A linear programming and stochastic analysis of mining replacement rate for typical Bushveld Complex platinum reef conventional mining under variable geological losses

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Abstract

The Merensky and UG2 reefs of the Bushveld Complex in South Africa are currently the largest source of known platinum resources and reserves in the world. Conventional, hybrid and mechanised mining methods are used to extract the platinum reefs. Conventional mining is the most prevalent mining method. In conventional mining, development precedes stoping primarily to provide access to and demarcate the stopes. Mining replacement rate is the rate at which development generates new stopes to replace depleting ones thus, sustaining production. Therefore it is imperative to adopt an appropriate mining replacement rate that carefully balances development and stoping, noting that financial wisdom demands deferring development as far as possible into the future because it is a cost, while operationally, deferring development sacrifices operating flexibility. Inadequate operating flexibility often leads to production shortfalls or working in inadequately prepared stopes often compromising safety. This problem is compounded by geological losses in the form of potholes, dykes and faults whose exact locations, extent and characteristics are never known with certainty prior to mining. Existing operations use mining replacement rates based on empirical approaches. This paper presents a linear programming and stochastic analytical approach to explore mining replacement rate within the range 10% to 60% for geological losses typical of Bushveld Complex platinum reefs.

This paper reports work on a current honours research project in the School of Computational and Applied Mathematics at the University of Witwatersrand (Wits), which is being pursued as an off-shoot project from a PhD research study in the Wits School of Mining Engineering and was presented to the 2009 Mathematics in Industry Study Group (MISG) in South Africa.

Keywords*:* Bushveld Complex; Merensky and UG2 reefs; pothole; dyke; fault; conventional mining; mining replacement rate; linear programming; stochastic.

1. Introduction

This paper describes the work that has been done on a current honours research project in the School of Computational and Applied Mathematics at the University of Witwatersrand (Wits), Johannesburg, South Africa. The project is being pursued as off-shoot work from a PhD research study by Musingwini (2009) in the Wits School of Mining Engineering. The project has previously been presented to the 2009 Mathematics in Industry Study Group (MISG) in South Africa.

The Bushveld Complex, previously known as the Bushveld Igneous Complex, is currently the only known source of economically mineable platinum group metal (PGM) or platinum group element (PGE) mineral resources and reserves in South Africa. The acronyms PGM and PGE are used synonymously in this paper. The Bushveld Complex is a geological formation located in the north-eastern part of the country (Figure 1) and has an areal extent just over 65,000km^2 (Cawthorn, 1999). The Bushveld Complex comprises two main limbs, the eastern and western limbs, and a much smaller northern limb (Figure 1).

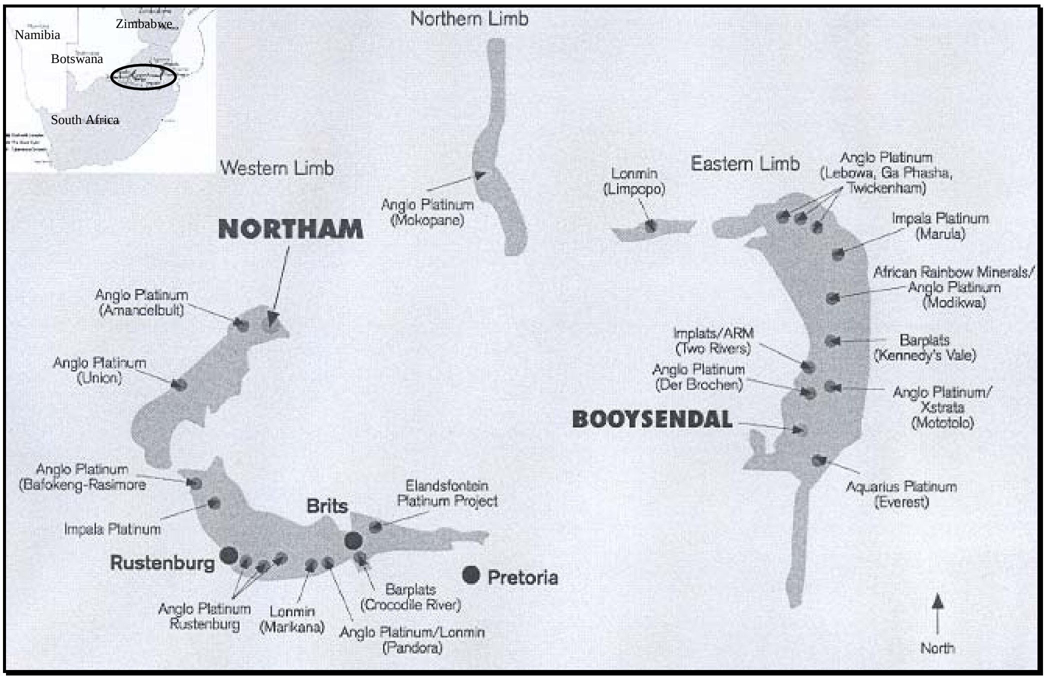

Figure 1: Platinum operations and projects on the Bushveld Complex (not to scale)
(Adapted from: Northam Platinum Limited, 2009; Implats, 2009)

The Bushveld Complex is exploited for its three distinct platinum reefs namely the Merensky, UG2 and Platreef. The Platreef occurs on the northern limb only, while the Merensky and UG2 reefs both occur on the eastern and western limbs of the Bushveld Complex but not on the northern limb (Cawthorn, 1999). The Merensky and UG2 platinum reefs are examples of shallow-dipping, inclined, narrow, tabular deposits. The reefs are shallow-dipping because they display a fairly consistent dip that ranges between 9^0 and 25^0, averaging about 10º (Watson, 2004). Shallow dips dictate that mechanical means of ore movement, such as scraper winches, must be employed in the production process unlike in mining methods on steeply-dipping reefs that rely solely on gravity to move blasted ore material from stopes faces to tramming levels.

The Bushveld Complex platinum reefs are regarded as narrow because their average thickness is typically less than 1m (Moxham, 2004). The reefs are classified as tabular because they exhibit fairly consistent planar geometry over a strike length of about 300km and over a dip distance of about 2km (Cawthorn, 1999). However, the lateral continuity is often disrupted at local mine scales by essentially four major geological structural disturbances namely potholes, iron-rich ultra-mafic replacement pegmatite intrusions (IRUPs), faults and dykes (Schoor and Vogt, 2004). Most of these geological features are not mined for geo-technical and grade control reasons, and quite often part of the mineralized reef must be left *in-situ* as bracket pillars to support these geological features. In this way potential mining areas are lost to geological losses. Geological losses typically vary between 10% and 60% for the Bushveld Complex reefs, the major losses being attributable to potholes. When geological losses exceed 60%, the reef horizon is so criss-crossed with different geological structural discontinuities that it looks like a '*dog's breakfast*' and is no longer economically mineable (Rogers, 2007). For example, geological losses experienced at some of Impala Platinum's shafts average about 40% (Table 1).

Table 1: Average geological losses for some Impala Platinum shafts
(Source: Impala Platinum Mining Projects, 2008)

Project	Geological loss
No. 4 shaft	40%
No. 7 shaft	39%
No. 10 shaft	45%
No. 12 shaft	37%
Average	**40%**

The mining methods currently used for ore extraction on the Bushveld Complex can be broadly categorised into conventional, hybrid and mechanised mining (Figure 2). Conventional mining is the most prevalent and is likely to remain as the dominant mining method in the medium to long term as indicated by Figures 3 and 4.

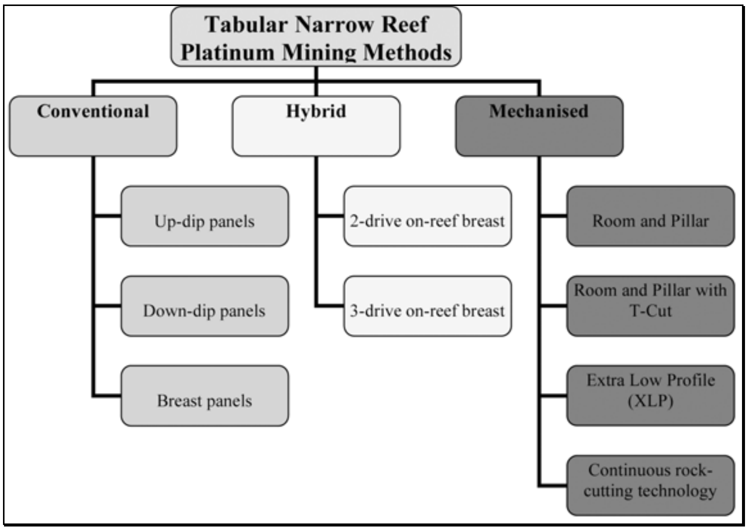

Figure 2: Classification and examples of underground platinum mining methods in South Africa
(Musingwini and Minnitt, 2008; Musingwini, 2009).

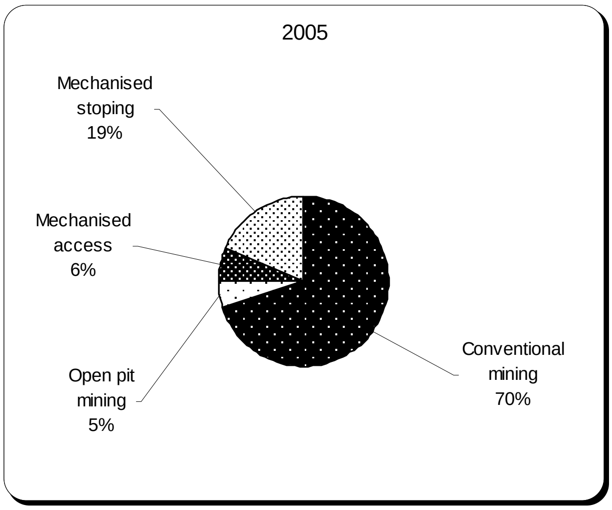

Figure 3: Distribution of PGM production output by mining method in 2005
(Pickering, 2007)

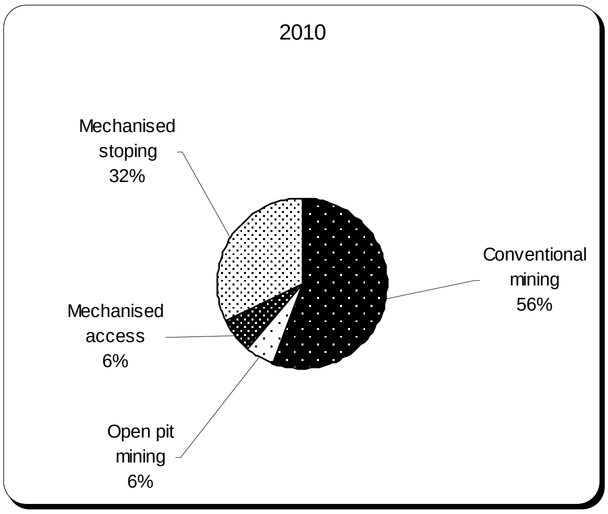

Figure 4: Forecast distribution of PGM production output by mining method in 2010
(Pickering, 2007)

For this study the laybye-access conventional breast mining method was used for analysis using guidelines obtained from Impala Platinum Mining Projects, a department within Impala Platinum. Impala Platinum, which is the second largest platinum mining company in South Africa, employs laybye-access conventional breast mining method as the principal method of ore extraction.

2. Conventional mining: operating flexibility and mining replacement rate

In a typical laybye-access conventional breast mining operation, excavation proceeds from a central shaft or decline area going down along dip and out to strike boundaries (Figure 5). Access to and preparation of stopes is done through a network of lateral and inclined tunnels called development that are excavated mostly in waste rock in the footwall of the reef horizon. Examples of development include strike drives, laybyes, travelling ways, step-overs, boxholes, raises and winzes (Figures 5 and 6). The primary functions of development in underground mines are to (Fleming, 2002):

- ❖ Provide access to the orebody or reef horizon;
- ❖ Delineate the orebody into manageable sections or stopes;
- ❖ Generate additional geological information for evaluating the orebody;
- ❖ Prepare the orebody for subsequent extraction of stopes to produce ore; and
- ❖ Provide a network of arteries for the transport of ore and waste, and movement of services into and out of the mine such as water, compressed air, ventilating air and backfill material.

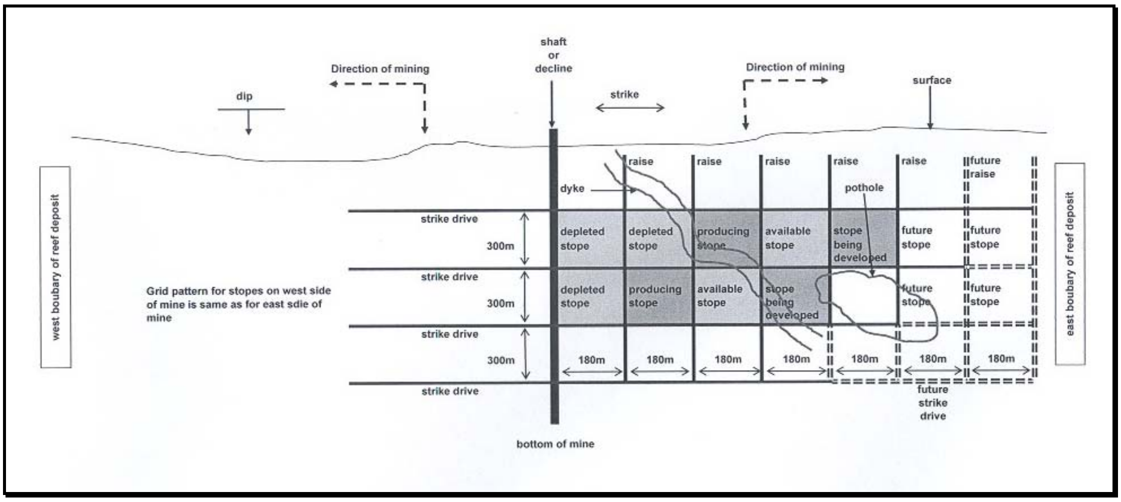

Figure 5: Schematic sequence for a conventional breast mining layout

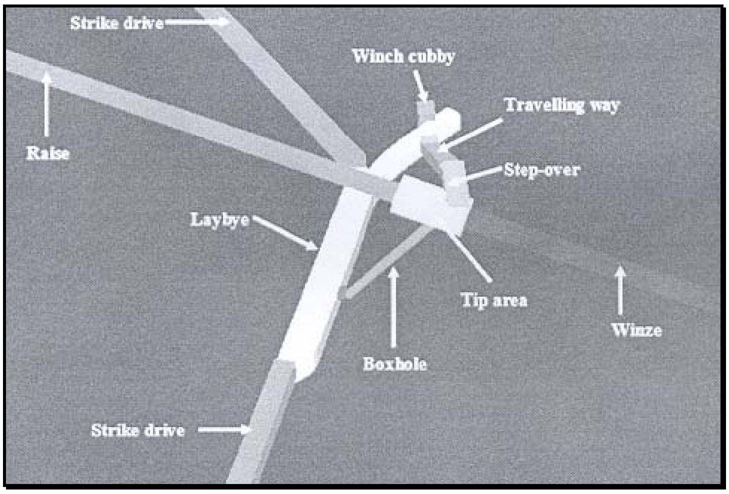

Figure 6: A 3D perspective of a laybye-access connection in conventional breast mining

The mining sequence of the development illustrated by Figure 6 starts with the strike drive, followed by the laybye. Once the laybye is completed, the winch cubby and boxhole stub then break away from the laybye. Mining of the travelling way then follows after completion of the winch cubby and muck from the traveling way is scraped using a scraper winch installed in the winch cubby. The step-over is then mined once the travelling way is completed. From the step-over, the tip area is then mined along the reef horizon. When both tip area and boxhole stub have been completed, the boxhole is then raise-bored from the stub to hole into the tip area.

Once the boxhole has holed, there is a delay of about 3 months to install two scraper winches to service the future raise and winze, install a grizzley over the tip area and equip the travelling way with a mono-winch for material transportation into and out of the stope area. The main development crew then moves from the tip area to continue mining the strike drive to get to the next raise position. Two secondary development crews then take over and start mining the raise and winze. Once the raise and winze have holed and there is through ventilation, ledging is then done to prepare the raiseline for stoping. Once ledging is complete, Advanced Strike Gulleys (ASGs) are then mined at about 20º-25º above strike. Panels, each about 30m long then follow behind the ASGs in order to use the ASGs as free breaking face. For this study it was prudent to combine ledging with stoping since ledging is similar to stoping, although ledging is mined at a slower rate than stoping.

The standard Impala Platinum laybye-access conventional breast mining layout grid is 180m raiseline spacing and 300m backlengths. Backlength refers to the dip distance along a raiseline following the reef horizon measured between two consecutive levels. After stopes have been demarcated by development they are further sub-divided into production panels. The average panel face is about 30m long. A maximum of 5 panels can be mined in any stope at any time due to safety and logistical constraints. Panel advance rates vary from about 8m/month to 17m/month and a typical industry average panel advance rate is about 12m/month. Each stope is therefore planned to produce about 1,800m^2 of mined reef area per month. The number of production panels required to meet a planned production target can be determined from the set production rate. A typical shaft produces about 100,000tpm run of mine (ROM) ore which equates to about 25,000m^2/month of reef area at an ore density of about 4t/m^3, typical of UG2 reefs. This production requirement equates to about 14 stopes that must be blasted daily. In order to ensure that 14 stopes are blasted daily, additional stopes must be made available because some stopes will mine out too early because, as Figure 7 shows, part of the reef horizon area is left unmined due to dykes, potholes and regional pillars (indicated by the circled 1), while in-panel or in-stope chain pillars (represented by the circled 2) are left unmined to provide local support for panels to complement artificial face support (represented by the circled 3). The in-stope pillars are typically 6m long by 4m wide with 2m wide ventilation holings separating them.

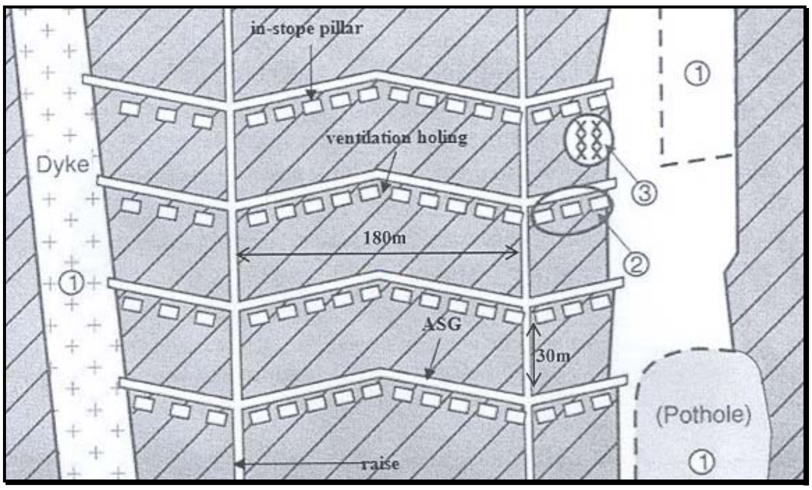

Figure 7: Typical longitudinal section of a platinum reef conventional breast mining layout
(Source: Jager and Ryder, 1999)

When there are inadequate panels or stopes for production, stoping crews sometimes end up working in inadequately prepared panels leading to minor accidents, fatalities or loss of production when shifts are lost during re-location of production crews. In response to these production uncertainties, mines usually keep development well-ahead of stoping by ensuring that additional stopes over and above those planned, 14 stopes in this case, are fully prepared and available so that stoping crews can easily be re-located to other stopes in order to make up the required number. A measure of the amount of fully developed stopes held as security against disruptions is referred to as ore availability. The number of additional stopes required as ore availability should typically equate to between 12-24 months worth of mining at current production rates (Storrar, 1977; McCarthy, 2002; Lanham, 2004; Musingwini, 2009). In this way, mines are able to subsequently create operating flexibility so that they can swiftly respond to stoping disruptions should they occur. For example, Swanepoel (2002:401) described the strategy for creating operating flexibility at Thorncliffe Chrome Mine on the Eastern Bushveld Complex as, *"this flexibility is created through development where the company currently utilizes, on cycle, 40 panels of the available 100 panels"*. However, Swanepoel (2002) does not explain how the additional 60 panels were specifically derived, but it can be assumed that the figure was based on past empirical geological and logistical experiences on the mine. Smith and Vermeulen (2006) described a similar but clearer strategy that is practised at Anglo Platinum whereby operating flexibility is created through having spare mining face that is determined by adjusting the required face length, which in their example was 200m, by the estimated global geological loss. In this strategy, Smith and Vermeulen (2006:S9.9) noted that, *"If a geological loss of 17% is considered and a simple rule of maintained spare face equivalent to the geological loss is applied a minimum of 234m of face (200 x 1.17) is required to sustain production"*. The problem with this approach is that when mining in an area where the actual geological loss is less than the global geological average, there is excess spare mining face but when the actual loss is more than the global average then there is inadequate spare face. This approach inherently incorporates operating flexibility that accounts for geological losses into the mine plan, although it can still be further adjusted depending on other factors which the mine planners may consider important. Musingwini, Minnitt and Woodhall (2007) discussed the concept of operating flexibility and derived a flexibility index, *FI*, shown by Equation 1, as a metric for measuring operating flexibility.

$$FI = \frac{Available\ Fully\ Equipped\ Stopes + Stopes\ Already\ in\ Production}{Production\ Stopes\ Required\ to\ meet\ Planned\ Production\ Rate} \qquad \textbf{Equation 1}$$

From Equation1, if $FI<1$, then for the period under consideration the operation is inadequately developed for production and has no flexibility at all since there are fewer stopes available than are required to meet the planned production rate. If $FI=1$, then the operation is temporarily inflexible because any unforeseen loss of panels causes the operation to slip back into a situation of no flexibility at all. If $FI>1$, then the operation is flexible.

Creating operating flexibility by developing well-ahead of stoping is associated with the following advantages:

- Current production can be smoothed out and future production assured because the operation is able to respond to any temporary or permanent loss of production panels;
- New geological information is generated which the mine planners can use to plan ahead and minimise production disruptions;
- Profitability of a mining operation is well-managed and enhanced over time (Woodhall, 2002; Kazakidis and Scoble, 2003); and
- An optimal path of ore extraction or an optimal production profile can be created by utilising the created flexibility (Macfarlane, 2005; Steffen and Rigby, 2005).

However, developing well-ahead of stoping has the following attendant contradictions:

- Developing well-ahead defies economic wisdom by unnecessarily tying up capital;
- It costs money to keep development ends open (support and ventilation costs) before the ends are used for production purposes; and
- There is an external factor of fluctuating mineral prices which can make initially *pay* panels to become *unpay* and vice-versa, again reducing or increasing available production panels.

These contradictions are expected because operating flexibility comes at a cost as indicated by Figure 8. An optimal mining replacement rate is therefore one which gives the best trade-off between operating flexibility and NPV. The mining replacement rate is the number of additional fully developed stopes that are created as part of ore availability in any given time period, under steady state production conditions. In order to determine ideal mining replacement rates under variable geological losses, a model was created and solved as explained in the next section.

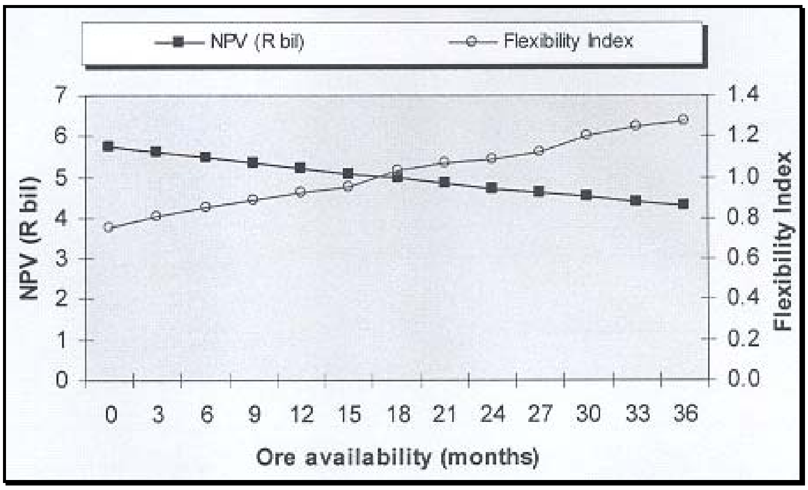

Figure 8: Relationship between operating flexibility and NPV trade-off
(Source: Musingwini, Minnitt and Woodhall, 2007)

3. Optimal mining replacement rate model

An optimal mining replacement rate is one that ensures that the operation is able to:

- Maximise operating flexibility by ensuring that *FI* is always greater than unity;
- Maximise NPV by deferring development costs resulting in creating as few spare stopes as possible and reducing the *FI*.

In order to establish an ideal mining replacement rate, consider a stope bounded by two consecutive raiselines each of length *B* (=300m made up as 100m winze portion and 200m raise portion) and at a raiseline spacing of *R* (=180m) as shown by the longitudinal section in Figure 9.

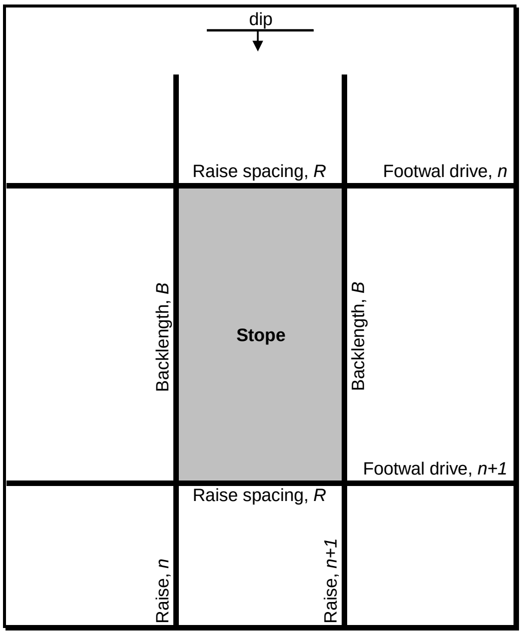

Figure 9: Schematic illustration of a typical stope without geological features

In order to establish a stope, the necessary development that must be mined and the associated mining rates, duration and costs are shown in Table 2. The corresponding mining rates, duration and costs for stopes are also shown in Table 2. The main assumptions made for the data in Table 2 are:

- A total of 14 stopes are mined per unit time every year to meet planned production of 100,000tpm which is equivalent to 25,200m^2 per month;
- Stope preparation schedule and costs are for establishing 1 stope;
- Length of strike drive to be mined per stope is equal to 134m which equal to raiseline spacing of 180m less 46m for the length of a laybye;
- Winch cubby and boxhole stub are mined concurrently;
- Boxhole is raise-bored by an EPCM contractor at EPCM costs and rates;
- Raise and winze are mined concurrently;
- Ideal conditions of 0% geological loss and 91.5% area extraction after accounting for in-stope pillars are expected to exist;
- Each stope has an area of 54,000m^2 which is reduced to 49,410m^2 to account for in-stope pillars;
- Delay to equip tip area, boxhole and travelling way is about 3 months; and
- About 1,800m^2 per stope per month or raiseline per month to give 25,200m^2 per year stoped, assuming panel advance of 12m/month and maximum 5 panels per raiseline.

Table 2: Typical industry mining rates and unit cost (in 2009 monetary terms)

Development or stoping end	Length or Area	Mining rate	Unit cost (2009 terms)	Time to mine (months)	Total elapsed time (months)
Strike drive	134m	26m/month	R3,267/m	5.15	5.15
Laybye	290m^2	145m^2/month	R204/m^2	2.00	7.15
Winch cubby	4m	22m/month	R2,883/m	0.18	7.33
Boxhole stub	2m	18m/month	R3,267/m	0.11	7.33
Travelling way	30m	18m/month	R2,339/m	1.67	9.00
Step-over	8m	22m/month	R2,883/m	0.36	9.36
Tip area	10m	18m/month	R3,424/m	0.56	9.92
Boxhole	25m	40m/month	R18,000/m	0.63	10.55
Equipping delays	---	---	---	3.00	13.55
Raise	200m	18m/month	R1,495/m	11.11	24.66
Winze	100m	18m/month	R1,554/m	5.56	24.66
14 Stopes	Area is variable	25,200m^2 per year stoped.	R204/m^2	27.45	

Table 2 shows that it takes about 2 years to prepare an additional stope for production. The stope preparation costs per stope are about R1.19mil in year 1 and R0.54mil in year 2. These costs (for each year) will have to be multiplied by the number of additional stopes required at any time to get the total stope preparation costs. The 14 production stopes being mined concurrently will each take about 2 years to mine. The cost of mining each stope in year 3 is R6.6mil and in year 4 is R7.2mil. The revenue from production when all 14 stopes are mined per unit time, in year 3 is R293.66mil and in year 4 is R76.85mil. The revenue is declining significantly because the price forecast that was obtained for the study indicated a fall in PGE mineral prices. The typical industry discount rate for PGE projects in South Africa (SA) 12% real terms and this equates to a nominal discount rate of 20% assuming long-term (LT) SA inflation of about 8.85%, a forecast that was obtained for this study.

By taking into consideration the parameters noted above, the following Linear Programming (LP) model was formulated. The LP model assumes that the mine has already passed the ramp-up period and is already in steady-state production with 14 stopes already available to meet the planned production of 100,00tpm. It is also further assumed that the mine has no space constraint and can create as many additional stopes as required.

x: available stopes and y: additional stopes in excess of 14 required for a given geological loss

An opportunity loss is made if more stopes are available than required because money is spent on stopes that are subsequently not utilized within the time period they are expected to be utilized. The opportunity loss is captured by the term (x-y) in the LP model below. The discount rate applied is 20%. The objective function is to maximise NPV.

maximise $NPV = -1.19x/1.2^1 - 1.19(x-y)/1.2^1 - 0.54x/1.2^2 - 0.54(x-y)/1.2^2 + 293.66/1.2^3 + 76.85/1.2^4$

The NPV function simplifies to: maximise $NPV = 207 - 2.734x + 1.367y$

subject to:

$(x + 14) / (14+y) \geq 1$ (flexibility index greater than unity)

$y \leq x$ (y is a sub-set of x)

$y \geq 14*0.1$ (10% geological loss lower limit)

$y \leq 14*0.6$ (60% geological loss upper limit)

$x,y \geq 0$ (non-negativity constraints)

4. Model results and interpretation

The LP model was solved using the Microsoft Excel® Solver Add-In and the result obtained as shown in the Appendix was $x=y=1.4$. This result can be expected because the NPV is only maximised when the available stopes are kept to a minimum. However, the model does not capture the situation when the stopes required for a given geological loss exceeds the available stopes, thus representing a production or revenue loss because the flexibility index will be less than unity. In order to understand how the *FI* and NPV behave under variable geological conditions ranging between 10% and 60%, the LP model was subsequently analysed stochastically using simple Monte Carlo simulation in Microsoft Excel® by taking the number of available stopes in increments of 1 between 0 and 9 based on the limits of 1.4 and 8.4 for available stopes. The results based on 10,000 simulation runs show that the NPV starts getting almost stable when available stopes are between 3 and 9 but the flexibility index is less than unity when available stopes are less than 3 as shown in the Appendix. It would appear therefore that an ideal mining replacement rate should be about 3 available stopes per unit time.

5. Conclusion

This paper has demonstrated that the problem of mining replacement rate can be solved as an LP model to get the upper and lower bounds for available stopes that can be kept as security against production disruptions. However, a subsequent stochastic analysis using a simple Monte Carlo simulation in Excel indicated that a narrower range can be defined. In the final analysis it may also be prudent to consider the excavation stand-up times in the final analysis as highlighted in the paper by Musingwini, *et al* (2003).

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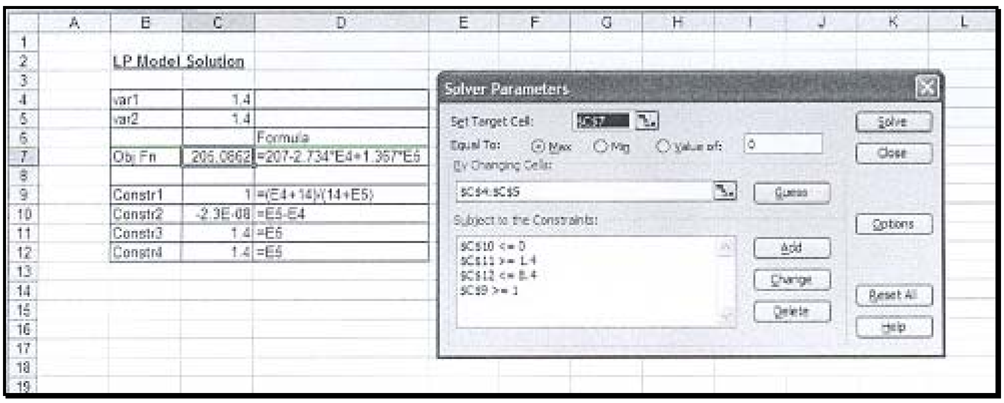
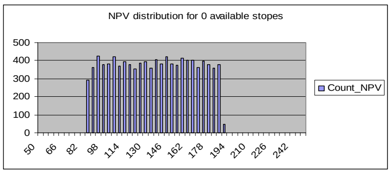
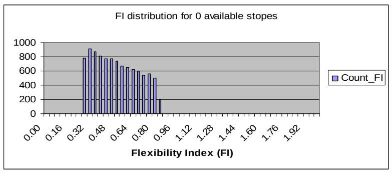
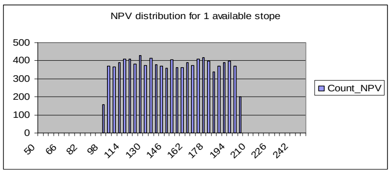
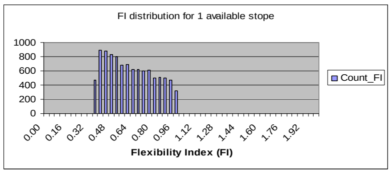
Appendix

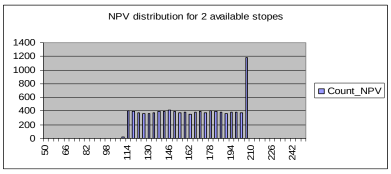


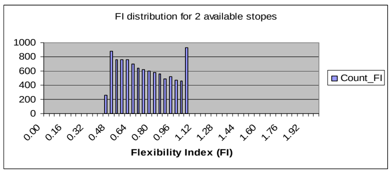

NPV distribution for 0 available stopes
Count_NPV

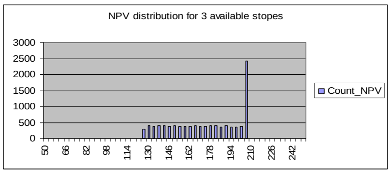

FI distribution for 0 available stopes
Count_FI
Flexibility Index (FI)

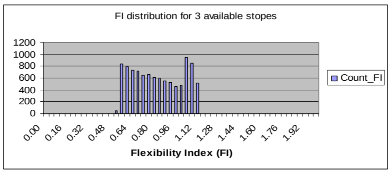

NPV distribution for 1 available stope
Count_NPV

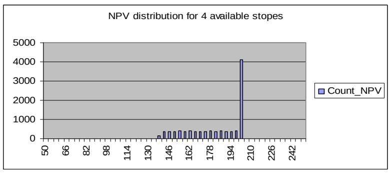

FI distribution for 1 available stope
Count_FI
Flexibility Index (FI)

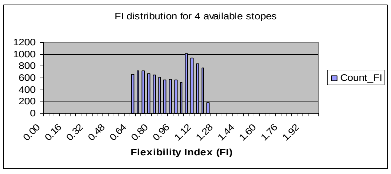

NPV distribution for 2 available stopes
Count_NPV

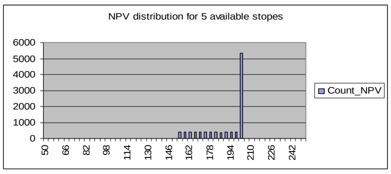

FI distribution for 2 available stopes
Count_FI
Flexibility Index (FI)

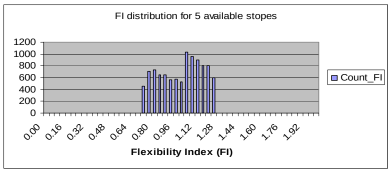

NPV distribution for 3 available stopes
Count_NPV

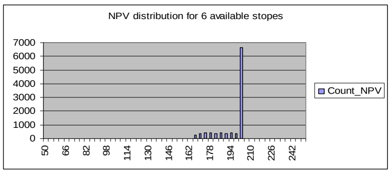

FI distribution for 3 available stopes
Count_FI
Flexibility Index (FI)

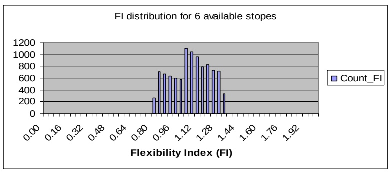

NPV distribution for 4 available stopes
Count_NPV

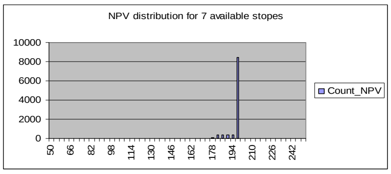

FI distribution for 4 available stopes
Count_FI
Flexibility Index (FI)

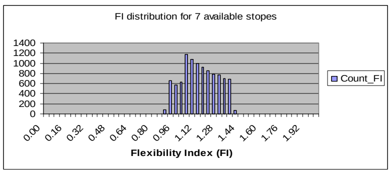

NPV distribution for 5 available stopes
Count_NPV

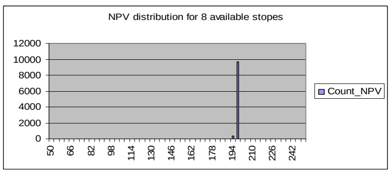

FI distribution for 5 available stopes
Count_FI
Flexibility Index (FI)

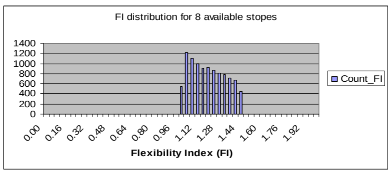

NPV distribution for 6 available stopes
Count_NPV

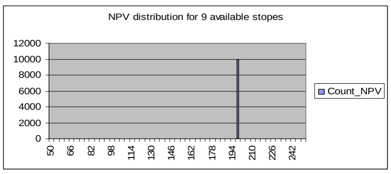

FI distribution for 6 available stopes
Count_FI
Flexibility Index (FI)

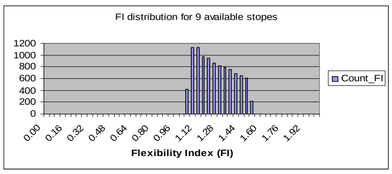

NPV distribution for 7 available stopes
Count_NPV


FI distribution for 7 available stopes
Count_FI
Flexibility Index (FI)


NPV distribution for 8 available stopes
Count_NPV


FI distribution for 8 available stopes
Count_FI
Flexibility Index (FI)


NPV distribution for 9 available stopes
Count_NPV


FI distribution for 9 available stopes
Count_FI
Flexibility Index (FI)

Biographies

Cuthbert Musingwini currently lectures at the University of Witwatersrand, Johannesburg, South Africa. He is also an associate consultant responsible for mineral asset valuation at Venmyn, a South African advisory consulting company specialising in mineral project valuation and statutory compliance. Cuthbert is a registered professional mining engineer with the Engineering Council of South Africa (ECSA), the Zimbabwe Institute of Engineers (ZwIE), is a member of the Southern African Institute of Mining and Metallurgy (SAIMM) and Mining Technical Programmes Committee (TPC), a sub-committee of the SAIMM. He has just submitted a PhD thesis for examination in multi-criteria, techno-economic optimisation in planning Bushveld Complex conventional breast mining platinum mines. Cuthbert has presented papers at international conferences and authored/co-authored several peer reviewed journal papers. His research interests lie in mine financial valuation, operations research and optimisation in mine planning.

Montaz Ali is currently an associate professor in the School of Computational and Applied Mathematics at the University of Witwatersrand, Johannesburg, South Africa. He lectures on topics in statistics and optimisation. He holds a PhD in Stochastic Global Optimization from the University of Technology at Loughborough, UK, 1994. In 2001 he was rated by the South African National Research Foundation (NRF) as a C-Rated Scientist. He has to date supervised 2 PhD, 5 MSc and 50 Honours research projects. He has authored and/or co-authored over 30 papers in journals and conference proceedings. He is a peer reviewer for 17 journals in the broad areas of optimisation, operations research and applied mathematics, including the IEEE journal.

Thato Dikgale is currently an honours student in the School of Computational and Applied Mathematics at the University of Witwatersrand, Johannesburg, South Africa.