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Sustainable ore reserves generation at variable stoping rates for a UG2 conventional mining layout

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Ore reserves are the foundation of any mining project or producing mine as these are expected to be exploited over the life of mine. Ore reserves are created through exploration and development. Ore reserves generation at a rate lower than the depletion or mining rate will lead to a shortage of places to be mined, a situation sometimes referred to as 'lack of operating flexibility' that further results in attendant production shortfall problems. Therefore it is important to critically evaluate a capital project before implementation with a view to establish if there exists an optimal balance between depletion and replacement of ore reserves in that particular project. This paper analyses a project that was initially planned to deliver 100 000 tonnes per month (≈1.2 million tonnes per year). Mine design, production scheduling and optimization were carried out in Mine2-4D® and Enhanced Production Scheduler (EPS®) and the results were subsequently exported to Microsoft Excel® for further analysis. The analysis revealed three main conclusions. Firstly, the planned production level of 100 000 tonnes per month could not be achieved at best practice development rates. Secondly, production levels above 33 000 tonnes per month (≈400,000 tonnes per year) at an average 15 m/month face advance rate would take the system out of balance. Lastly, the optimal production rate that could be maintained was 33 000 tonnes per month achieved in 4 years and maintained over 9 years. This paper demonstrates that project flexibility should be assessed before any project can be approved because it could make or break a project.

Keywords: project flexibility; ore reserves generation; best practice development rates; mining replacement rate; engineering front-end loading.

Introduction

The Bushveld Complex in South Africa 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). This paper describes work done a UG2 project in the area of sustainable ore reserves generation at the project stage of mine planning.

A sustainable generation of ore reserves is fundamental to the continued survival of a mining operation; however, most companies expend more time and resources in developing a mine to steady state capacity but pay little consideration to the requirements to maintain steady state, once achieved. Part of the problem stems from the fact that typical industry average stoping rates and best practice development rates are usually assumed in the planning process, yet these may not ultimately balance out to ensure sustainable operating flexibility specific to the project. For example, in a typical conventional breast mining it is not uncommon to assume a stoping face advance rate of between 10 m/month to 15 m/month, yet this assumption ignores operational challenges that will be unique to the mine. A 1 m/month face advance rate may be sufficient for a project with 30% overall mining losses, whereas the same face advance rate may not be sufficient for overall mining losses of 50%. In most instances it is taken for granted that once steady-state production is achieved, everything will fall into place and planned production tempo would be maintained. 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 are evident of the fact that operating flexibility is usually taken for granted at the planning stage. Musingwini, Minnitt and Woodhall (2007) tabulated previous analyses that were done on mine plans and noted that some of the analyses took no account of operating flexibility, whereas those that made reference to flexibility in evaluating the mine plans did not consider it in the final analysis. As a consequence, most previous studies have underplayed the importance of flexibility in mine plans, yet flexibility is intricately linked to profitability and project value (Musingwini, Minnitt and Woodhall, 2007). Musingwini, Minnitt and Woodhall (2007) and Stange and Cooper (2008) noted that there is now an increasing awareness in the mining industry of the value of project flexibility and robustness of a mining plan. One way of ensuring operating flexibility is by having a sustainable generation of ore reserves.

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. Keeping development ahead of stoping introduces the concept of mining replacement rate. 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 tend to rely on empirical, industry-specific rules of thumb.

Smith *et al.* (2006) stated that the mineral resource is a fundamental asset of any mining company. In order to optimize economic return by maximizing 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 in itself an alternative way of stating 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 hence the need to ensure that an appropriate mining replacement rate is maintained.

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 has a direct impact on the shareholders' equity and return on investment. Secondly, capital spent on an unprofitable 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 it is necessary to incorporate mining replacement rate as part of the planning process. This paper therefore explores the planning of sustainable ore reserves generation. An appropriate mining replacement rate for a capital project should be ascertained as part of the process of ensuring adequate project flexibility.

Flexibility can make or break a mine (Mohloki, 2010). The level of flexibility required will depend on geological losses and logistical problems such as, employee absenteeism due to sickness, incomplete cleaned panels due to poorly blasted faces, falls of ground making the area unsafe, and slow development arsing out of poor advances, or loss of blasts, due to poor ground conditions. On most UG2 reefs operations on the Bushveld Complex, the prevalent major geological losses are due to geological features such as 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 geotechnical 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.

Project flexibility is created by planning mining face length (or spare panels) in excess of the face length (or producing panels) required to meet planned production. This is achieved initially by keeping development and stope preparation well ahead of stoping activities. 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 200 m, 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 234 m of face (200 × 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 more spare mining face than planned 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.

Rule of thumb on mining replacement rate

The concept of mining replacement rate was discussed in the previous section and this section discusses the associated rules of thumb that are applied to it. 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 unexpected poor ground conditions. Therefore the balance between development and stoping of two years should ideally be adjusted to suit site-specific conditions determined by the geological complexity of the orebody and operational constraints.

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 that panel advances of four to five metres per month were typical of the mid-90s era on 30 m panel facelengths. Currently some tabular narrow reef mines can achieve panel advances in excess of 15 m/month on a sustained basis in panels with up to 36 m of face length. These improvements have been necessitated by, amongst other things, the rising cost of production. The occasional achievement of these high stoping rates has resulted in many companies being highly optimistic in estimating their production rates for new projects, resulting in mining companies underperforming as compared to their business plans.

The mining replacement rule of thumb should be applied in cases where the conditions are close to or almost similar to the conditions observed while they were derived. This hypothesis was tested by conducting project flexibility analyses on a highly faulted orebody that would not be typical of relatively undisturbed orebodies already in operation on the Bushveld Complex. Production rates that

could be sustained without running out of ore reserves were determined and compared to the original production profile that was adopted without flexibility assessment.

Description of project

The project was initially planned to produce at a rate of 100 000 tonnes per month (≈1.2 million tonnes per year). The orebody for the project is highly faulted and made up of displaced mining blocks as shown in Figures 1 and 2. High resolution scheduling was done down to panel and team levels for Phase 1 of the project area in order to fully understand how project flexibility would be affected at operational levels. The project area was divided into two areas:

- Area 1 (top 3 levels), which was made up of small mining blocks to panel resolution and referred to as Phase 1 in this study
- Area 2 (bottom 3 levels), which was made up of larger mining blocks to stope resolution and referred to as Phase 2 in this study.

The orebody thus, did not fit the 'relatively undisturbed' profile of orebodies already in production on the Bushveld Complex and on which the rule of thumb on replacement rate is based. Most platinum mines use some form of a quotient called replacement ratio or replacement factor to determine the balance between stoping and development. Quotients differ from one company to another in the input to the calculation and obviously the result of such calculations.

The most common ratio is where the total project m^2 are divided by the total project metres and the resulting quotient is believed to determine the amount of m^2 that will be exposed by a linear metre of development. While this method, in the absence of any detailed scientific analysis, gives some sort of control and a feel for what sort of performance should be aimed for at planning level, it does however, give some false sense of security in that the reserves that are supposedly exposed by development are not immediately available for mining.

Another method in use is where the reserves are classified into different categories varying from 'immediately available' to 'not available for mining' in trying to get a more meaningful quotient. This method, although more meaningful than the first, still is misleading in that it does not take into account the variability of available reserves in individual raiselines. For example, a raiseline containing a significant amount of reserves which is ready for mining may give a sense that there are ample reserves for teams to mine. The reality will, however, be that only a limited number of production teams can logistically be deployed in that raiseline. Thus, this 'ready-to-mine' raiseline actually increases the replacement ratio but does not alleviate the problem of idle teams seeking places to mine.

In order to carry out a meaningful analysis and overcome the shortcomings of the rules of thumb approaches above, it was therefore appropriate to decide on the classification of reserves into the following categories:

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

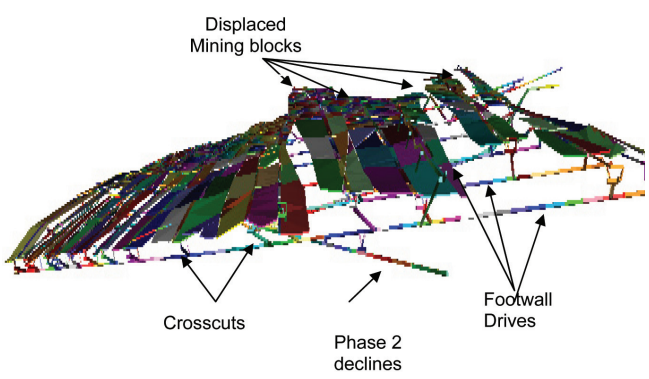

Figure 1. A 3D illustration of the project area in Mine2-4D®

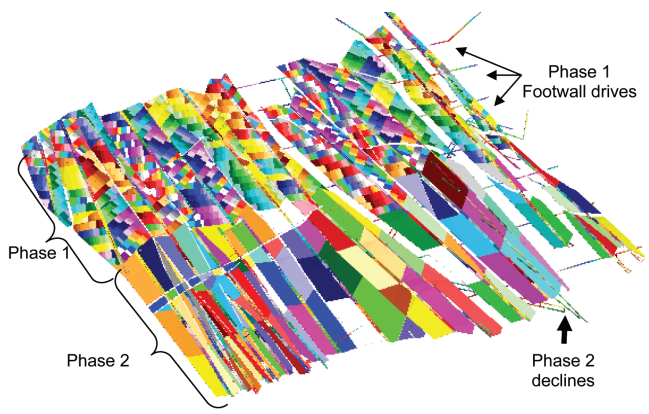

Figure 2. A 3D illustration of the two project phases in Mine2-4D®

Access strategy

The project would be accessed from a neighbouring shaft area by extending advanced strike drives (ASDs) steeply at 12° in order to quickly gain depth to the edge of the project area. The ASDs would become footwall drives that would be located off reef about 35 m below the UG2 reef horizon. The main infrastructure will not be developed on reef as in the neighbouring projects due to highly faulted and displaced ground conditions.

The distance of 35 m below the UG2 layer was calculated taking into account the following factors:

- 18 m middling between the UG2 and the UG1 layers
- The UG1 chromitite stringers and mottled anorthosite would account for an additional 5 m
- The footwall drives would be placed 12 m skin-to-skin below the UG1 chromitite stringers.

In the neighbouring projects, two different types of ASDs are developed, namely the normal ASDs and the belt ASDs. Every third ASD is a belt ASD, meaning that there are two normal ASDs being cleaned into one belt ASD. The ASDs that are extended into the new project area are the belt ASDs.

Panel layout and ore flow in stopes

Panels were designed at 38 m centres along dip direction. Provision for a maximum scraping distance of 160 m resulted in 228 m backlength which was found to be optimal. Figure 3 illustrates the planned stoping layout. 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 (ASG) is at the toe of the panel and this is the takeover point of the centre raise

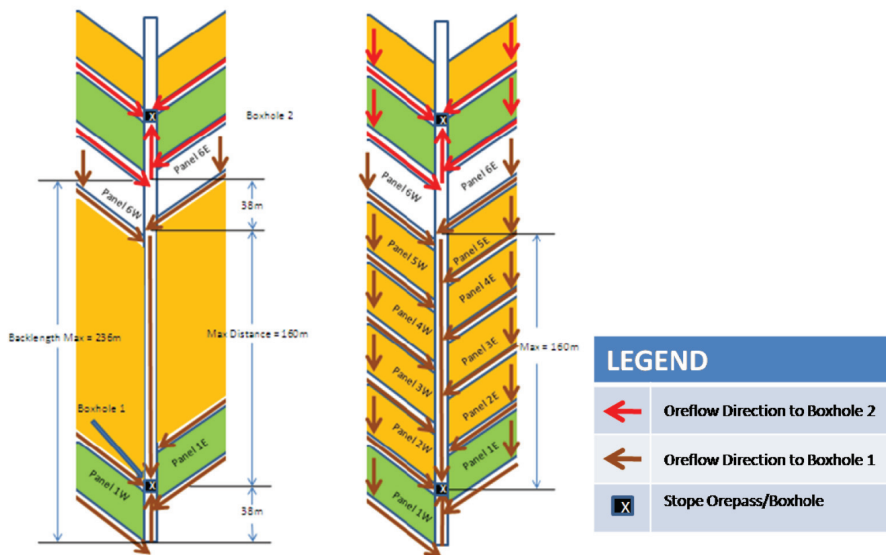

Figure 3. A schematic illustration of the panel layout

scraper winch. The channel and mining widths were 90 cm and 149 cm, respectively.

Scheduling criteria

The following principles guided the scheduling process when logistical constraints were factored in:

- There would be six development teams. Each flat end would have a dedicated team, and the other two teams would be used in the declines.
  - When a flat development crew reaches the cross-cut, they will develop 10 m beyond a cross-cut and stop. They would then complete all the flat development associated with the raiseline before continuing with the main development. The excavations to be completed before continuing with the main flat development are the cross-cut, cross-cut extension and funkhole. After the travelling way position is achieved, the main development team would go back to continue with development to the next raiseline.
- Belt extensions would be done every 150 m along strike. A delay of one month was incorporated, a delay in which the belt would be extended and an underpass created.
- The raise was scheduled to advance 10 m beyond the scraper diagonal position and then the development of the scraper diagonal would begin. The scraper diagonal development and box construction would cause a delay of one month during which time the raise development would be stopped to allow work to be carried out in the scraper diagonal.
- Ledging in a raiseline would start only after a conveyor belt had been installed in the cross-cut below, the raise has holed and permanent services have been installed in the raise.
- A total of 12 raiselines could be established over the strike length.
- On every level, there would be a total of six stoping teams and one ledging team. Stoping teams would be divided into three per raiseline. Two raiselines would be stoped while the third raiseline is being prepared for mining.
- Three levels were modelled since steady state production was planned to be achieved from these levels to form the complete Phase1 of this project.
- A travelling way would be developed for 10 m before a scraper winch could be installed at the bottom of the travelling way. A delay of five days was incorporated for the construction of a winch bed.
- After ledging, a delay of two weeks would be incurred before mining begins due to conversion from temporary to permanent services.
- A further 5-day delay is incurred in moving the winch from the foot of the travelling way into the reef horizon.

In the scheduling process also, typical South African platinum industry best practice development rates were used. These rates are shown in Table I.

Each stoping team would have a planned mining rate of 400 m^2 and a ledging team would have a planned mining rate of 380 m^2 per month with a centares-to-tonnes conversion factor of 6.21 tonnes per m^2. Therefore, all 18 stoping crews and three ledging crews are expected mine a total of 52 000 tonnes per month (≈600 000 tonnes per year), a figure that is way below the 100 000 tonnes per month (≈1.2 million tonnes per year) that were initially contemplated.

Variable scheduling rates vs. flexibility

The mine design was done in Mine2-4D® and development and production scheduling were done in Enhanced Production Scheduler (EPS®). This process was informed by inputs from geology, rock engineering, ventilation, mining

Table I
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
Cross-cut	30 m/month
Funkhole	30 m/month
Scraper diagonal	30 m/month

production, engineering, human resources, best practice, projects planning and finance departments. Best practice development rates were used for the process because these represent the maximum constraint on how much development the project could feasibly achieve, unless there was a huge departure from the current modus operandi, and such 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 area is interlinked with that of neighbouring shafts implying that such a redesign would inevitably cause interruptions in production for the integrated shaft complex with the attendant cost implications likely to outweigh its benefits.

The development rates were therefore fixed at best practice rates and only stoping rates were varied to try and establish a good balance between the best practice rates and the stoping rates. There was no scope in varying the development rates below best practice rates because doing so would constrain the stoping rates further and no benefit would be derived from such an analysis.

EPS® scheduling results and discussion

The scheduling results indicated that it took an average of 24 months to establish a raiseline for production. This figure agrees with Woodhall's (2002) findings for conventional mining in the gold mines of the Witwatersrand Basin. Table II shows the spread of total mineable tonnes and the mining duration of a raiseline in months. The reserve tonnes per raise ranged from 161 089 t to 584 733 t mainly due to geological features present in each raiseline. Mining duration ranged from 7 months to 26 months with an average of 16 months, the variation being due to variable reserve tonnes contained in a raiseline.

An analysis was carried out to determine the total amount of reserves that would be available for mining every year for the duration of the project based on the scheduling best practice rates. Figure 4 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 4 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 planned to mine all available reserves as they become available, the following can be expected:

- Temporary shutting down of mining areas due to safety or logistical constraints resulting in underutilisation of labour.
- Labour will be left idle in years where fewer places are available for mining, unless the excess labour is retrenched and rehired from time to time.

In order to minimize these impacts it is necessary to smooth out the production profile. If a production profile is smoothed at a higher mining rate, the mining duration at steady-state is shortened, whereas at lower mining rates the steady-state is extended due to the inverted bell-shape of the production profile. 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. At lower face advance rates, reserves are depleted at lower rates, thus enabling the system to create more flexibility. Stoping rates were varied from a linear advance rate of 10 m per month in increments of 1m per month to a maximum of 20 m per month. However, production smoothing results for only 10 m, 15 m, and 20 m face advance per month are illustrated Figures 5, 6 and 7 respectively, to show the trend that was identified.

The 'accessible reserves' shown in Figures 5, 6 and 7 represent tonnes available and ready to be mined; the 'added reserves' represent the apparent ore reserves created. These tonnes are the difference between unmined tonnes carried over from the previous mining period, plus tonnes added in the measuring year less the tonnes that would be mined in that particular year.

In Figure 5 at 10 m/month face advance rate, the steady-state peak production rate is 300 000 t/year (≈25 000 t/month) sustained over a period of 13 years and the apparent ore reserve ranges is between three to five years, which is higher than the rule of thumb figure of two years. From an NPV point of view this profile is undesirable but from a flexibility point of view it is very desirable. An advance rate of 10 m 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.

In Figure 6 at 15 m/month face advance rate, the steady-state peak production rate is 400 000 t/year (≈33 300 t/month) sustained over eight years. There is a substantial reduction in the apparent ore reserve to a range of one to two years, which is well within the rule of thumb figure of two years. The profile in Figure 6 shows a balanced production profile with a stable production tempo and flexibility at acceptable levels. Flexibility towards the end

Table II
Reserves statement summary

Raise no.	Reserves for mining (tonnes)	Average life (months)
36	161 089	7
37	376 951	17
38	584 733	26
38A	331 874	15
39	548 096	25
40	359 169	16
41	376 293	17
42	449 166	20
42A	246 407	11
43	290 303	13
44	492 552	22
45	151 719	7
Total or average	**4 368 352**	**16**

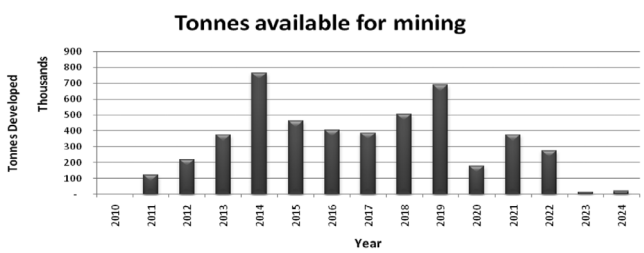

Figure 4. Reserves available for mining

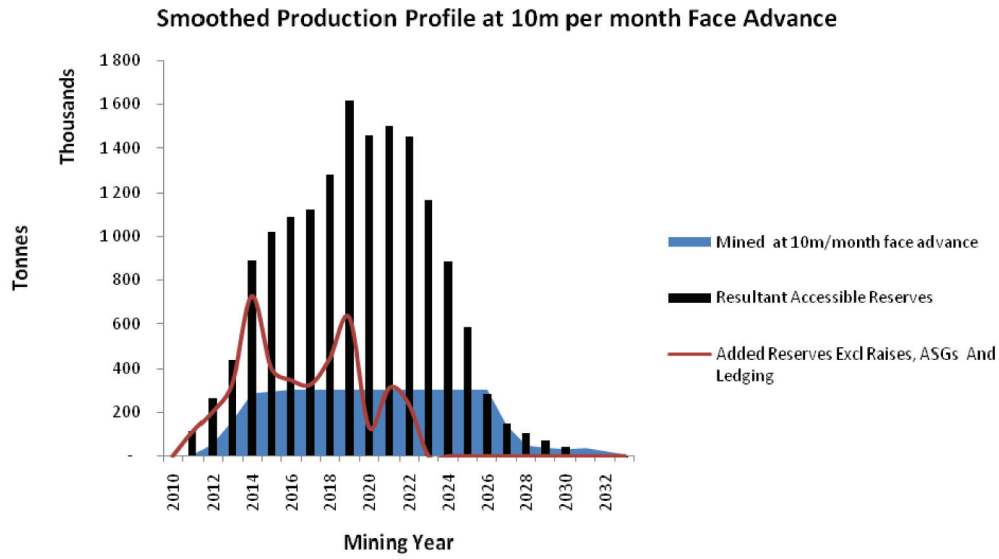

Figure 5. Smoothed profile 10 m per month face advance

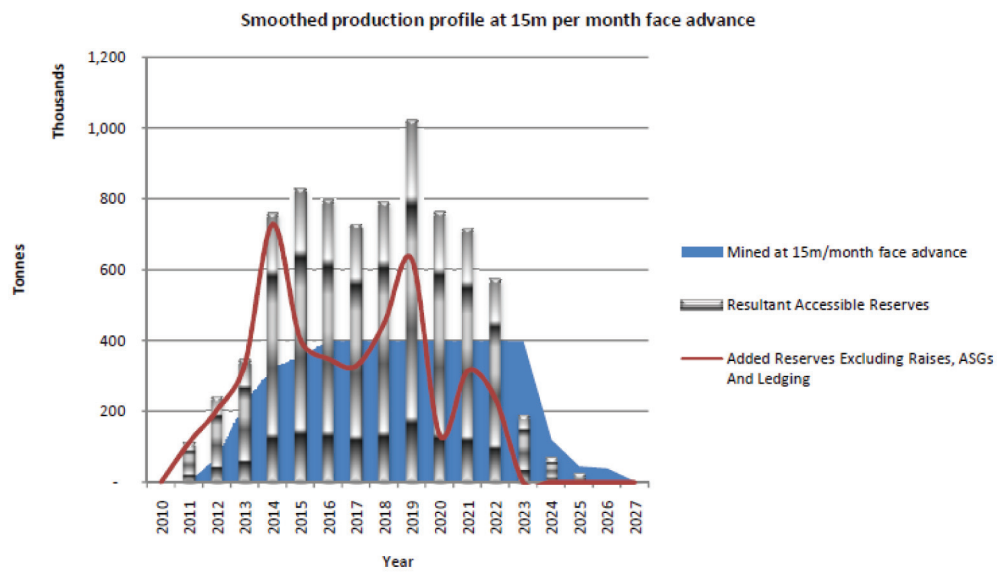

Figure 6. Smoothed production profile at 15 m per month face advance

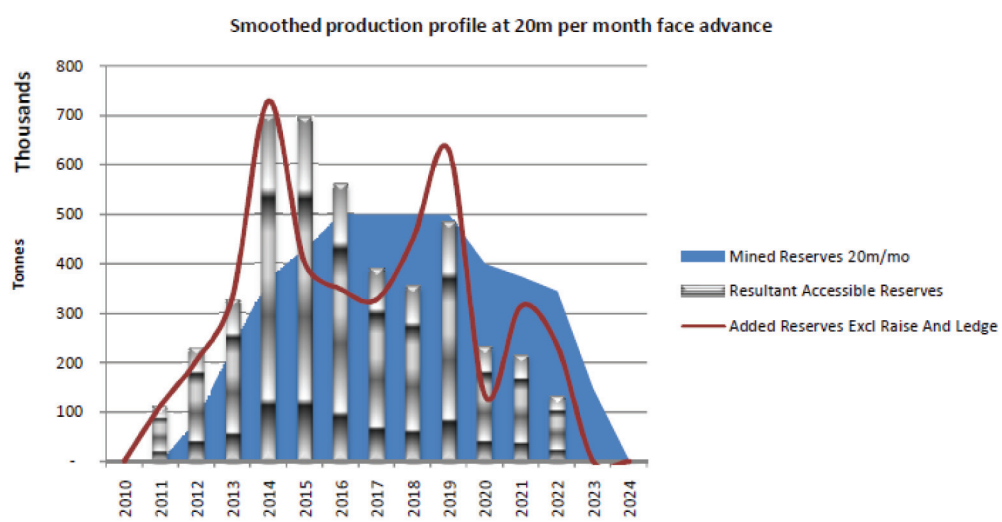

Figure 7. Smoothed production profile at 20 m per month face advance

of the project decreases but this is of no concern since the second phase of the project will kick in and new flexibility will be provided in Phase 2. Phase 2 will be executed as a replacement project for Phase 1, meaning that production teams that run out of mining reserves will be relocated to Phase 2 area.

In Figure 7 at 20 m/month face advance rate, the steady-state peak production rate is 450 000 t/year (≈37 500 t/month) sustained for a short period of only four years. There is further, a substantial reduction in the apparent ore reserve to a range of less than one year, which is unacceptably lower than the rule of thumb figure of two years. This production profile is optimal from an NPV point of view but undesirable from the flexibility point of view. It is therefore apparent that the 15 m per month advance rate and a production target of 400 000 tonnes per annum is appropriate for this project. This scheduling process that was done using the planning parameters of Table I, showed an even further drop from the reduced mining rate of 52 000 tonnes per month (≈600 000 tonnes per year) to a more realistic 33 300 tonnes per month, when NPV and flexibility are considered and detailed scheduling was done in Mine2-4D® and EPS® which took into account the different geological loss per raiseline (which gave the different tonnages in Table II) and not a global geological loss.

Conclusions

The reserves classification categories described in this paper can be useful in analysing new projects to be implemented so that the reserves availability risk can be minimized during steady-state production and therefore ensure sustainable operations. The planned production target of 100 000 tonnes per month (≈1.2 million tonnes per year) 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 and NPV desired by the mining team. 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. This paper has therefore demonstrated the need to adequately analyse project flexibility in ensuring sustainable net ore reserves generation for mining projects, as part of the engineering front-end loading for mining projects. This is critical because flexibility can break or make a mine.

References

CAWTHORN, R.G. The platinum and palladium resources of the Bushveld Complex, *South African Journal of Science*, vol. 95, November/December 1999, pp. 481–489.

LANHAM, A. Northam makes the breakthrough, *Mining Weekly*, vol. 10, no. 43, 5–11 November 2004, pp. 17–20.

MCCARTHY, P.L. Production Scheduling, Available online at: http://www.amcconsultants.com.au/Members/PDF_Papers/MP/Production_Scheduling.pdf, Posted in 2002 and accessed on 31 March 2006.

MOHLOKI, L.J. Ore reserves generation at variable development and stoping rates for a UG2 Bushveld Complex platinum reef conventional mining layout, MSc research report, University of the Witwatersrand, Johannesburg, South Africa. 2010.

MOSENTHAL, H.G. A review of stoping operations in South African gold mines, MSc dissertation, University of the Witwatersrand, Johannesburg, South Africa. 1992.

MUSINGWINI, C., ALI, M.M., and DIKGALE, T. A linear programming and stochastic analysis of mining replacement rate for typical Bushveld Complex platinum reef conventional mining under variable geological losses, *Proceedings of the Eighteenth International Symposium on Mine Planning and Equipment Selection (MPES 2009)* and the *Eleventh International Symposium on Environmental Issues and Waste Management in Energy and Mineral Production (SWEMP 2009)*, Banff, Canada, 16–19 November, 2009, pp. 551–565.

MUSINGWINI C., MINNITT R.C.A., and WOODHALL, M. Technical operating flexibility in the analysis of mine layouts and schedules, *Proceedings of the 2nd International Platinum Conference —Platinum Surges Ahead*, Sun City, South Africa, 8–12 October 2006, The Southern African Institute of Mining and Metallurgy, pp. 159–164. An electronic copy of the paper is available on http://www.platinum.org.za/Pt2006/Papers/159-164_Musingwini.pdf. 2006.

VAN SCHOOR, M. and VOGT, D. Geophysical techniques and their application to mechanized mining, *Proceedings Colloquium: Mechanized Mining and Support in the Bushveld Complex*, The South African Institute of Mining and Metallurgy, Rustenburg, 12 May 2004.

SMITH, G.L. A model for the economic evaluation of alternative stoping systems for deep level, narrow reef, gold mining operations, MSc dissertation, University of the Witwatersrand, Johannesburg, South Africa. 1992.

SMITH, G.L., PEARSON-TAYLOR, J., ANDERSEN, D.C., and MARSH, A.M. Project valuation, capital investment and strategic alignment—tools and techniques at Anglo Platinum, *Second International Platinum Conference 'Platinum Surges Ahea*d', The Southern African Institute of Mining and Metallurgy, 2006, pp. 35–42.

SMITH, G.L., and VERMEULEN, A. Strategic vs. tactical approaches to mine planning—evolution and application at Anglo Platinum, *Proceedings of the 2nd International Seminar on Strategic versus Tactical Approaches in Mining*, Perth, 8–10 March 2006, pp. S9.1–S9.11.

STORRAR, C.D. *South African Mine Valuation*, Chamber of Mines of South Africa, Johannesburg, 1977, 472 pp.

SWANEPOEL, E. Mining in the Eastern Complex, *Papers and Discussions 1997–2001*, Association of Mine Managers of South Africa and Chamber of Mines of South Africa, 2002. pp. 375–403.

TAMLYN, N. An investigation of face utilisation in the stoping operations of a deep gold mine, MSc project report, University of the Witwatersrand, Johannesburg, South Africa. 1994.

WOODHALL, M. Managing the mining life cycle: mineral resource management in deep level gold mining, MSc dissertation, University of the Witwatersrand, Johannesburg. 2002.

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