical constraints, portability of skills, skills shortage or driving through poor ground conditions. Therefore the balance between development and stoping of two years should be adjusted to suit site-specific conditions determined by the geological complexity of the orebody.

A lot of improvements have been made in the mining industry, many of which were brought about by the introduction of technology. Tamlyn (1994) noted panel advances of four to five metres per month as very typical in the mid 90's. Currently it is not unusual to hear of mines that achieve panel advances in excess of fifteen metres per month on a sustained basis in panels with up to 36m of facelength. The improvements were necessitated, by amongst other things, the rising cost of production. The achievement of these high stoping rates has resulted in many companies being highly optimistic in estimating their production rates for their new projects while neglecting the need for development rates to be increased to keep up with faster depletion rates

Companies conduct benchmark studies of rates and fall into traps of assuming that they would achieve these optimistic rates without taking into account:

- the geological conditions under which the rates are achieved,
- the organisational environment or behaviour in those benchmarked organisations,
- literacy levels of labour,
- skills levels of labour,
- management ability to comprehend and institute the required measures,
- internal human relations between management and the union and other stakeholders,
- reward strategies for management and labour,
- motivation of employees,
- relationship with and the strength or influence of the union on the workforce, and
- logistics and integration of different functions.

The rules of thumb stated earlier, as with any other rule of thumb, need to be applied in cases where the conditions are close to or almost similar to the conditions observed while they were derived. This fact will be obviated with the research at hand. The orebody is highly faulted and the findings show that this orebody defies all logical assumptions that would hold true to other, undisturbed orebodies. The research was prompted by the very application of rules of thumb to such a faulted and geologically different orebody as if nothing was different. Further, the main thrust of this research is to obviate the need to conduct technical flexibility analyses on the tactical level to ensure sustainable mining operations.

As part of this research, an investigation was undertaken to establish how platinum mines determine their replacement rates and what the value represents. Most mining houses were found to take the total stated ore reserves, divided by the total planned development metres necessary to open up the reserves. The quotient would then be assumed to represent the amount of ore-reserves exposed per metre of development advanced, a factor called replacement ratio or replacement factor.

While the method, in the absence of any scientific measure, gives one some sort of control and a good feel of what sort of performance should be aimed for, it does however give some false sense of security. Most of these reserves that are supposedly exposed are not immediately available to be mined. Only one mining group in the platinum mining sector was found to state their reserves as a ratio and further have different categories into which they would classify their reserves. Some of the categories used are:

- Reserves being developed
- Developed reserves not equipped,
- Developed reserves available for mining,
- Developed but not available for mining

The classification method mentioned above is a step in the right direction; however there are pitfalls in its application. If a raiseline containing a vast amount of reserves is ready for mining, it may give a sense that there is ample space for teams to mine while there may be a limitation of the

number of teams that can be deployed in that raiseline. This “ready to mine” raiseline may increase the ratio but may not alleviate a problem of idle teams seeking places to mine.

It would seem logical that the classification categories stated above would be used in analysing new projects to be implemented so that the reserves availability risk, as defined above, could be obviated, guarded against and possibly eliminated.

The project in question is to be implemented by the mining group that has the classification system outlined above but this analysis and classification was not done and the project was planned to deliver 100 000 tonnes per month, which can be considered to be highly optimistic and not achievable.

The analysis revealed that the project can, at most, achieve 50% of the planned production and this will only be achieved at an exceptional stoping face advance rate of 20m per month which is purely not sustainable. Another point that is obviated by the research is that the increased stoping rate erodes the mining flexibility in that the reserves available to be mined drops substantially leaving few areas to be mined in cases of emergency, where geological features are encountered and there is a safety or logistical need to temporarily abandon the working place.

1.2. RESEARCH METHODOLOGY

The design was done in Mine2-4D® and scheduling carried out in Earthworks Production Scheduler. This design was informed by the input from geology, rock engineering, ventilation, valuation, mining production, engineering, human resource, best practice, planning, strategic planning, projects, technical services and the finance departments.

The initial intention was to establish a balance between stoping rates and the development rates if any such balance exists as described in the research proposal. The research was decided upon while the project was still underway. It became apparent that there was an even bigger problem on which the project was premised and hence the departure from the initial topic.

During the design and scheduling processes, it became apparent that there was little flexibility in varying the development rates upwards since these were the best practice rates that were generally being achieved by the company at large. The company was confident that these were the best achievable rates and there was virtually little room for improvement unless there was a huge departure from the current *modus operandi*, the departure would amongst other things necessitate multi-blasting of development ends. Multi-blasting would necessitate a total redesign of the ventilation system. The ventilation system for this project is interlinked with the neighbouring shafts and it was felt that the redesign would cause interruptions in production for the total complex and the cost coupled therewith would outweigh the benefits thereof.

For the purpose of this research, the development rates were fixed and only stoping rates were varied to try and establish a good balance between the best practice rates and the stoping rates. This was decided because as stated in the previous paragraph, there was only scope to vary the development rates down which would further constrain the stoping rates and no benefit would be derived from such an analysis. Stoping rates were varied from a linear advance rate of 10m per month in increments of 1m per month to a maximum of 20m per month.

1.3. REPORT DESCRIPTION

The report is divided into 3 chapters and the 4th chapter gives the conclusions of the research.

1.3.1. Chapter 1

This first chapter is aimed at giving the reader a brief background to the problem that has been investigated and the reasons why the issues should have been investigated. This chapter covers some work that was done in the past and the current situation with regard to the technical flexibility analyses.

1.3.2. Chapter 2

Section 2.1, the first part of chapter 2, gives a brief background of the project area focusing mostly on the geological structure. It further gives details about how geological data was acquired and the geological structure defined. This section concludes with the stratigraphy of the project area and the neighbouring shafts.

Sections 2.2 and 2.3 describe how the backlength was optimised and further discusses the way panels were laid out.

Section 2.4 covers the primary development of the project and how the project area will be accessed.

Section 2.5 covers the discussion of different geological formations and the justification for the geological formation in which the footwall drives would be located.

Section 2.6 is devoted the description of different types of secondary and tertiary development excavations contained in the design.

Section 2.7 covers the derivation of factors that were applied during the resource to reserve conversion.

Section 2.8 covers the scheduling principles and criteria used in the analysis.

1.3.3. Chapter 3

This chapter is devoted to the analysis of the results of the research and further discusses the analysis in details.

1.3.4. Chapter 4

This chapter covers the conclusions drawn from this research.

2. BRIEF PROJECT BACKGROUND.

The resource is situated at a depth of 30m at the sub-outcrop to 400m at the deepest extreme which forms a boundary with other down-dip shafts. The magnetic survey in Figure 1 shows that the area is traversed by major geological features although the magnetic survey picture does not give depth perception or definition of these structures in three dimensions. The aerial and ground magnetic surveys conducted are consistent with the known geological features in this area.

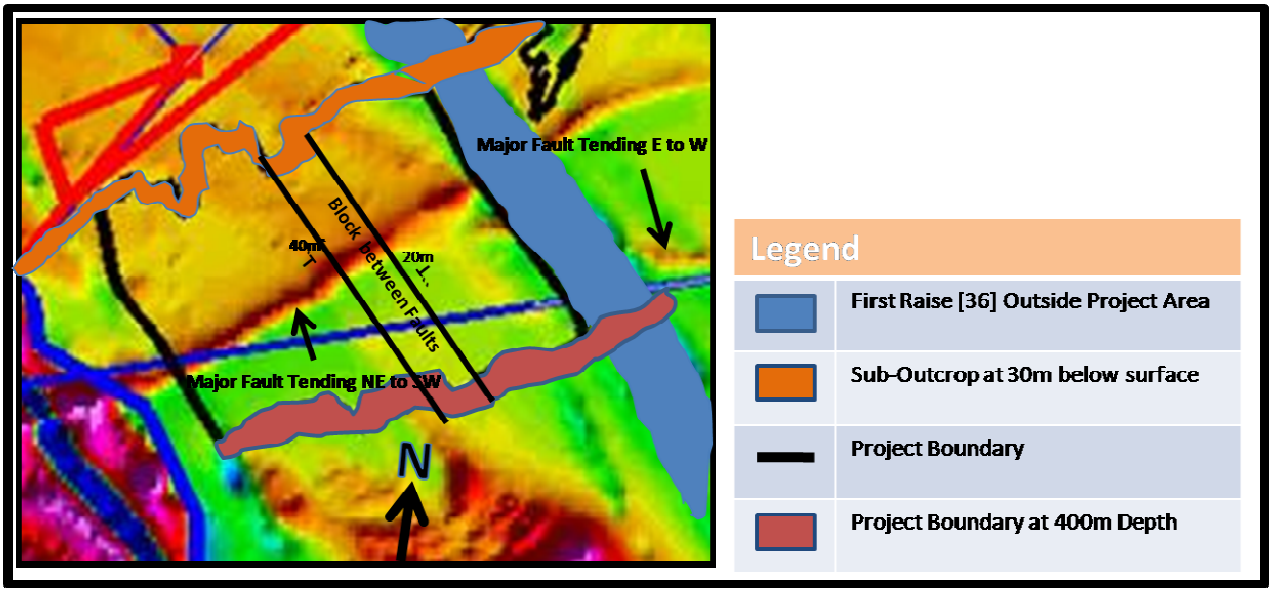


Figure 1 : Aerial magnetic survey (Courtesy of Anglo Platinum)

The confidence in the mineral resource is high because the area has been extremely explored by :

- drilling surface exploration boreholes,
- wire logging of two surface holes 5km outside the property,
- drilling exploration holes from the overlying, mined out Merensky horizon to intersect the UG2,
- underground geological mapping on the Merensky horizon and UG2 intersections ,

- super-imposition of Merensky structures on the UG2, and
- a two dimensional seismic survey that was conducted to further define geological structures in the project area.

Figures 2, 3 and 4 illustrate the mine planning and associated geology in the project area.

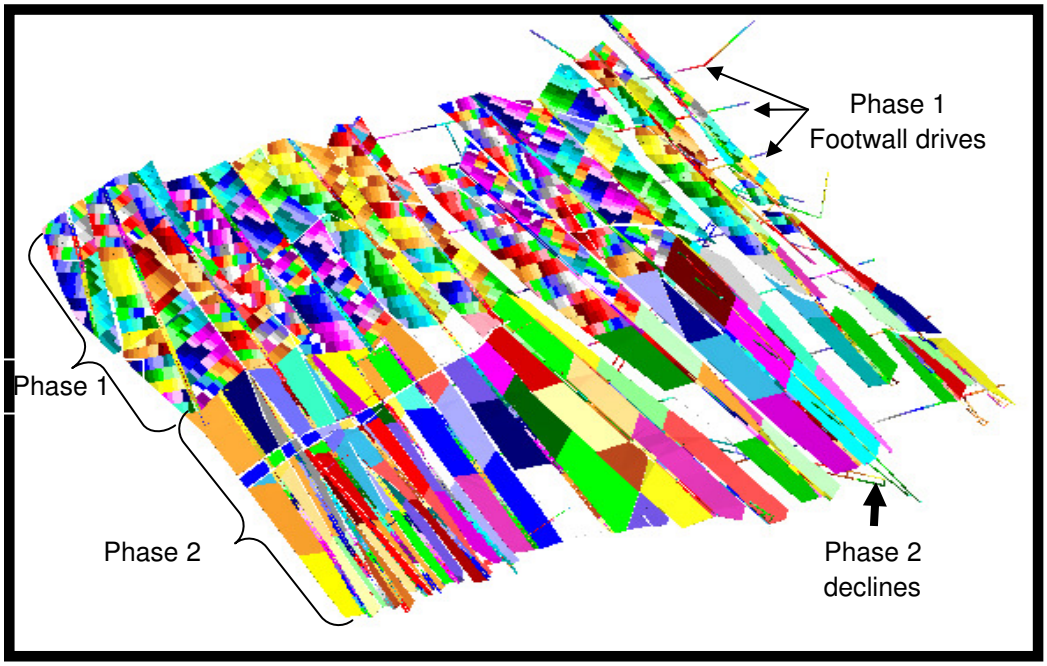


Figure 2 : Project schedule in plan

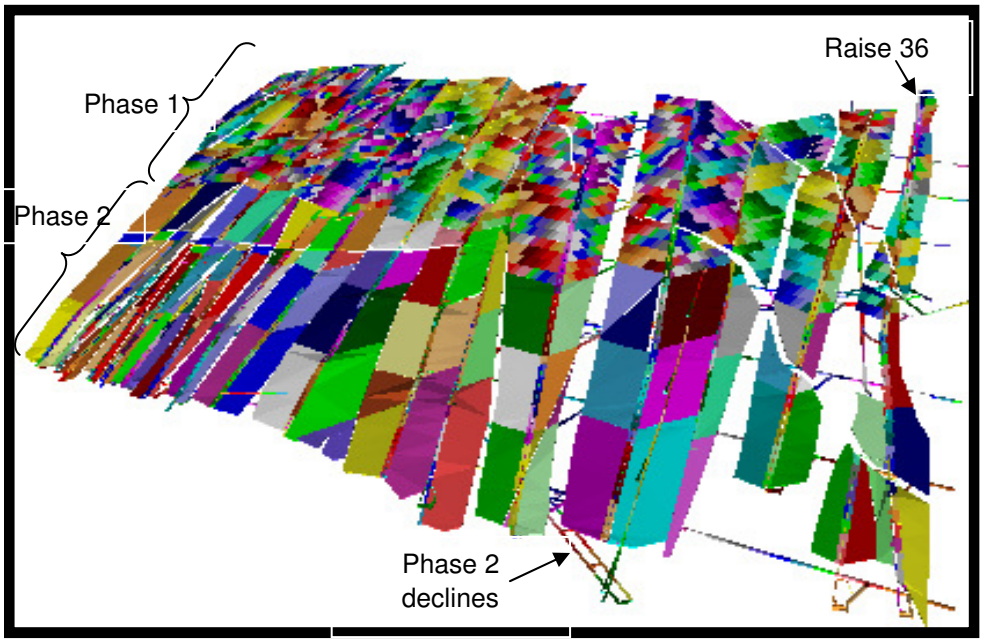


Figure 3 : A 3D perspective view of the project schedule in Mine2-4D®

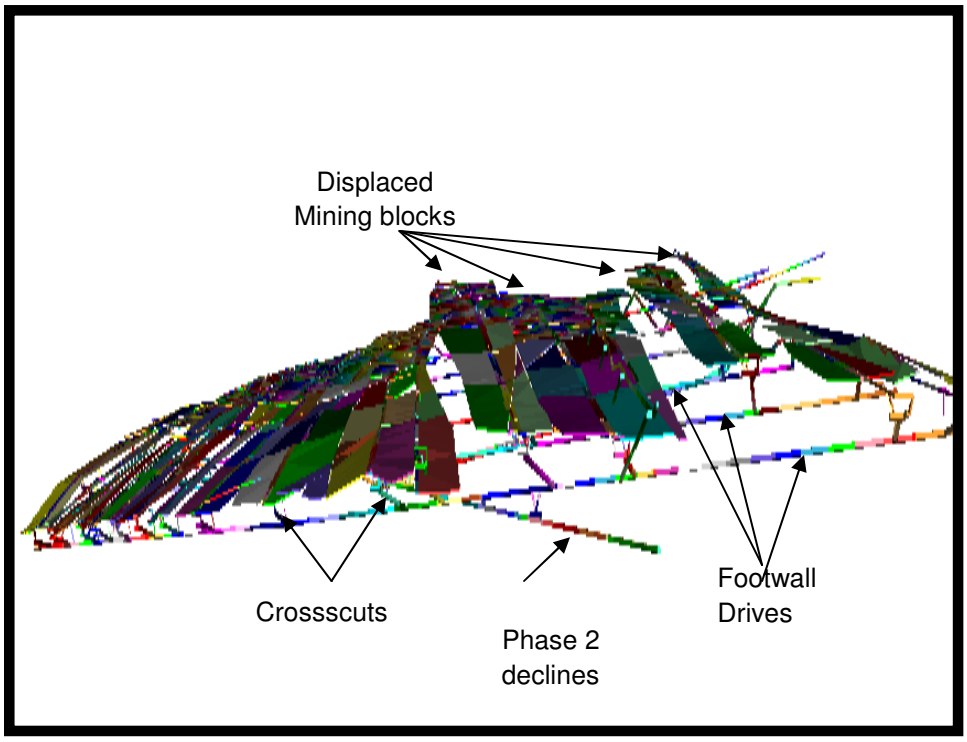


Figure 4 : A 3D illustration of the project area in Mine2-4D®

Figures 2, 3 and 4 show different perspective views that obviate the complexity of the geological structures in this area. In Figure 2, raise line No. 36 which is the area to the extreme right of the picture is demarcated by faults to the left and right of the raise line with only 138 000 tonnes of mineable ore. The area does not belong to any project area but was included in the current project because the main infrastructure will be developed connecting project areas on either side of the raise line and the area can easily be mined from the infrastructure to be put in. Reserves in raiseline No. 36 will however have to pay for the infrastructure that will be required further to connect it with the main infrastructure. The 138 000 tonnes have been included in the assessment at the end of the life of the project to avoid skewing of the results by adding the infrastructure and tonnes from raiseline No. 36. Figures 2 and 3 show a horst structure that is situated towards the middle of the project area. The horst has been thrown up ±40 and ±20m on the left and right hand sides of the structure respectively. A high resolution scheduling was done down to panel and crew level. The project area is divided into two areas:

- Area 1 (Top 3 Levels) – Small blocks subject of this investigation referred to as Phase 1
- Area 2 (Bottom 3 Levels) – Big blocks referred to as Phase 2.

2.1. GEOLOGICAL SETTING AND STRATIGRAPHY.

The investigation is for the mining of the Upper Critical Zone (UCZ) of the western limb of the Bushveld Complex. The Bushveld Complex is a layered intrusive complex. Layering has been caused by repetitive influxes of fresh magma, followed by magmatic processes such as pressure variations, magma mixing, crystallisation, fractionation and crystal settling. Layering is continuous and can be followed over large distances along strike and dip. The Bushveld Complex is subdivided into a number of zones, of which the Upper Critical Zone (UCZ) is of economic significance in terms of Platinum Group Elements (PGE) and Chrome mineralisation.

The layering consists primarily of pyroxenites, norites and anorthosites with occasional chromitites and harzburgites. Within the UCZ a number of groups (Lower, Middle and Upper) are recognised. Important markers within the layering are the Upper Group 1 (UG1) and the Upper Group 2 (UG2) Chromitite layers, the Merensky Reef and the Bastard Pyroxenite. The economic PGE mineralisation occurs within the UG2 and Merensky Reef.

Figure 5 illustrates the general stratigraphy of the project area and three neighbouring shafts.

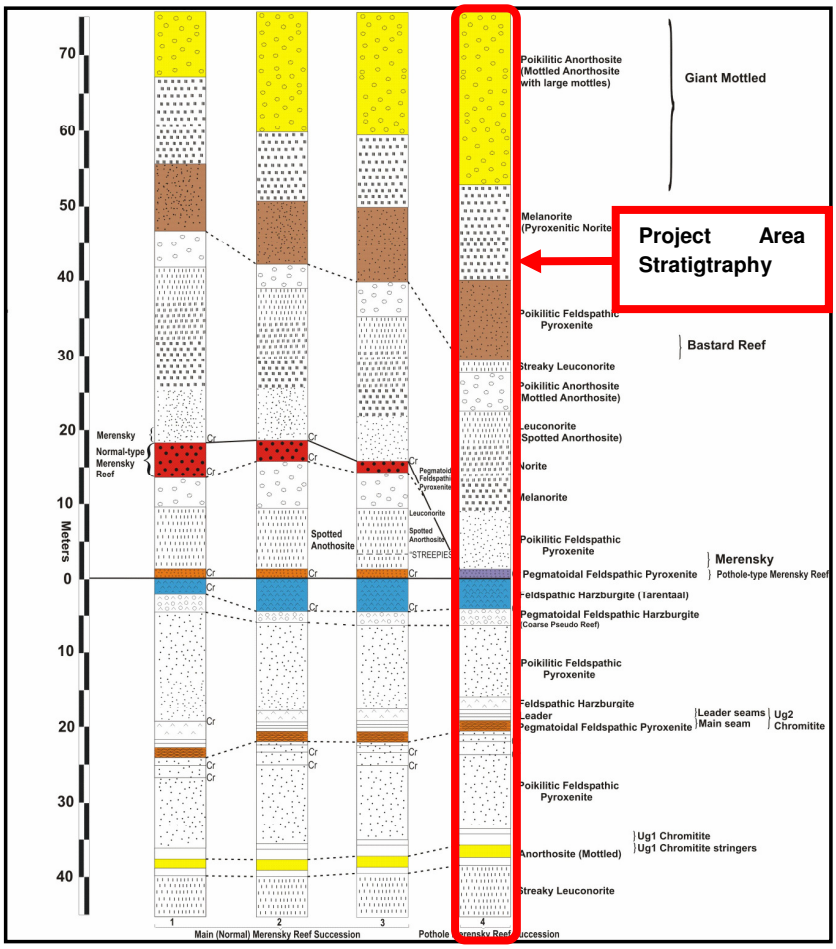


Figure 5 : General stratigraphic column of project area and four adjacent shafts

(Courtesy of Anglo Platinum)

2.2. BACKLENGTH PLANNING

The backlength has been optimised taking into account the scraper winches that the company has decided to standardise on. The standardisation was informed by optimal achievable scraping distances juxtaposed against the safety of employees considering visibility in the line of sight of scraping.

The project in question was planned to use scraper winches with 55 kW [75 hp] double drum electric motors in both their advanced strike gullies (ASGs) and the centre raises (Table 2). The rope used in both the above mentioned excavations is a 20mm diameter six strand wire rope with a rated pulling capacity of 235m. A 1.5 safety factor resulted in a maximum pull distance of 156m which was rounded off to 160m. The distance was also found to be acceptable in ensuring longer life of the rope, ensure shorter cycle times and to facilitate quicker tandem scraping. In both the advanced strike gullies and the centre raises, a Funky scoop is used in tandem with a Diamond scraper to ensure delivery of higher tonnage per cycle of scraping.

Smaller 37.5 kW [50 hp] scraper winches (Table 2), are used for scraping on the stope panel face. Rock is scraped down on dip into the advanced scraper gullies from where it is scraped along strike by a 55kW scraper winch to the centre raise. Rock is then scraped down on dip in the centre raise with another 55 kW scraper winch to the orepass.

Table 2 : Scraper winch specifications (Courtesy Joy Sullivan Mining)

					Dimensions				
Motor	rpm rating	Max Rope Pull in [T]	Mean Rope Speed [m/s]	Max Rope Speed [m/s]	Length [m]	Height [m]	Width [m]	Winch Mass [kg]	Motor Mass [kg]
LF 230 Joy Sullivan 22 kW	1400	2.14	1.2	1.4	1.96	0.950	0.720	1320	220
LF 230 Joy Sullivan 37 kW	1400	3.56	1.2	1.4	1.96	0.950	0.720	1320	340
JCF 212 Joy Sullivan 55 kW	960	7.03	0.92	1.35	2.025	1.124	0.914	2110	420
JCF 212 Joy Sullivan 75 kW	960	8.79	0.92	1.35	2.025	1.124	0.914	2110	580

Table 3 : Ropes in use versus scraping distances for scraper winches (Courtesy Joy Sullivan Mining)

	Rope Diameter			
	18 mm	20 mm	22 mm	26 mm
Scraper Winch	Pull Capacities [m]			
LF 230 Joy Sullivan 22 kW	185m	115m	105m	70m
LF 230 Joy Sullivan 37 kW	230m	140m	130m	90m
JCF 212 Joy Sullivan 55 kW	375m	235m	200m	130m
JCF 212 Joy Sullivan 75 kW	470m	295m	250m	165m

In Tables 2 and 3, the green shaded blocks show the selected options for combinations of scraper winches, ropes used and the resultant rated pulling distances.

2.3. PANELS LAYOUT

Panels were designed at 38m centres along dip. Provision for a maximum scraping distance of 160m resulted in 228m backlength which was found to be optimal. The movement of ore within the production panel and ultimately to the raiseline boxhole is shown in Figure 6.

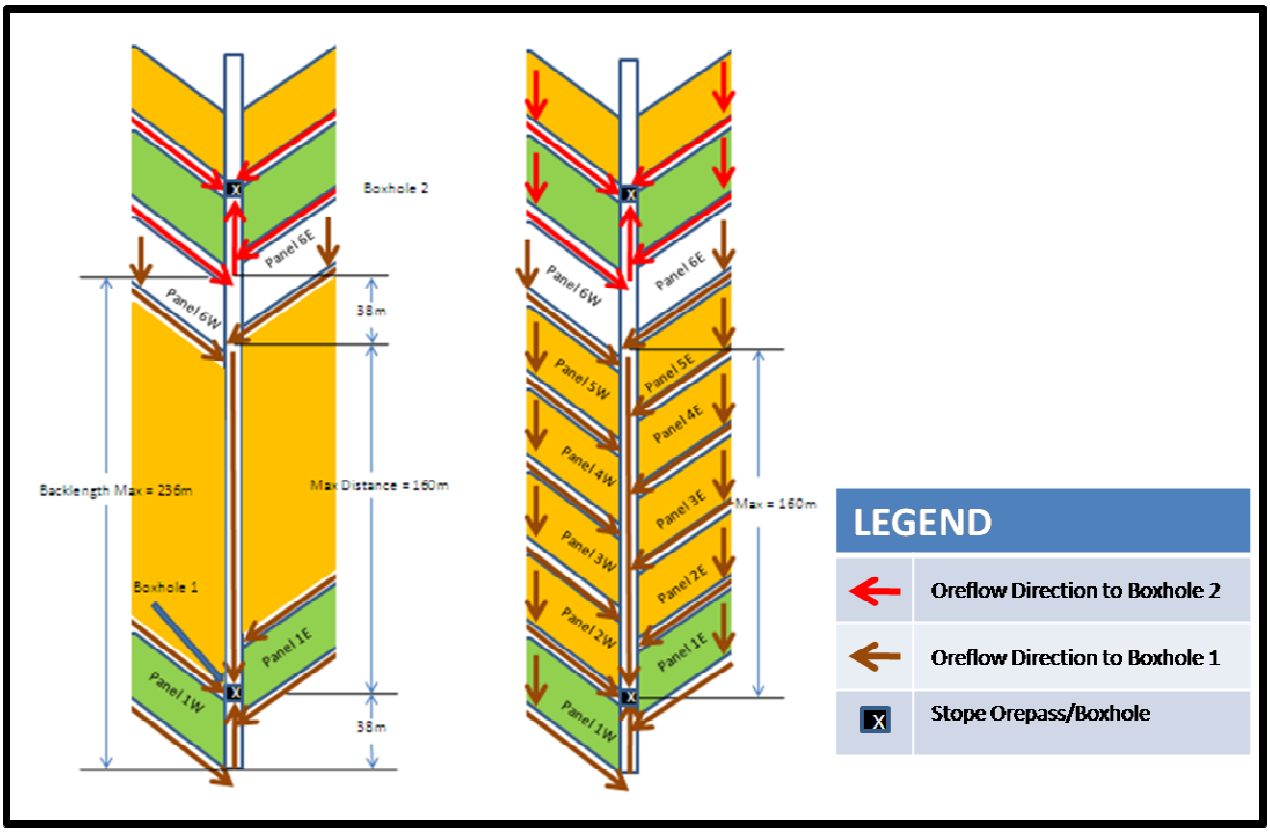


Figure 6 : Oreflow direction in production panels

Based on the maximum scraper winch reach adjusted for the safety factor, the backlengths were planned as shown in Figure 7.

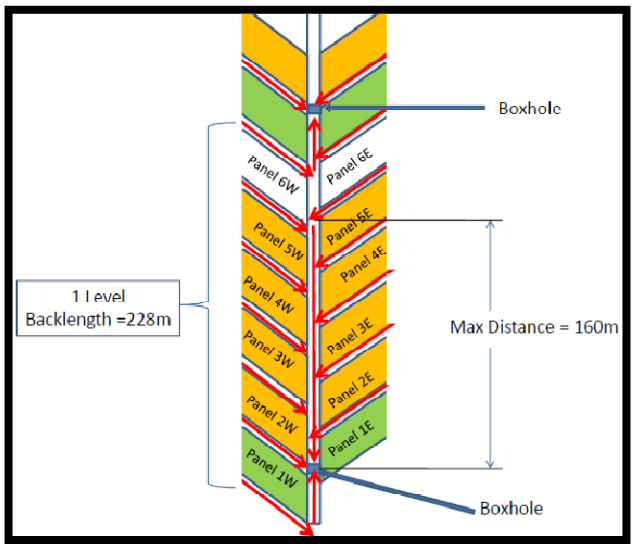


Figure 7 : Backlength planning for a raiseline

The maximum scraping distance is limited to 160m as indicated in Figure 7. Blasted ore from the green coloured panels is scraped up-dip from the toe of the panel. Blasted ore from the white coloured panels is scraped down-dip from the toe of the panel, meaning that there is no need for the centre raise scraper to scrape beyond the toe of the white coloured panels. The Advanced Strike Gully is at the toe of the panel and this is the take-over point of the centre raise scraper winch.

2.4. ACCESS STRATEGY

The project area will be accessed from a neighbouring decline area meaning that there will be no primary access for the area in question and the neighbouring area's primary access will be used for this area. The neighbouring areas primary access is a decline shaft cluster system that is placed at an apparent dip. The decline cluster consists of three barrels for men, materials and ore conveyance. The men and materials declines flank the conveyor decline as shown in Figure 8. The decline shaft cluster was placed at apparent dip to reduce the inclination from 17^0 which is the inclination of the UG2 layer to an inclination of 9^0 which is favourable inclination for trackless machinery. Currently, mining on the existing neighbouring project is carried out on-reef due to favourable geological conditions.

The project area is characterized by regular faults with large vertical displacements of up to 40m which necessitated that the main infrastructure be carried off-reef, in the footwall of the UG2 chromitite layer.

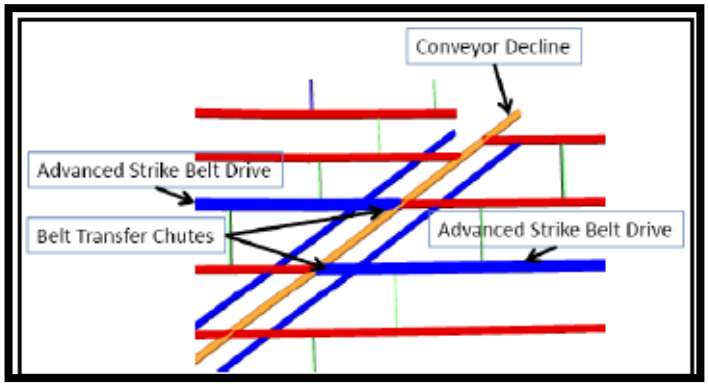


Figure 8 : Neighbouring project declines arrangement

Mining in the neighbouring project area is carried out on-reef. Advanced Strike Drives (ASD's) are mined out of the decline shaft and are kept on reef along strike. Two different types of ASDs are developed namely the normal ASDs and the Belt ASDs.

The Belt ASDs on either side of the decline cluster are slightly off-set in order to accommodate the large transfer points which transfer ore from the conveyor belt in the Belt ASD onto the conveyor belt in the conveyor decline. Every third drive is a belt drive, meaning that there will always be two ASD's being cleaned into one Belt ASD.

The top four levels of the project area are planned to be accessed by haulages developed from the current project area's ASDs. These haulages will be developed by trackless means and will be developed steeply at 12 degrees to quickly gain depth to the edge of the project area. Footwall drives will be located 35m below the UG2 layer as described in the following section.

2.5. PLACEMENT OF THE FOOTWALL DRIVES

Footwall drives could not be placed on-reef due to the bad geological conditions encountered in the project area. The drives could either be placed in the competent formation of the Poikilitic Feldspathic Pyroxenite between the UG2 and the UG1 Layers (Figure 5). The middling between the layers is only 18m resulting in a effective 14m skin to skin middling between the UG2 layer and the 4m high footwall drive. It was decided, from the rock engineering point of view, that this would not be a suitable place for the footwall drives. This decision was made because if any slight variation in stratigraphy could occur then the drive could go through the UG1 horizon or the chromitite stringers (triplets) which are within 5m below the UG2 and within the 14m effective middling between the UG2 and UG1 horizons.

Footwall drives were placed in the Streaky Leuconite that is situated further below the UG1 Chromitite layer. A distance of 35m below the UG2 layer was calculated taking into account the following:

- 18m middling between the UG2 and the UG1 Layers,
- UG1, chromitite stringers and Mottled anorthosite account for an additional 5m, and
- the drives are then placed 12m skin to skin below the UG1 Chromitite layer.

Crosscuts were designed at a 90^0 angle to the footwall drives towards the reef horizon. The 90^0 angle of intersection between the crosscuts and haulages was necessitated by conveyor transfer points, which ideally, should be constructed in such a way that the belts are placed at right angles. Where there was sufficient length (more than 37m), the crosscut was divided into two sections. The first section was designed to be excavated to the size of the footwall drive and intersect the

footwall drive at right angles. The second portion of the crosscut was designed to be 37m long and was designed to run parallel to the raise and stop 8m below the UG2 layer. The main crosscut out of the footwall drive (before the kink) was designed to be equipped with a conveyor belt as with the footwall drives. These excavations were kept straight to accommodate the conveyor belts.

2.6. GENERAL ARRANGEMENTS

2.6.1. Funkhole

A funkhole was planned to be developed perpendicular to the second piece of the crosscut. The centre of the funkhole would be developed 15m from the end of the 37m second crosscut. Rock would be scraped into this funkhole until a permanent orepass, that would be located in the first part of the crosscut is in place and ready to be used. The permanent orepass was designed in such a way that it would be equipped with a "spillminator" that would discharge rock directly onto the conveyor belt. Rock from the funkhole would be dumped onto the conveyor belt by a Load Haul Dump (LHD) machine.

2.6.2. Travelling way

A travelling way would be developed at the end of the crosscut at a maximum inclination of 34.5^0 to intersect the UG2 layer. A scraper winch would be located at the bottom of the travelling way to facilitate cleaning until the raise is developed beyond the funkhole orepass. Ore scraped down the travelling way would also be dumped into the conveyor belt by an LHD. The scraper winch would be moved up into the reef horizon after the raise had been developed 10m beyond the funkhole orepass. The funkhole orepass would not be equipped. Rock scraped into this orepass would be allowed to fall under gravity to the floor of the funkhole from where it would be trammed to the conveyor belt by an LHD. Access to the funkhole would be limited to the LHD to prevent injuries to personnel due to falling rock.

2.6.3. Diagonal scraping

To prevent water from getting into the orepass, a small diagonal was designed in and would be equipped with a scraper winch. The diagonal would be developed in a direction 25^0 above strike for dewatering and an orepass would be located at the end of the scraper diagonal. The scraper diagonals' length would be a maximum of 30m and would be positioned in such a way that they link the raise to the slightly off-set orepass, off-set from the raise. Where the total length of a crosscut is shorter than 35m which is the length of the last part of the crosscut, the orepass would be developed to discharge directly onto the conveyor belt in the footwall drive and a conveyor belt would not be constructed in the crosscut.

2.6.4. Top level access drive

The project area comprises of three main levels from which production would be accounted for. There is however a top level which has been designed in for ventilation requirements. Ore that would be mined from the top level would be in small quantities and would not justify ore handling facilities on this level. The top access drive would be equipped with a conveyor belt for half the distance of the drive. This is due to the fact that the amount of rock that would be handled in the last part of the drive would not justify the expense of fitting in a conveyor belt for the full length. Trucks would therefore be used to tram rock to the conveyor. The dimensions of the top access drive would be reduced where a conveyor belt is not fitted. The top access drive would also serve as a Return Air Way (RAW).

2.7. RESOURCE TO RESERVE CONVERSION FACTORS

Panels were laid out at 38m centers. The span was determined as optimal from rock engineering modeling. Panels were designed in to improve the accuracy and resources were planned down to crew level. Figure 9 was used in determining factors for in-stope pillar losses, ASGs, ventilation holings and as a result determine the mining extraction per panel which was subsequently applied in the resource to reserve conversion.

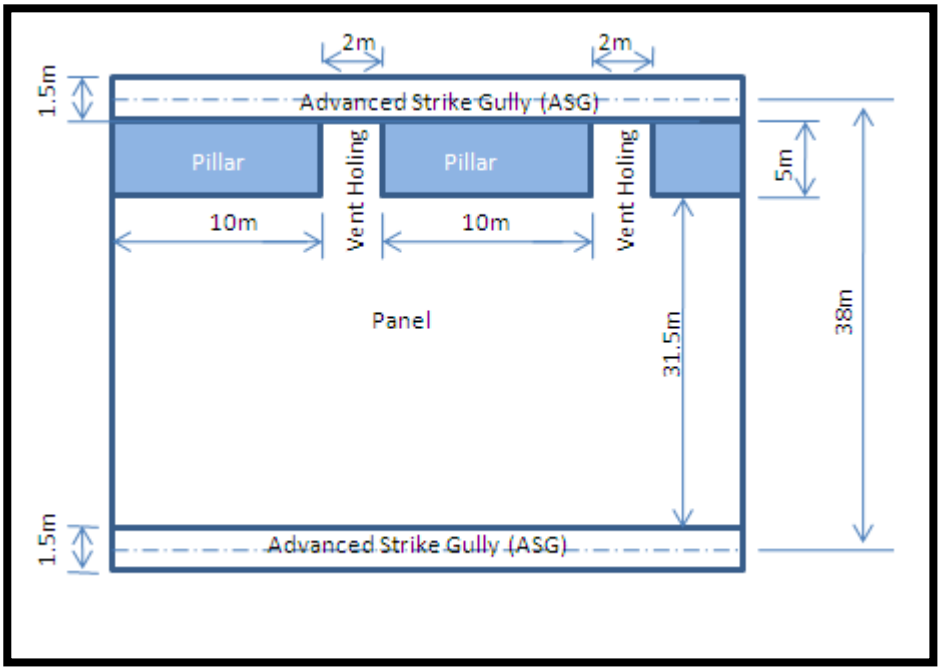


Figure 9 : Panels layout – rectangular sketch for simplicity

2.7.1. Derivation

For every 12m advanced there would be a pillar left and a vent holing mined, therefore this was considered to be reflective of the total on-reef mining and as a result the derivation was based on this piece of ground.

AREA		DERIVED PERCENTAGE
Total Area	= 38m X 12m	
	= 456 m^2	100%
Ventilation Holing	= 2m X 5m	
	= 10 m^2	2.193%
Advance Strike Gulley	= 1.5m X 1.2m	
	= 18 m^2	3.948%
In-Stope Pillar	= 10m X 5m	
	= 50 m^2	10.965%
Panel	= 31.5m X 12m	
	= 378 m^2	82.895%

The factors derived above have been applied to spare panels which are panels in the top access level that falls within the project boundary above the first mining level. If the spare panels were to be mined from the first mining level, the backlengths on the top level would exceed the maximum

specified and these panels would be mined from the top access drive. Dilution calculations were performed in Earthworks Production Scheduling taking into account the channel width of 90cm and the mining width of 149cm.

2.8. SCHEDULING CRITERIA

Scheduling was done to closely represent the project execution environment. As stated above, the project area would be accessed from a neighbouring shaft area by extending the Advanced Strike Drives into footwall drives that would be located off-reef. The development of the footwall drives and associated development was kept at best practice rates as benchmarked through the particular company and the stoping rates were varied between 10m and 20m. The impact of variable stoping rates was then analysed.

2.8.1. Scheduling Principles

The following principles guided the scheduling process:

- There would be 6 development crews. Each flat end would have a dedicated crew while the other two crews would be used in the declines.
- Each crew would stop after developing 10m beyond a crosscut to complete all the flat development coupled with the raise line before continuing with the main development. Excavations to be completed before continuing with the main flat development are the crosscut, crosscut extension and funkhole. After the travelling way position is achieved, the main development crew would go back to the flat end that would be 10m beyond the crosscut and continue with development to the next raise line.
- Belt extensions would be done every 150m. A delay of 1 month was incorporated, a delay in which the belt would be extended and an underpass created.
- Development in the raise is stopped while the on-reef scraper diagonal is being blasted.
- Ledging only starts after a conveyor belt is installed in the crosscut below.

- On every level, there would be a total of 6 stoping crews and 1 ledging crew. Stoping crews would be divided into three per raise line. Two raises would be stoped while the third raise line is being ledged.
- Three levels have been modeled since steady state is planned to be achieved from these levels to form the complete Phase1 of this project.
- The travelling way would be developed 10m in before a scraper winch can be installed at the bottom of the travelling way. A delay of 5 days was built in for the construction of a winch bed.
- The raise is scheduled to go 10m beyond the scraper diagonal before commencing with the development of the scraper diagonal. The scraper diagonal and box cause a delay of 1 month. The raise is stopped for 1m while work is carried out in the diagonal.
- After ledging, a delay of 2 weeks is incurred before mining commences due to conversion from temporary to permanent services.
- A 5-day delay is incurred in moving the winch from the foot of the travelling way into the reef horizon.

The best-practice rates in the local South African platinum mining industry are show in Table 4. These were the rates used in the scheduling process.

Table 4 : Scheduling parameters

Description	Value
Footwall Drives	40 m/month
Dropraise	15 m/month
Travelling Way	30 m/month
Stepover	30 m/month
Raise	30 m/month
Ledging	380 m^2/month
Stoping	400 m^2/month
Crosscut	30 m/month
Funkhole	30 m/month
Scraper Diagonal	30 m/month

3. RESEARCH RESULTS

3.1. ANALYSIS OF RESERVES AVAILABLE

Holing dates for different raiselines and the reserves to be mined in these raiselines have been tabulated in Section A-4 in [Appendix A]. Table 5 contains a summary of the results in Section A-1 of Appendix A. The results include only tonnes to be mined from the mining blocks and the spare panels. Tonnes from raises and ASGs have been left out because they are immaterial since they comprise less than 3% of the total tonnes and do not determine the life of a raiseline. Raise and ASG tonnes have however been included in the main analysis below.

Table 5 : Reserves statement summary

Raise no	Mining blocks tonnes	Spare panels tonnes	Total	Average Life months
36	138,381	22,708	161,089	7
37	314,467	62,484	376,951	17
38	566,683	18,050	584,733	26
38A	331,874	-	331,874	15
39	530,925	17,171	548,096	25
40	353,863	5,306	359,169	16
41	361,672	14,620	376,293	17
42	448,617	548	449,166	20
42A	246,407	-	246,407	11
43	277,720	12,583	290,303	13
44	489,105	3,447	492,552	22
45	151,719	-	151,719	7
Total	**4,211,434**	**156,918**	**4,368,352**	
Total reserve			**4,368,352**	

The average life of a raiseline was determined from the principle, stated in the scheduling criteria, that only three crews are allowed to work in a raiseline on one level. Each team would mine 400 m^2 per month which is translated to tonnes by a factor of 6.21 tonnes per m^2. Nine crews in a raiseline would mine a maximum of 22 356 tonnes per month in total for all the three levels.

Table 6 shows the spread of total mineable tonnes and the mining duration of a raiseline in months. The tonnes range from 151 000t to 364 000t. Mining duration ranges from 7 months to 26 months with an average of 16 months.

Table 6 : Vital statistics on reserves statement per raiseline

	Tonnes	Life [months]
Minimum	151,719	7
Maximum	584,733	26
Average	364,029	16

The results in Table 6 indicate an average life of raiseline of 16 months while the time it takes to develop one raiseline is more than 24 months. The analysis above is based on the total raiseline including all three levels on which mining would happen simultaneously.

A similar analysis was carried out on individual raiselines and levels. In this case, three teams would work in one raiseline on any particular level. Different raiselines have varying amounts of reserves on different levels and as a result have different mining durations. A histogram of the distribution of the different mining durations of the different raiselines is shown in Figure 10.

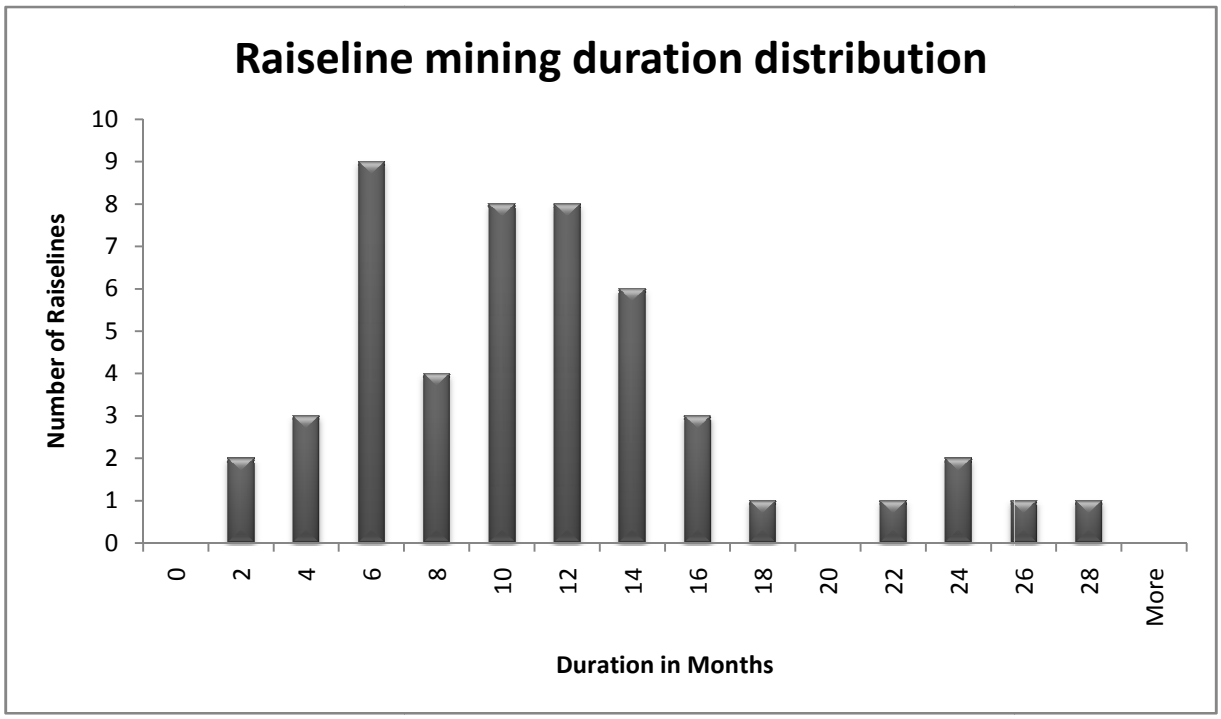


Figure 10: Histogram of mining durations of raiselines

Figure 10 shows that the distribution is negatively skewed and the bulk of the figures are less than 18 months. An average mining duration of 12 months was calculated as shown in Table 7.

Table 7: Statistics on individual raiseline mining duration analysis

	Tons	Life
Minimum	4,602	1
Maximum	234,940	32
Average	85,948	12

Table 7 shows the spread of total mineable tonnes and the mining duration of a raiseline in months. The tonnes range from 4 600t to 235 000t. Mining duration ranges from 1 month to 32 months with an average of 12 months. The analyses above show some differences in all

parameters, however small. In the case where the total raiseline was analysed, the mining duration ranged from 7 to 26 months and in the case were raiselines were analysed per level the duration ranged from 1 to 32 months.

Six crews will be operational on each level in two adjacent raiselines, divided into three crews per raiseline. From Tables 6 and 7 it is evident that by the time the second raiseline is established for the second set of crews to start mining, the first raiseline will be almost or totally mined out necessitating the first set of 3 crews to be idle if it takes more than 18 months to develop a raiseline and an average of 12 months to mine it out. It is therefore evident that the amount of reserves in each raiseline on each level in this project is not sufficient to motivate high production levels with the current development rates.

3.1.1. Analysis of available reserves conclusion

The analysis shows that the raiseline analysed per level has a minimum of 1 month, a maximum of 32 months and an average life of 12 months. This analysis shows that the amount of reserves per meter advanced is very low and as a result, high development rates are required to justify high production rates. Geological conditions in this project further complicate the application of rules of thumb and strengthen the argument that all projects should be evaluated and motivated on their own merit.

3.2. TONNAGE PROFILE

Figure 11 shows a graph of reserves available for mining yearly at the given development rates.

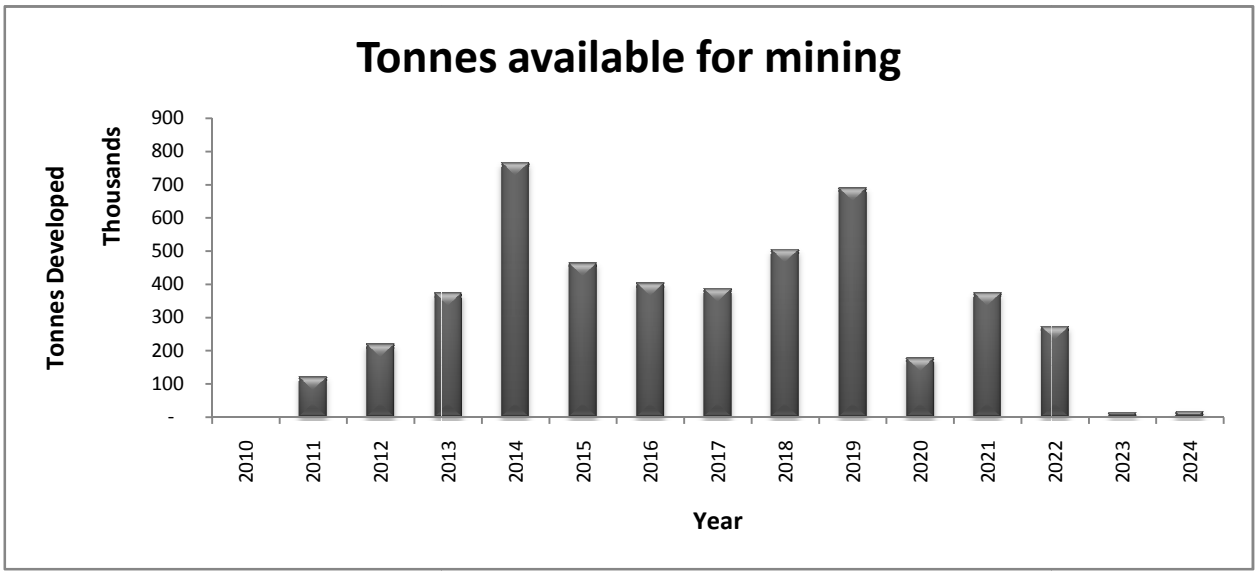


Figure 11: Reserves available for mining

Figure 11 shows that the maximum available reserves peak at 800 000 tonnes and therefore the planned production level of 100 000 tonnes per month which equates to 1 200 000 tonnes per annum is highly optimistic and not achievable. The profile shown in Figure 11 represents the total available reserves without any provision for flexibility. There should be some places developed and ready for mining to cater for temporary unavailability of planned mining areas due to safety or logistical constraints. If a mine is deployed to mine all available reserves as they become available, the result would be in an inefficient operation that would result from:

- temporary shutting down of mining areas due to safety or logistical constraint will result in underutilization of labour,
- labour will be left idle in years where fewer places are available for mining.

3.2.1. Face advance at 10m per month

Mining production was varied between 10m to 20m per month in increments of 1m per month face advance. However only 10m, 15m, and 20m face advance per month are illustrated in the report to show the trend identified. Raising, ASGs and ledging tonnes were brought into account in this analysis. The accessible reserves stated in the graphs in figures 12 onwards represent tonnes available and ready to be mined. These tonnes are the difference between tonnes carried over from the previous mining period, tonnes added in the measuring year less the tonnes that would be mined in that particular period.

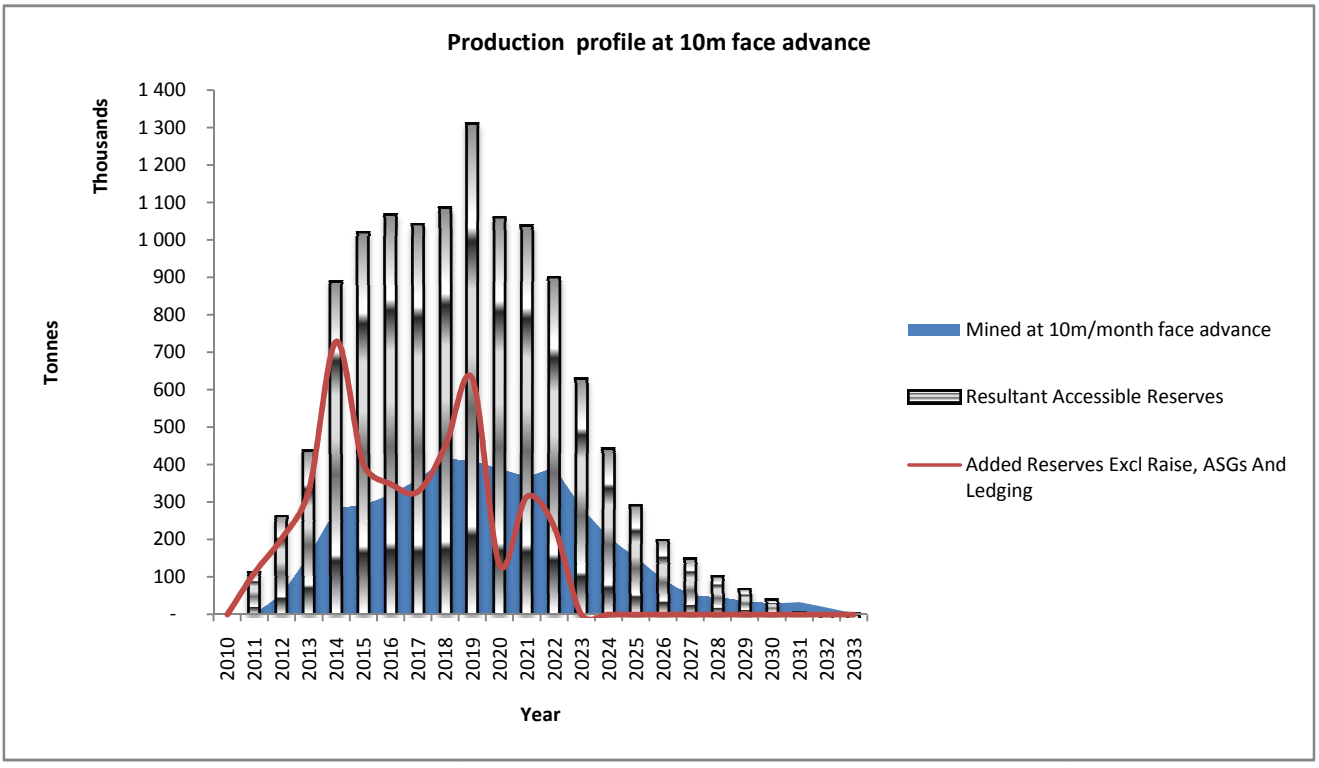


Figure 12 : Face advance of 10m per month

The production profile at 10m face advance in Figure 12 shows more available reserves than the recommended 1 or 2 years reserves. Production peaks at 400 000 tonnes per annum with a resultant life of mine of 24 years. The profile could be smoothed out and would result in a yearly production of 300 000 tonnes per annum sustained over a period of 13 years as shown in Figure 13. If a profile is smoothed at a higher mining rate, the mining duration at steady-state is shortened due to the inverted bell-shape of the production profile.

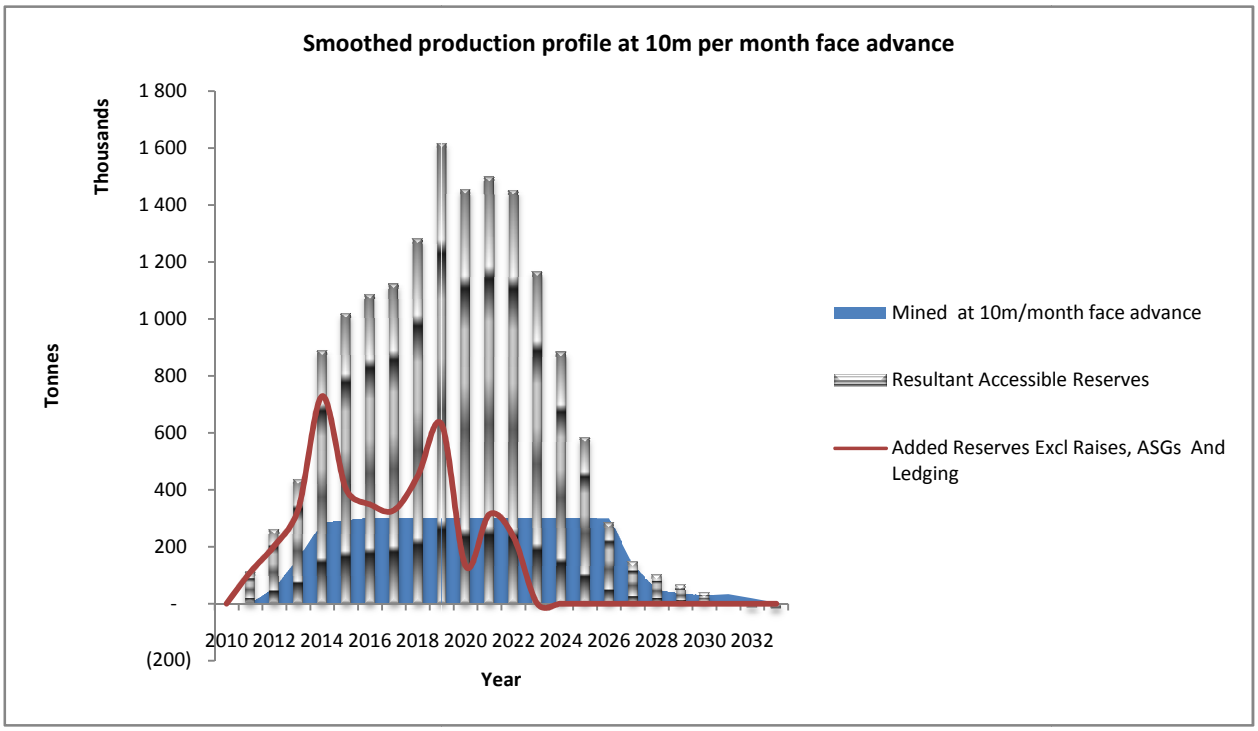


Figure 13 : Smoothed profile 10m per month face advance

The resultant accessible reserves increases as a result of the smoothing process. This is due to delaying production in years where production exceeds the required total tonnes. In the smoothing process, tonnes can be delayed to create flexibility because they are ready to be mined, however

tonnes cannot be moved forward because the mining areas are still not ready to be mined. In the case above, the resultant accessible reserves are, for the bigger portion of the mining duration, at a level of five years. An advance rate of 10m per month at the current development rates is not optimal and the situation could be improved by reducing the development rates or by increasing the mining rate.

3.2.2. Face advance at 15m per month

The production profile at 15m per month face advance in Figure 14 shows a substantial reduction in flexibility from the high level of 3 and 5 years to a level with the unsmoothed and smoothed production profiles, respectively. Available reserves are within the recommended 1 to 2 years ready to be mined reserves. Production peaks at 500 000 tonnes per annum with a resultant life of mine of 18 years.

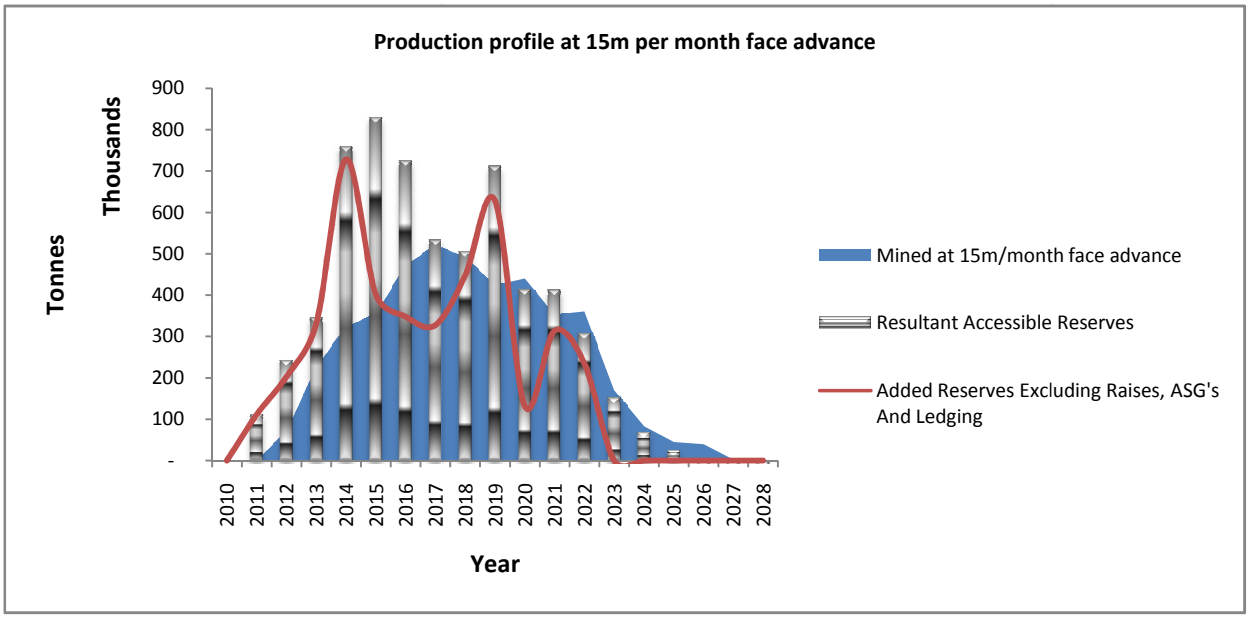


Figure 14 : Production profile at 15m per month face advance

The profile was smoothed out and resulted in a yearly production of 400 000 tonnes per annum sustained over a period of 18 years as shown in Figure 15.

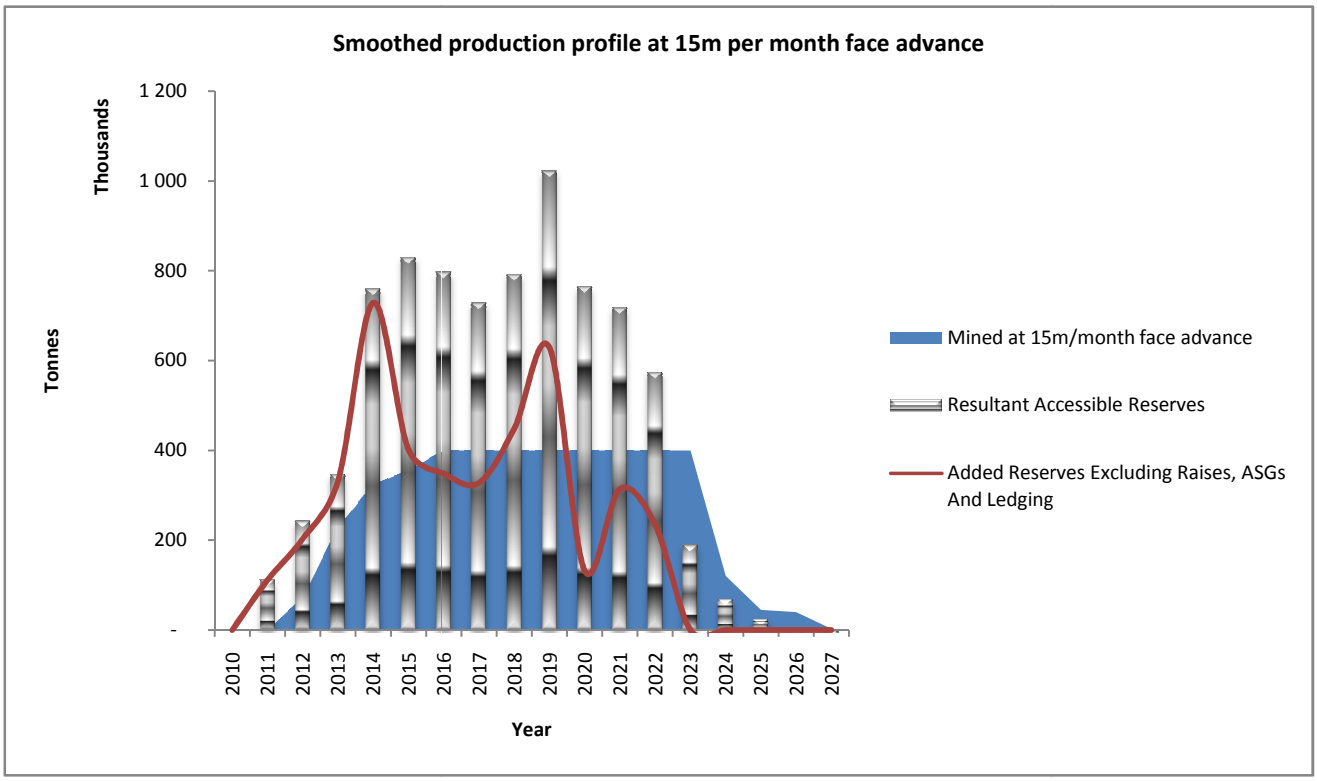


Figure 15 : Smoothed production profile at 15m per month face advance

An increase of 5m in face advance resulted in an addition of 100 000 tonnes per annum to the production profile. The profile in Figure 15 shows a balanced production profile with a stable production tempo and flexibility at the right levels. Flexibility towards the end of the project decreases but this is of no concern since the second phase of the project will kick in and flexibility will be provided in Phase 2. Phase 2 will be executed as a replacement project for phase 1, meaning that teams that run out of mining reserves will be re-located to Phase 2 area.

3.2.3. Face Advance at 20m per month

The production profile at 20m per month face advance in Figure 16 shows a further substantial reduction in flexibility to levels below the recommended 1 or 2 years. Production peaks at 600 000 tonnes per annum with a resultant life of mine of 15 years. The production profile is not ideal and sub-optimal. The 600 000 tonnes mark is reached in one of the 15 years life of mine of the project. Production would then drop down to 500 000 tonnes for the next two years after which there would be a steep decline in production in the remaining years.

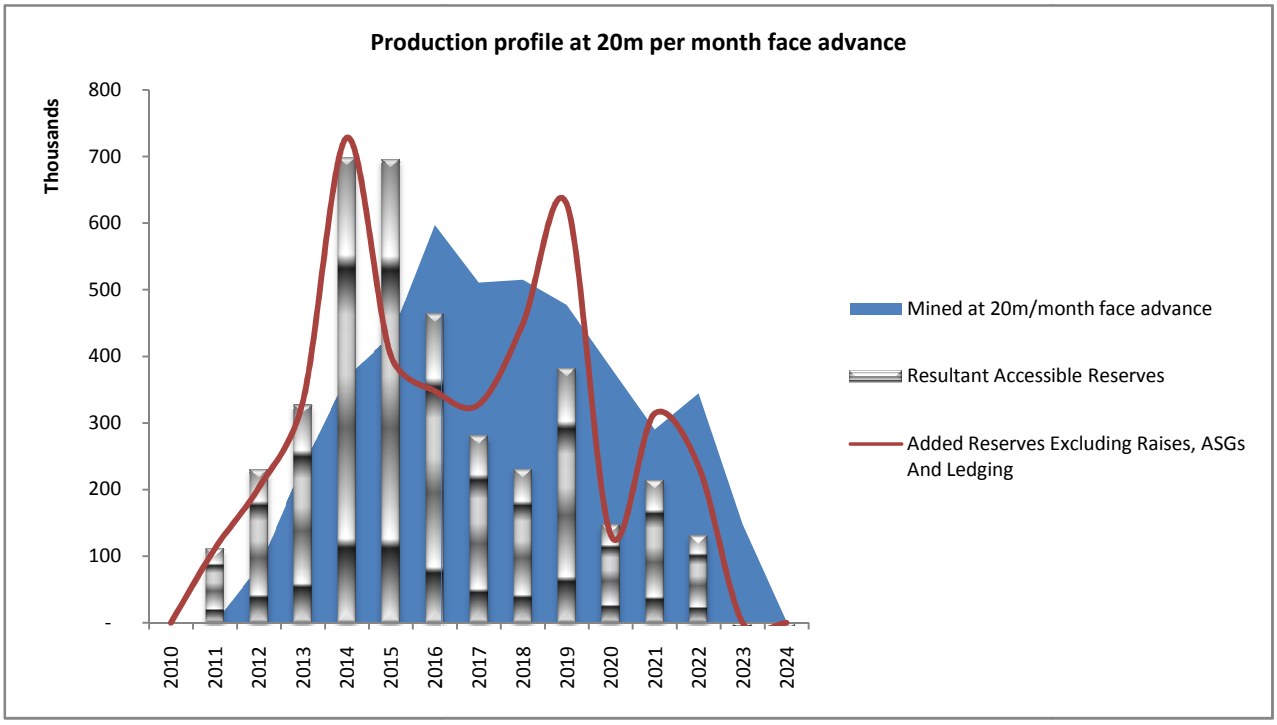


Figure 16 : Production profile at 20m per month face advance

The production profile was smoothed out and the resultant production profile is shown in Figure 17.

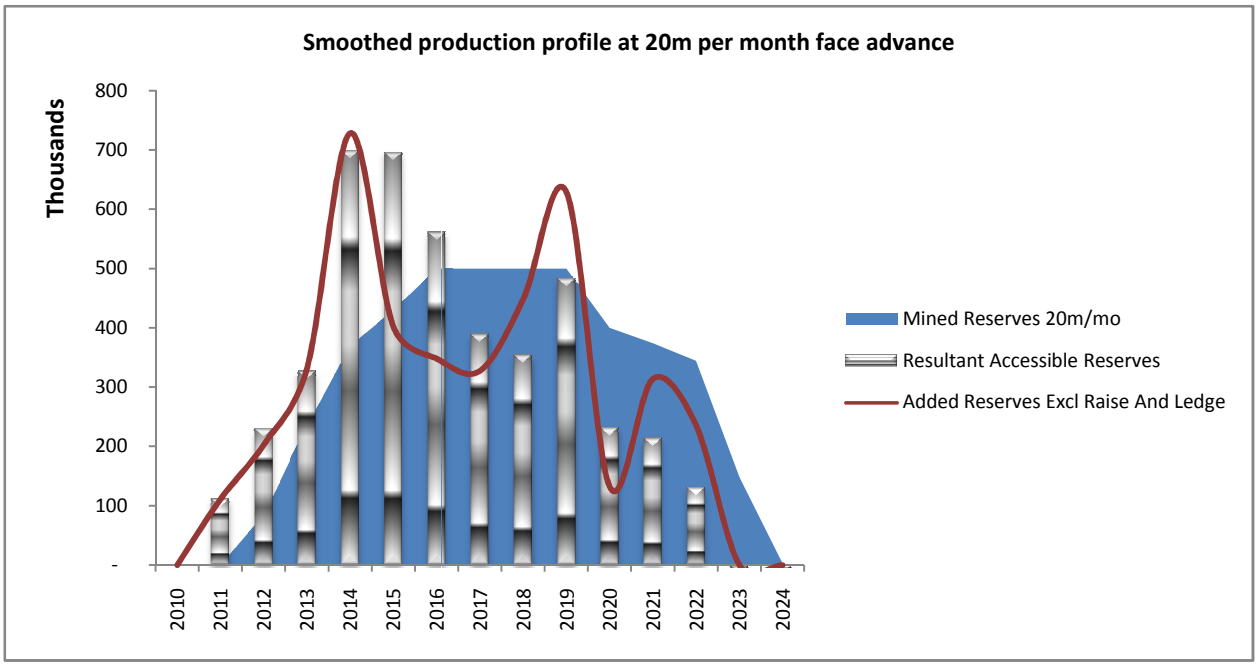


Figure 17 : Smoothed production profile at 20m per month face advance

A higher production level of 500 000 tonnes per annum is achieved, higher than the two cases where advances of 10m and 15m per month were used. The other two cases of 10m and 15m per month face advance peaked at production levels of 300 000 tonnes and 400 000 tonnes per annum, respectively. The higher production rate is however sustained only for a period of 4 years.

The profile was smoothed again at 450 000 tonnes per annum and resulted in a better profile as shown in Figure 18.

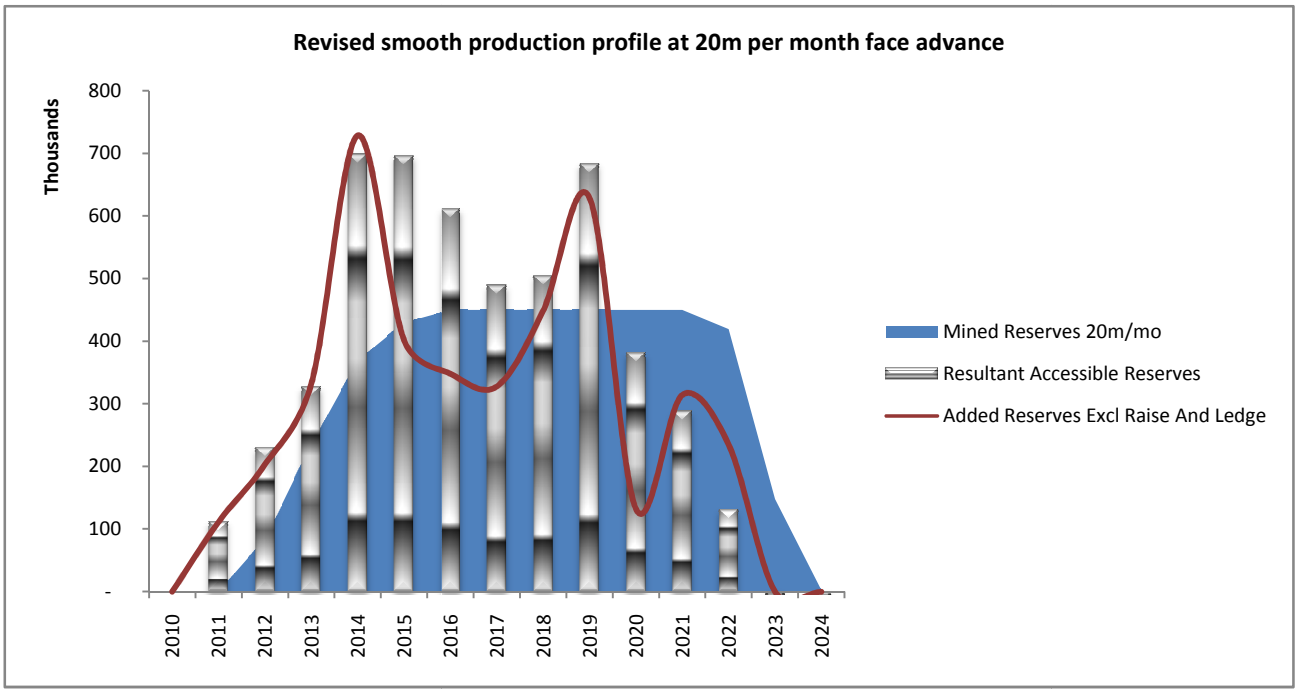


Figure 18 : Production profile smoothed at 450 000 tonnes and 20m face advance

The profile at 20m per month face advance with targeted production rate of 450 000 tonnes per annum resulted in the targeted production rate of 450 000 tonnes being sustained for a period of 8 years as shown if Figure 18. This production level resulted in available reserves of about 1 year in the first five years of steady state production and drops below the 1 year mark in the last 3 years.

3.2.4. Production profile conclusion

An advance rate of 10m per month was found to be sub-optimal because of too much reserves opened up while minimal mining took place. It is costly to keep ends and stopes open without mining activities because all excavations have to be periodically examined and made safe. Ventilation requirements also increase with the number of places having to be ventilated at all times.

An advance rate of 15m per month provides enough flexibility while a targeted production rate of 300 000 tonnes is achieved over a period of 13 years. An advance rate of 20m per month provides a desirable tonnes profile however available reserves flexibility was found to be marginal. This is an optimal production rate from the NPV point of view but not desirable from the flexibility point of view. It is therefore concluded that the 15m per month advance rate and a production target of 400 000 tonnes per annum is appropriate for this project. Individual crews' tonnage profiles are provided in Appendix A as follows:

- A-1 Individual crews' tonnage profiles at an advance rate of 10m per month
- A-2 Individual crews' tonnage profiles at an advance rate of 15m per month
- A-3 Individual crews' tonnage profiles at an advance rate of 20m per month.

The tonnage profiles for individual crews at an advance rate of 15m per month are more favourable than the generous profiles at 10m per month and the erratic profiles at 20m per month.

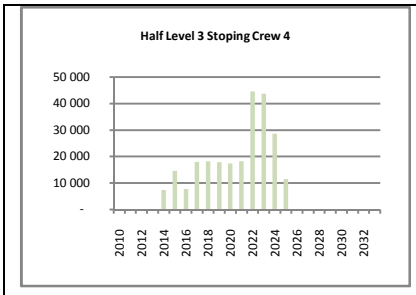
4. CONCLUSION

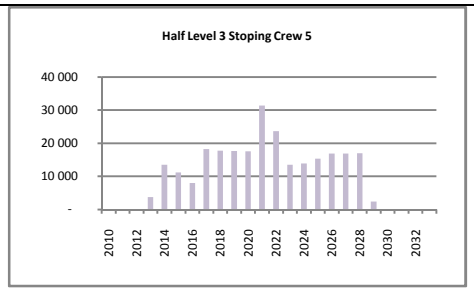
The planned production target of 100 000 tonnes per month is highly optimistic and not achievable. The production target that should be aimed for is between 300 000 and 450 000 tonnes per annum depending on the amount of flexibility wished for. An aggressive mining team would go for 450 000 tonnes while a conservative mining team would settle for 300 000 to 350 000 tonnes per annum.

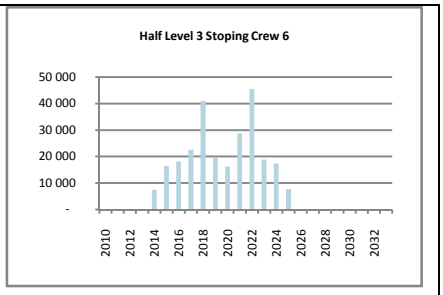
APPENDIX A

A-1. INDIVIDUAL CREWS PRODUCTION PROFILES: 10m PER MONTH ADVANCE RATE

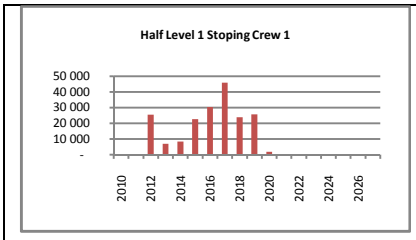


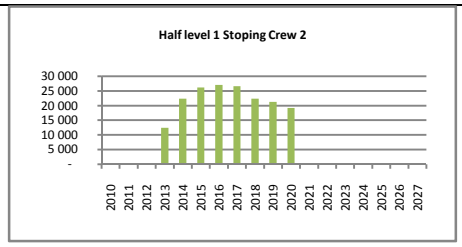

Half Level 3 Stoping Crew 4
50 000
40 000
30 000
20 000
10 000
-
2010
2012
2014
2016
2018
2020
2022
2024
2026
2028
2030
2032

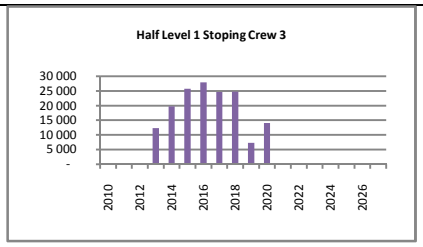

Half Level 3 Stoping Crew 5
40 000
30 000
20 000
10 000
-
2010
2012
2014
2016
2018
2020
2022
2024
2026
2028
2030
2032

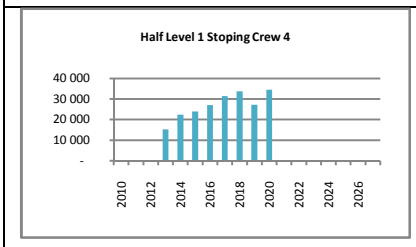

Half Level 3 Stoping Crew 6
50 000
40 000
30 000
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10 000
-
2010
2012
2014
2016
2018
2020
2022
2024
2026
2028
2030
2032

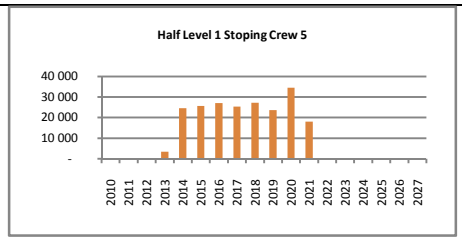
A-2. INDIVIDUAL CREWS PRODUCTION PROFILES: 15m PER MONTH ADVANCE RATE

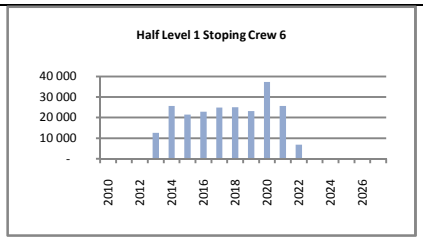


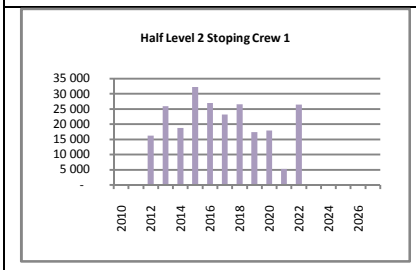


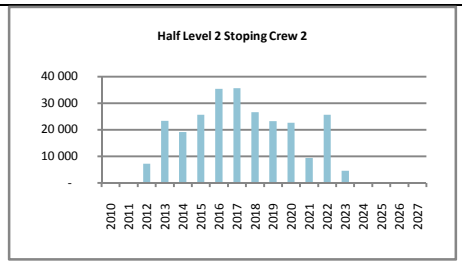


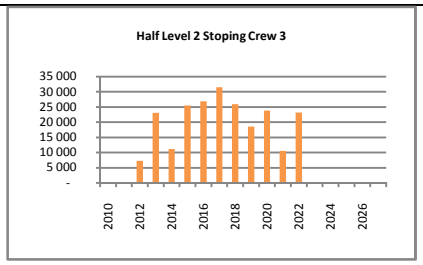


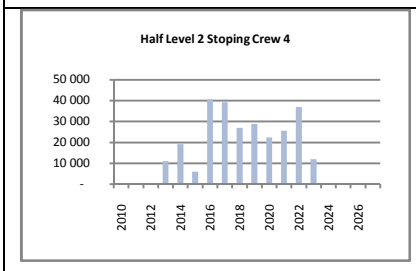


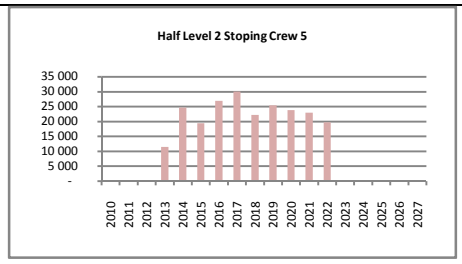


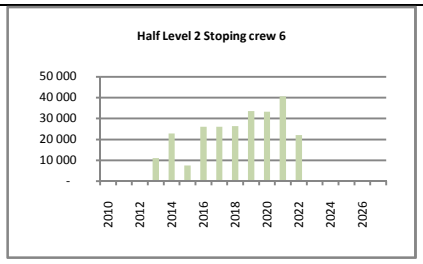


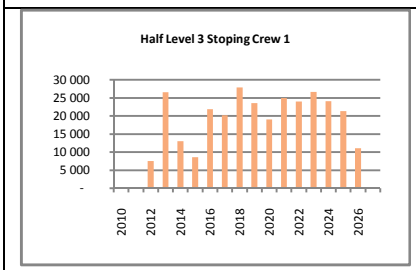


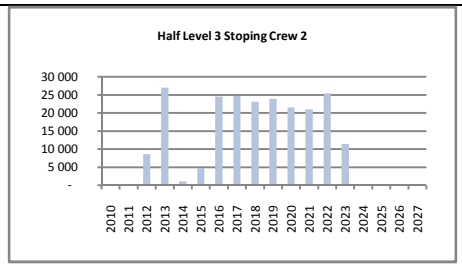


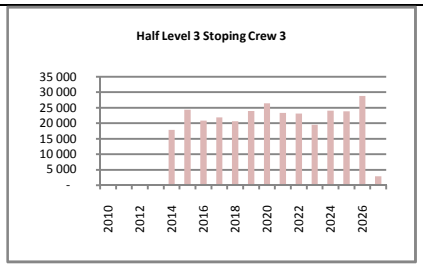


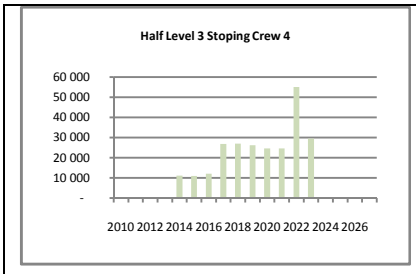


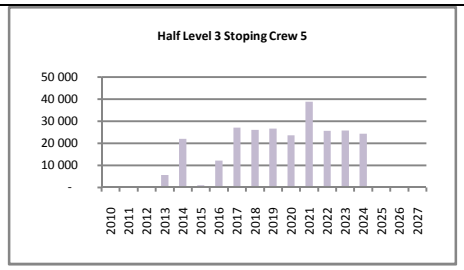


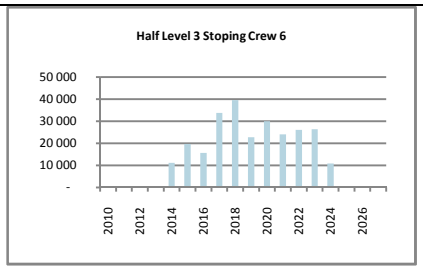




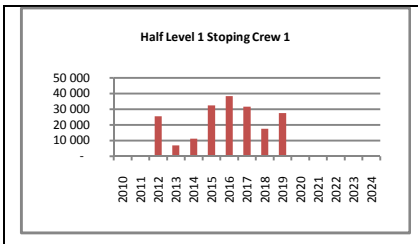


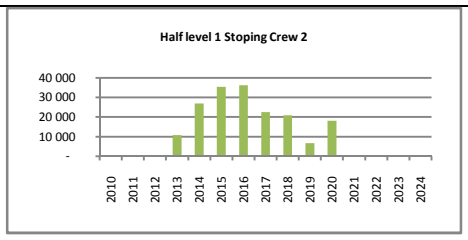

Half Level 3 Stoping Crew 4
60 000
50 000
40 000
30 000
20 000
10 000
-
2010 2012 2014 2016 2018 2020 2022 2024 2026

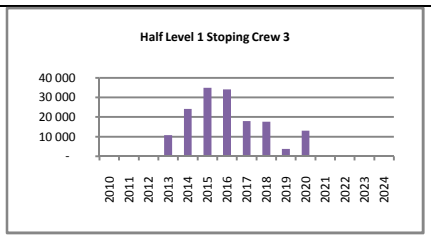

Half Level 3 Stoping Crew 5
50 000
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024 2025 2026 2027

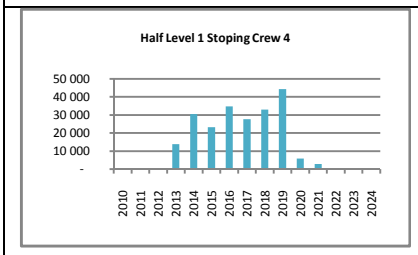

Half Level 3 Stoping Crew 6
50 000
40 000
30 000
20 000
10 000
-
2010 2012 2014 2016 2018 2020 2022 2024 2026

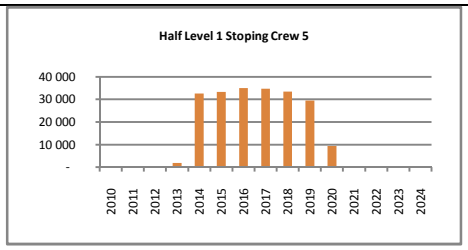
A-3. INDIVIDUAL CREWS PRODUCTION PROFILES : 20m PER MONTH ADVANCE RATE

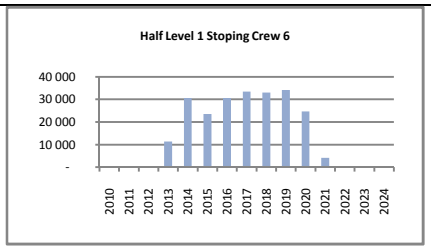


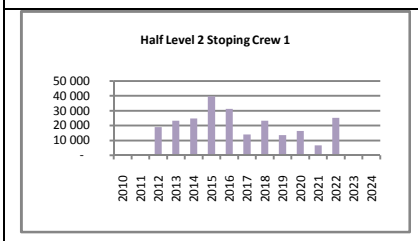


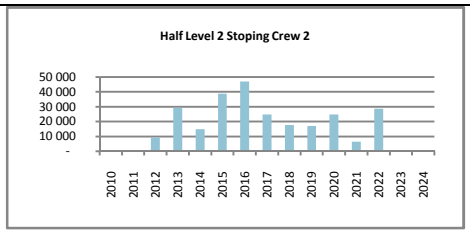


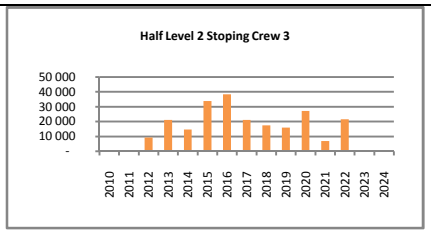


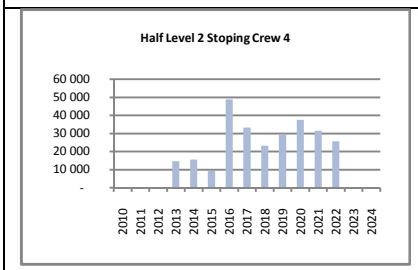


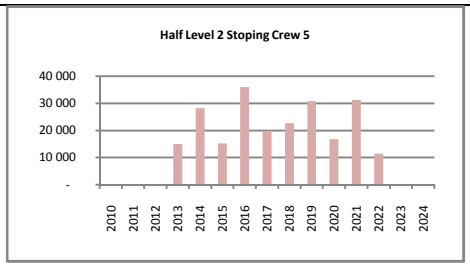


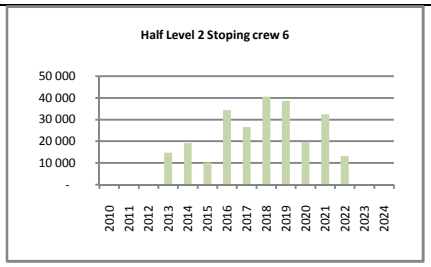


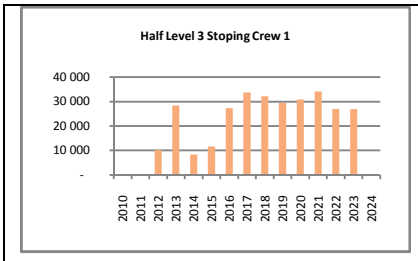


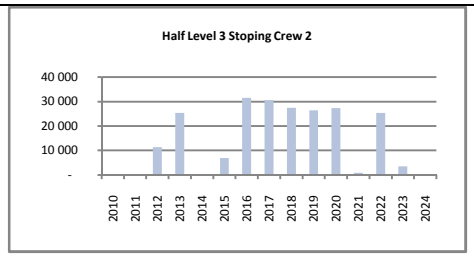


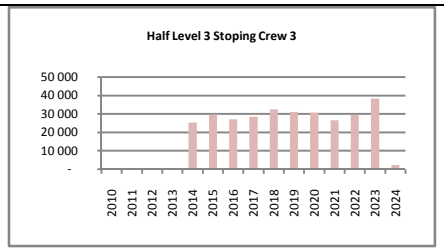


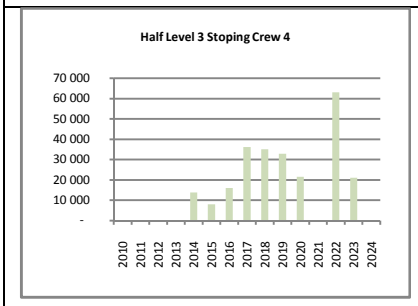


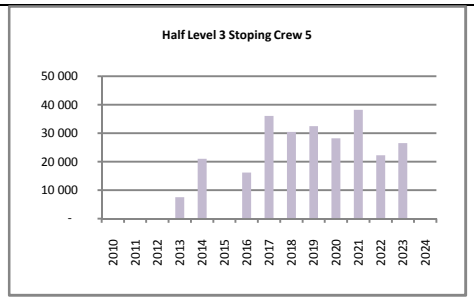


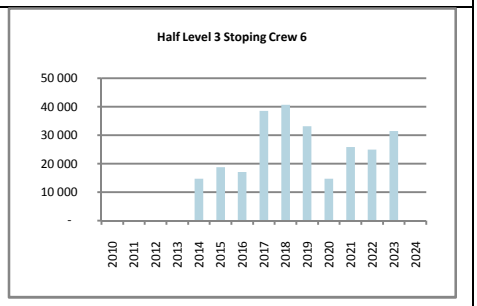

Half Level 3 Stoping Crew 1
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024


Half Level 3 Stoping Crew 2
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024


Half Level 3 Stoping Crew 3
50 000
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024


Half Level 3 Stoping Crew 4
70 000
60 000
50 000
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024


Half Level 3 Stoping Crew 5
50 000
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024


Half Level 3 Stoping Crew 6
50 000
40 000
30 000
20 000
10 000
-
2010 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021 2022 2023 2024

A-4. HOLING DATES AND RESERVES STATEMENT

Raise	Level	Stoping Start Date	Panel Tons	Spare Panels	Total Tons
36	Top Access Drive	26 May 2020	22,560	22,708	
36	Drive 1	25 June 2020	19,142		
36	Drive 2	17 March 2021	36,322		
36	Drive 3	26 February 2021	60,356		
		Raise 36 Sub-Total	**138,381**	**22,708**	**161,089**
37	Top Access Drive	11 November 2011	42,651	62,484	
37	Drive 1	23 October 2011	68,704	-	
37	Drive 2	10 May 2012	98,352	-	
37	Drive 3	21 June 2012	104,761	-	
		Raise 37 Sub-Total	**314,467**	**62,484**	**376,951**
38	Top Access Drive	27 April 2014	119,951	18,050	
38	Drive 1	31 March 2014	112,712	-	
38	Drive 2	31 March 2014	206,814	-	
38	Drive 3	08 March 2015	127,206	-	
		Raise 38 Sub-Total	**566,683**	**18,050**	**584,733**
38A	Top Access Drive	28 March 2013	109,807	-	
38A	Drive 1	09 May 2013	73,417	-	
38A	Drive 2	01 August 2013	96,973	-	
38A	Drive 3	02 October 2013	51,678	-	
		Raise 38A Sub-Total	**331,874**	**-**	**331,874**
39	Top Access Drive	01 March 2015	12,442	7,779	
39	Drive 1	08 August 2014	88,593	9,392	
39	Drive 2	26 August 2014	200,542		
39	Drive 3	04 May 2015	187,563		
39	Drive 3 Lower	24 July 2016	41,785		
		Raise 39 Sub-Total	**530,925**	**17,171**	**548,096**
40	Top Access Drive	24 September 2015	74,383	5,306	
40	Drive 1	13 March 2016	50,568		
40	Drive 2	30 March 2016	107,078		
40	Drive 3	06 February 2017	121,834		
		Raise 40 Sub-Total	**353,863**	**5,306**	**359,169**

Raise	Level	Stoping Start Date	Panel Tons	Spare Panels	Total Tons
41	Top Access Drive	17 May 2016	75,183	14,620	
41	Drive 1	17 May 2016	73,567		
41	Drive 2	16 April 2017	117,866		
41	Drive 3	06 March 2018	95,056		
		Raise 41 Sub-Total	**361,672**	**14,620**	**376,293**
42	Top Access Drive	21 October 2017	78,138	548	
42	Drive 1	09 March 2018	92,101		
42	Drive 2	05 February 2019	182,389		
42	Drive 3	30 August 2018	95,989		
		Raise 42 Sub-Total	**448,617**	**548**	**449,166**
42A	Top Access Drive	04 March 2017	44,251	-	
42A	Drive 1	20 July 2017	43,840		
42A	Drive 2	11 June 2018	95,618		
42A	Drive 3	25 January 2019	62,698		
		Raise 42A Sub-Total	**246,407**	**-**	**246,407**
43	Top Access Drive	02 May 2018	42,147	12,583	
43	Drive 1	27 June 2018	41,474		
43	Drive 2	23 October 2019	103,489		
43	Drive 3	11 February 2020	86,007		
43	Drive 3 Lower	11 February 2020	4,602		
		Raise 43 Sub-Total	**277,720**	**12,583**	**290,303**
44	Top Access Drive	26 January 2019	45,326	3,447	
44	Drive 1	19 July 2019	69,557		
44	Drive 2	08 March 2021	139,282		
44	Drive 3	10 May 2022	234,940		
		Raise 44 Sub-Total	**489,105**	**3,447**	**492,552**
45	Top Access Drive	05 March 2019	17,988	-	
45	Drive 1	13 May 2019	55,440		
45	Drive 2	22 October 2021	78,291		
45	Drive 3	-			
		Raise 45 Sub-Total	**151,719**	**-**	

A-5. RESOURCE TO RESERVE CONVERSION – CONSTANTS

Field	Filter	Constant	Source
Density	All Tasks	4.17 t/m^3	Ore Reserve Statement
Stoping Width	All Tasks	1.49 m	Ore Reserve Statement
Stoping H Wall Overbreak	All Tasks	0.064 m	Ore Reserve Statement
Stoping F Wall Overbreak	All Tasks	0.066 m	Ore Reserve Statement
Scaling	All Tasks	0.057 m	Ore Reserve Statement
FOG	All Tasks	0.002 m	Ore Reserve Statement
Declines H W Overbreak	All Tasks	18.8 cm	Ore Reserve Statement
Declines F W Overbreak	All Tasks	18.8 cm	Ore Reserve Statement
Declines S Wall Overbreak	All Tasks	34 cm	Ore Reserve Statement
Unknown Geoloss	All Tasks	29.10%	Ore Reserve Statement
Panel Percentage	All Tasks	82.90%	Calculated Figure
ASG Percentage	All Tasks	3.95%	Calculated Figure
Vent Holing Percentage	All Tasks	2.19%	Calculated Figure
In-Stope Pillar Percentage	All Tasks	10.97%	Calculated Figure
Mining Extraction	All Tasks	99.95%	Geology Report
Conentrator Recovery	All Tasks	85.00%	Bench Marking
Refineries Recovery	All Tasks	95.00%	Bench Marking
Grade	All Tasks	5.305 g/t	Ore Reserve Statement
Channel Width	All Tasks	0.91 m	Ore Reserve Statement
Raise/ASG Height	Raises : Stoping Only : Spare Panels :	2.1 m 2.1 m 2.1 m	Mine Standard Mine Standard Mine Standard

A-6. FORMULAE – RESOURCE TO RESERVE CONVERSION

Field	Filter	Formula/ Constant	Comments
Raise Centares	Raises :	[Metres]*1.5	
In-Situ Centares	Stoping Only : Ledging : Raises : Diagonal : Stepovers Spare Panels :	[Area] - ([Area]*[Unknown Geo Losses]) [Area] - ([Area]*[Unknown Geo Losses]) [Raise Centares] [Metres]*1.5 [Metres]*2.4 [Area] - ([Area]*[Unknown Geo Losses])	
Mined Panels Centares	Stoping Only : Spare Panels :	[In-Situ Centares]*[Panel Percentage] [In-Situ Centares]*[Panel Percentage]	In-Situ = Centares Less Geo-Loss
Mined ASG Centares	Stoping Only : Spare Panels :	[In-Situ Centares]*[ASG Percentage] [In-Situ Centares]*[ASG Percentage]	
In Stope Pillar Centares	Stoping Only : Spare Panels :	[Instope Pillar Percentage]*[In-Situ Centares] [Instope Pillar Percentage]*[In-Situ Centares]	
Mined Vent Holing Centares	Stoping Only : Spare Panels :	[Vent Holing Percentage]*[In-Situ Centares] [Vent Holing Percentage]*[In-Situ Centares]	
Ledging Centares	Ledging :	[In-Situ Centares]	

Stoping OB Dilution Volume	Stoping Only : Spare Panels :	([Mined Panels Centares]*([Stoping H Wall Overbreak] + [Stoping F Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling]) ([Mined Panels Centares]*([Stoping H Wall Overbreak] + [Stoping F Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling])	OB = Error Over Break
Face Winch Cubbies Centares	Stoping Only : Spare Panels :	([Metres]/25)*9 ([Metres]/25)*9	25 m Spacing And Area of $9m^2$ 25 m Spacing And Area of $9m^2$
Face Winch Cubby Volume	Stoping Only : Spare Panels :	[Face Winch Cubbies Centares]*[Raise/ASG Height] [Face Winch Cubbies Centares]*[Raise/ASG Height]	
Total ASG Dilution Volume	Stoping Only : Spare Panels :	([Mined ASG Centares]*([Raise/ASG Height] - [Channel Width])) + ([Mined ASG Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling]) ([Mined ASG Centares]*([Raise/ASG Height] - [Channel Width])) + ([Mined ASG Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling])	OB = Error Over Break
ASG Ore Volume	Stoping Only : Spare Panels :	[Mined ASG Centares]*[Channel Width] [Mined ASG Centares]*[Channel Width]	
Ledging Ore Volume	Ledging :	[Ledging Centares]*[Channel Width]	
Stoping Ore Volume	Stoping Only : Spare Panels :	[Mined Panels Centares]*[Channel Width] [Mined Panels Centares]*[Channel Width]	
Raise Ore Volume	Raises :	[Raise Centares]*[Channel Width]	
Raise OB Dilution Volume	Raises :	([Raise Centares]*([Stoping H Wall Overbreak] + [Stoping F Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling])	
Total Raise Dilution Volume	Raises :	([Raise Centares]*([Raise/ASG Height] - [Channel Width])) + ([Raise Centares]*([Stoping H Wall Overbreak] + [Stoping F Wall Overbreak])) + (([Raise/ASG Height] - [Channel	OB = Error Over Break

		Width])*[Scaling])	
Vent Holing Ore	Stoping Only : Spare Panels :	[Mined Vent Holing Centares]*[Channel Width] [Mined Vent Holing Centares]*[Channel Width]	
Vent Holing OB Dilution	Stoping Only : Spare Panels :	[Mined Vent Holing Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak]) [Mined Vent Holing Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])	OB = Error Over Break
Total Ore Volume	Stoping Only : Raises : Spare Panels : Ledging :	[Stoping Ore Volume] + [Vent Holing Ore] + [ASG Ore Volume] [Raise Centares]*[Channel Width] [Stoping Ore Volume] + [Vent Holing Ore] + [ASG Ore Volume] [Ledging Centares]*[Channel Width]	
Face Winch Cubby Dilution	Stoping Only : Spare Panels :	[Face Winch Cubbies Centares]*([Raise/ASG Height] - [Channel Width]) [Face Winch Cubbies Centares]*([Raise/ASG Height] - [Channel Width])	
Stoping Waste Volume	Stoping Only : Spare Panels :	[Mined Panels Centares]*([Stoping Width] - [Channel Width]) [Mined Panels Centares]*([Stoping Width] - [Channel Width])	
Total Dilution Volume	Raises : Stoping Only : Spare Panels : Ledging :	[Raise OB Dilution Volume]*(1 + [FOG]) ([Total ASG Dilution Volume] + [Face Winch Cubby Dilution] + [Stoping Waste Volume] + [Stoping OB Dilution Volume] + [Vent Holing OB Dilution])*(1 + [FOG]) ([Total ASG Dilution Volume] + [Face Winch Cubby Dilution] + [Stoping Waste Volume] + [Stoping OB Dilution Volume] + [Vent Holing OB Dilution])*(1 + [FOG]) ([Ledging Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak]))*(1 + [FOG])	
Total Ore Tons	Raises : Stoping	[Raise Ore Volume]*[Density] ([ASG Ore Volume] + [Vent Holing Ore] + [Stoping Ore Volume])*[Density]	

	Only : Spare Panels : Ledging :	 ([ASG Ore Volume] + [Vent Holing Ore] + [Stoping Ore Volume])*[Density] [Ledging Ore Volume]*[Density]	
ASG OB Dilution Volume	Stoping Only : Spare Panels :	([Mined ASG Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling]) ([Mined ASG Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])) + (([Raise/ASG Height] - [Channel Width])*[Scaling])	
OverBreak Dilution Tons	Raises : Stoping Only : Spare Panels : Ledging :	[Raise OB Dilution Volume]*[Density] ([ASG OB Dilution Volume] + [Face Winch Cubby Dilution] + [Stoping OB Dilution Volume] + [Vent Holing OB Dilution])*[Density] ([ASG OB Dilution Volume] + [Face Winch Cubby Dilution] + [Stoping OB Dilution Volume] + [Vent Holing OB Dilution])*[Density] [Ledging Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])*[Density]	
Mined Tonnes	All Tasks	[Total Ore Tons] + [Total Waste Contamination Tons]	
Ledging Tons	Ledging :	([Ledging Waste Volume] + [Ledging Ore Volume])*[Density]	
ASG Tons	Stoping Only : Spare Panels :	([Total ASG Dilution Volume] + [ASG Ore Volume])*[Density] ([Total ASG Dilution Volume] + [ASG Ore Volume])*[Density]	
Raise Tons	Raises :	([Total Raise Dilution Volume] + [Raise Ore Volume])*[Density]	
Panel Tons	Stoping Only : Spare Panels :	([Stoping Waste Volume] + [Stoping OB Dilution Volume] + [Stoping Ore Volume])*[Density] ([Stoping Waste Volume] + [Stoping OB Dilution Volume] + [Stoping Ore Volume])*[Density]	
Vent Holing Waste	Stoping Only : Spare Panels :	[Mined Vent Holing Centares]*([Stoping Width] - [Channel Width]) [Mined Vent Holing Centares]*([Stoping Width] - [Channel Width])	
Vent Holing Tons	Stoping Only : Spare	([Vent Holing OB Dilution] + [Vent Holing Ore] + [Vent Holing Waste])*[Density] ([Vent Holing OB Dilution] + [Vent Holing Ore] + [Vent Holing Waste])*[Density]	

	Panels :		
Face Winch Waste Tons	Stoping Only : Spare Panels :	[Face Winch Cubby Dilution]*[Density] [Face Winch Cubby Dilution]*[Density]	
Total Ore Centares	Stoping Only : Raises : Ledging : Spare Panels :	[Mined ASG Centares] + [Mined Vent Holing Centares] + [Mined Panels Centares] [Raise Centares] [Ledging Centares] [Mined ASG Centares] + [Mined Vent Holing Centares] + [Mined Panels Centares]	
Total Waste Contamination Tons	Raises : Stoping Only : Spare Panels : Ledging :	[Total Raise Dilution Volume]*[Density] ([Total ASG Dilution Volume] + [Stoping Waste Volume] + [Stoping OB Dilution Volume] + [Vent Holing OB Dilution] + [Vent Holing Waste])*[Density] ([Total ASG Dilution Volume] + [Stoping Waste Volume] + [Stoping OB Dilution Volume] + [Vent Holing OB Dilution] + [Vent Holing Waste])*[Density] (([Ledging Centares]*([Stoping Width] - [Channel Width])) + ([Ledging Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak])))*[Density]	
Ledging Waste Volume	Ledging :	[Ledging Centares]*([Stoping F Wall Overbreak] + [Stoping H Wall Overbreak]) + ([Ledging Centares]*([Stoping Width] - [Channel Width]))	
Mined Head Grade	Raises : Stoping Only : Spare Panels : Ledging :	([Raise Ore Volume]*[Density]*[Grade])/[Raise Tons] (([Stoping Ore Volume] + [ASG Ore Volume] + [Vent Holing Ore])*[Density]*[Grade])/([ASG Tons] + [Vent Holing Tons] + [Panel Tons]) (([Stoping Ore Volume] + [ASG Ore Volume] + [Vent Holing Ore])*[Density]*[Grade])/([ASG Tons] + [Vent Holing Tons] + [Panel Tons]) ([Ledging Ore Volume]*[Density]*[Grade])/[Ledging Tons]	
Recovered Grade	All Tasks	[Mined Head Grade]*[Process Recovery]	
Process Recovery	All Tasks	[Conentrator Recovery]*[Refineries Recovery]	

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