FINANCIAL EVALUATION OF THE UG2 AND MERENSKY REEF ON TWICKENHAM, NORTH EASTERN BUSHVELD COMPLEX, SOUTH AFRICA

By

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TABLE OF CONTENTS

SUMMARY

Title of Treatise:	Financial Evaluation of the UG2 and Merensky Reef on Twickenham, North Eastern Bushveld Complex, South Africa
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The Twickenham Platinum Mine (TPM) Project is located in the north eastern limb of the Bushveld Complex, north west of Steelpoort in the Limpopo Province. The property hosts platinum group metals (PGM) mineralisation in the Merensky Reef (MR) and Upper Group 2 Chromitite (UG2). The two reefs are separated by 400 m of mafic and ultramafic rocks of the Rustenburg Layered Suite.

The question that must be answered with this study relates to the economic viability of the MR compared to that of the UG2 at the TPM Project, as it stands in 2011. The assumption is that no mining has commenced on this project and that there is an equal opportunity to commence mining on one of the reefs.

The study describes the ore body characteristic for each reef, focussing on the lithologies, structure, and resources available. The discounted cash flow (DCF) method was used to determine the economic value of each reef. The net present value (NPV) and internal rate of return (IRR) were calculated and used to compare the ore bodies. The input parameters to the DCF are the main limiting factors to this method, as the results are heavily dependent on the assumptions made. The input parameters used were based on actual published values and generally accepted and motivated assumptions.

A sensitivity and risk analysis was completed to identify value ranges and potential risks to the projects. The outcome of the analysis has been compared to other projects as a benchmark to ensure the project assumptions were realistic. The world markets supply and demand for PGM is intricately related to exchange rates, metal prices, inflation, and investment risk. These have an influence on the strategic planning for a company as well as investment decisions through various project evaluation methods.

South Africa has a long history of mining and metals extraction. Extensive mining legislation has been developed to ensure the country's mineral wealth is protected and the health and safety of employees are high priority. Specific challenges related

to mining on the Eastern Limb are discussed in order to justify the high risk assigned to the project for this evaluation.

The DCF was calculated and the outcome indicated that neither the MR nor the UG2 is economically viable using these parameters in the 2011 economy.

The MR evaluation produced a negative NPV (R -1,664,541,443.47) and an IRR of 9 %, which is well below the required discount rate of 12 %. The initial project capital will be repaid after 19 years of the 33 year life of mine. The sensitivity analysis showed that by reducing the initial capital by 30 %, the project produces a positive NPV. The other factor that produced a positive NPV was by reducing the operating cost by 50 %. This project will have to be re-evaluated after all parameters have been tested and some re-engineering has been done to optimise the extraction of the MR ore body.

The UG2 evaluation produced a negative NPV (R -109,614,208.27) and an IRR of 12 %, equal to the required discount rate. The initial project capital will be repaid after 16 years of the 32 year life of mine. The sensitivity analysis showed encouraging results, as minor changes to the input parameters produced a positive NPV. The two parameters that were most significant were the recoveries and the capital requirements. By increasing the recovery percentage by 2 %, the project NPV becomes positive and a reduction of the initial capital by 10 %, also resulted in the NPV becoming positive. This indicates that with some refinement to the input parameters, the UG2 could be extracted as an economically viable project. The only concern is the sensitivity to changes in grade, which will have to be very well defined and controlled when mining commences.

The risk assessment related closely to the challenges identified for a mining operation on the Eastern Limb, with the relationship with the local community and the build-up phase of the project emerging as the highest risks. The limited infrastructure development and high levels of poverty that exists in the area has a direct influence on the support structures and services available for the build-up phase of a mine. The build-up phase requires substantial development and services that will have to be sourced at high risk and cost from substantial distances, to ensure that steady state is reached.

The socio-economic development of the local community is critical for the success of the mine. Upliftment of the local community in terms of education and training, job opportunities and health care will provide the foundation for a good relationship.

ACKNOWLEDGEMENTS

I would like to thank Anglo Platinum for allowing me to complete my studies, especially Quartus Snyman for his patience and time, and Allan Hartley for his input and interest.

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And to my husband, Duncan, thank you very much for the motivation, insight patience and sacrifices you made, getting us through this MSc.

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LIST OF ABBREVIATIONS

MR	Merensky Reef
UG2	Upper Group 2 Chromitite
Ton or t	Tonnes, one ton equals 1000 kg
4E	100% prill split of Pt, Pd, Rh, Au (also 3 PGE + Au)
NPV	Net Present Value
IRR	Internal Rate of Return
RA	Risk Assessment
TPM	Twickenham Platinum Mine
DCF	Discounted Cash Flow
ZAR	South African Rand (R)
US$	United States Dollar
LOM	Life of Mine
DMR	Department of Mineral Resources
PGM	Platinum Group Metals
BC	Bushveld Complex
oz	Troy ounces (one gram equals 31.10348 troy ounces)
pa	Per annum
g/cm^3	Grams per cubic centimetre (units for density)
PGE	Platinum Group Elements
km	Kilometre
m	Metre
cm	Centimetre
°	Degree
°C	Degrees Celsius
%	Percentage
m^2	Square metres
Mm	Million metres
g/t	Gram per ton
ktpm	Kiloton per month
R/t	South African Rand per ton

LIST OF FIGURES

LIST OF TABLES

LIST OF TABLES IN APPENDICES

1. INTRODUCTION

The Twickenham Platinum Mine (TPM) Project is situated on the North Eastern Limb of the Bushveld Complex, approximately 35 kilometres (km) north-west of Steelpoort in the Limpopo Province, South Africa (figure 1).

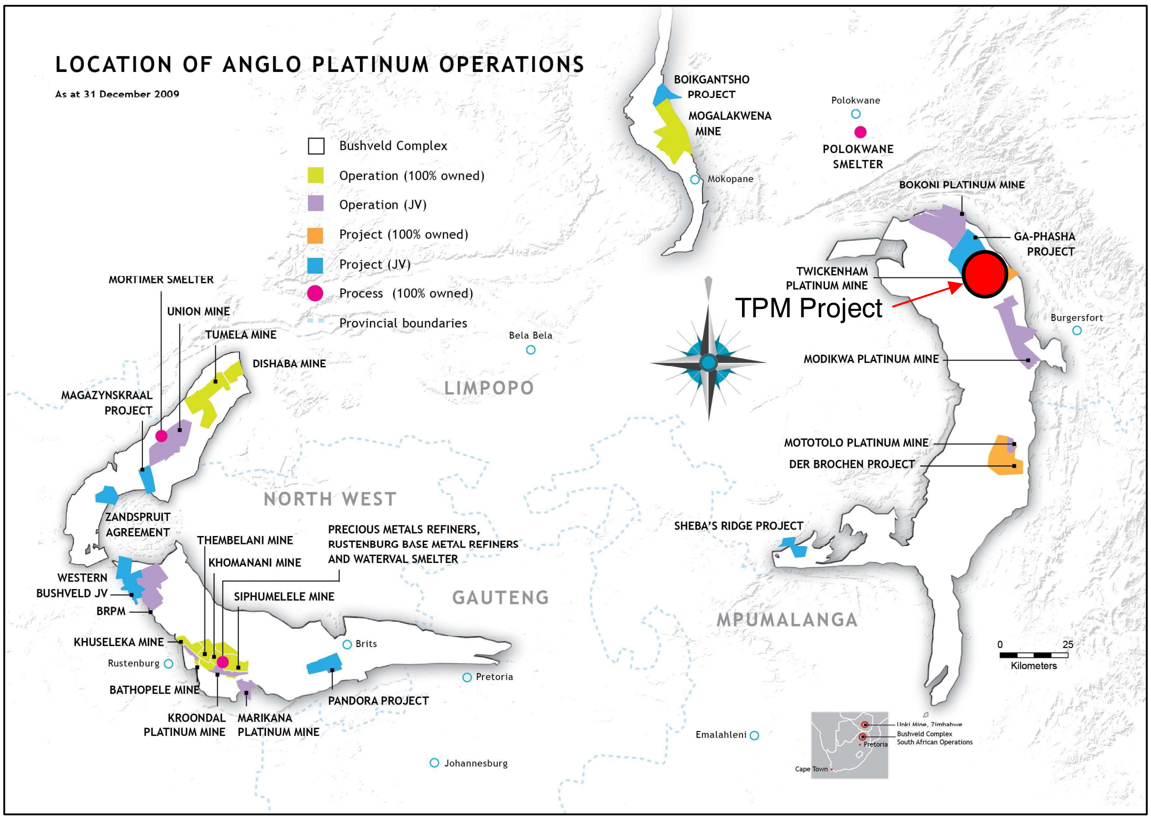

Figure 1. Locality plan showing the Bushveld Complex with the location of the Anglo Platinum Operations. The Twickenham Platinum Mine (TPM) Project location is highlighted in red (modified from Anglo Platinum Annual Report, 2009).

The platinum group metals (PGM) mineralisation at the TPM Project is hosted within the Merensky Reef (MR) and Upper Group 2 (UG2) Chromitite. These two economic horizons are separated by 400 metres (m) of mafic and ultramafic cumulate rocks of the Rustenburg Layered Suite.

The TPM Project is currently in a pre-production phase. Surface and underground infrastructure are being developed for the mining of the UG2. Stoping is planned in the build-up phase to produce a stockpile of UG2 for the commissioning of the concentrator plant in 2015, and mining is planned to reach a steady state during 2019.

The 400 m separation between the MR and UG2 means that the two reefs cannot be mined using the same underground infrastructure. Each reef will have to be accessed via separate decline/shaft systems.

Exploration for the MR and UG2 on the Eastern Limb date back to the 1960's, and was mainly conducted through trenching along the outcrop and small scale mining from adits on the hills. There were limited diamond drill holes, but as the interest in PGM mounted, more extensive diamond drilling programs were executed.

There are currently 1,382 diamond drill holes on the project area, of which 1,062 are UG2 intersections and 407 MR intersections.

The main focus for diamond drilling during 2000 – 2003 on the TPM Project was the shallow UG2 resource. The decision to focus exploration and mining efforts on the UG2, in preference to the MR, was based on the palladium price and demand at the time of the pre-feasibility study. During 2000 the palladium price showed an upward trend and peaked at around US$ 1,100 in December 2000/January 2001, where after the palladium price dropped down to just above US$ 300 at the end of 2001 (figure 2). The high palladium price is significant as the UG2 has lower platinum to palladium ratios (1:1) than the MR (2.3:1) on the project area (Viljoen and Schürmann, 1998).

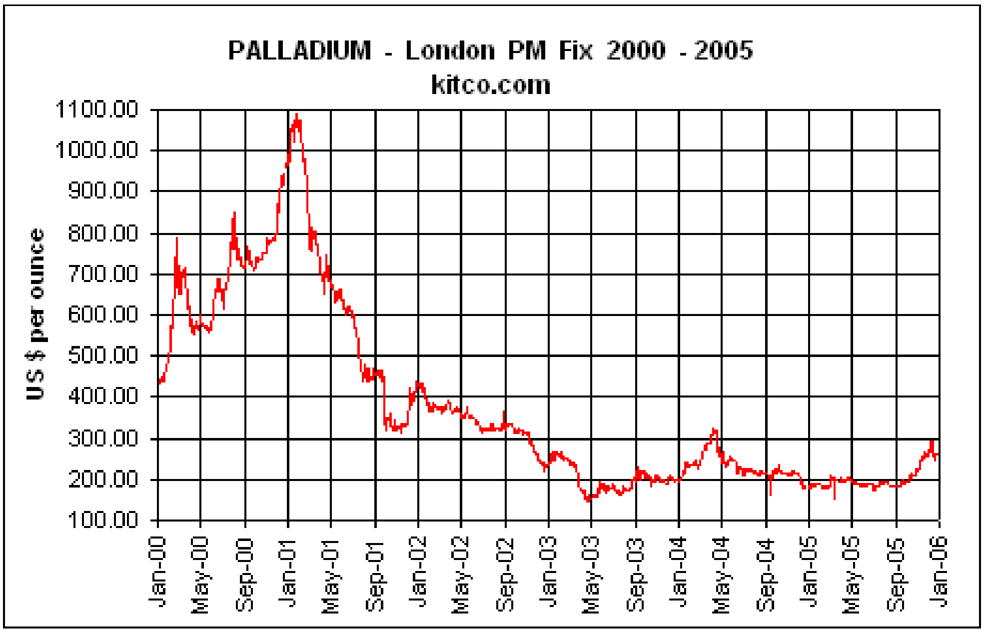

Figure 2. This graph shows the history of the palladium price (US$ per ounce) between January 2000 until January 2006. The peak in January 2001 is very prominent (Source: www.kitco.com).

The metal prices for platinum (Pt), palladium (Pd), rhodium (Rh), and gold (Au) also referred to as 4E, and the South African Rand/United States Dollar (ZAR/US$) exchange rates for January 2001 were used to calculate the approximate basket price for both reefs at the time (data from www.kitco.com and www.oanda.com). The MR 4E basket price was in the vicinity of R 6,034.37 per 4E ounce and the UG2 R 7,344.67 per 4E ounce. This would have made the UG2 the more attractive option.

The other factor that could have contributed to the selection of the UG2 rather than the MR could have been due to the difficulties encountered during mining of the MR at Atok. Atok (also known as Lebowa and now Bokoni) is situated approximately 60 km north of the TPM Project, towards Polokwane. MR mining commenced during 1968, and the main challenge has been the abundance of potholing on the reef horizon (Brown, 2003).

The MR resource was recently updated for inclusion as a replacement project for the UG2 in the long term mine extraction strategy of the TPM Project.

1.1. THE RESEARCH PROBLEM

How does the economic viability of the MR compare to the UG2 at the TPM Project as it stands in 2011?

1.1.1. The sub-problems

The following issues will be addressed in order to answer the above question:

i. What are the ore body characteristics (geological; metallurgical; structural; and dimensional) of the MR and UG2 at the TPM Project?

ii. What is the net present value (NPV) and internal rate of return (IRR) of the MR and UG2 at the TPM Project and what risk-related factors will influence the economic viability?

iii. How does the MR and UG2 at the TPM Project compare to similar projects in terms of ore body characteristics, mining costs and production?

1.1.2. Delimitations

Mining methods, mine design and scheduling related factors for the UG2 have already been selected and is in execution, thus will not be challenged during this study.

The various methods of ore processing and metal extraction are not in the scope of this study and will not be discussed.

The information for the financial evaluations will be based on published reports, general market trends and motivated assumptions.

Information for the project comparisons will be from published reports only.

1.1.3. Assumptions

For this comparison it will be assumed that neither reef (MR or UG2) has been mined yet and that there is an equal opportunity to commence one of the projects.

The commodity prices and exchange rates that will be used to calculate the basket price will be the same for both the MR and UG2.

Due to the similarities in reef thickness, minimum stoping width requirements and dip of the ore body, it will be assumed that the mining method, mine design and extraction strategy, already approved for the UG2, can be applied to the MR. The use of the same mining method for both reefs is successfully applied in the Rustenburg area.

The capital requirements will be the same for both the MR and UG2 projects, as the same basic infrastructure will be required, regardless of which reef is being extracted. Basic infrastructure will include roads, overhead power lines, offices, workshops and underground access to the ore body.

Operating cost for the build-up phase and steady state phase of mining will be the same for both the MR and UG2 projects. This can be assumed because the mining method, required infrastructure and extraction strategy, is the same for both projects.

The metal recovery data from mineralogical and metallurgical studies completed during the exploration phase of the TPM Project is accepted. These recovery percentages will be achieved during the life of mine.

1.1.4. Relevance of the study

This study will describe the MR and UG2 reefs at the TPM Project, comparing their characteristics and giving an indication of the value of each reef.

This information can be used to motivate for additional drilling and the approval for the MR pre-feasibility study.

2. PLATINUM REVIEW

2.1. PLATINUM IN THE WORLD

The mining of platinum group metals (PGM) has evolved from panning for nuggets in the alluvial deposits of South America (Columbia) and Russia (Ural Mountains), to the discovery of nickel in Canada's Sudbury area where PGM were produced as by-products in the 1800's. The most significant discovery of PGM occurred in the early 1900's when Dr. Hans Merensky discovered and subsequently delineated the Bushveld Complex in South Africa. Another significant discovery was the Norilsk nickel deposit in northern Russia during the 1930s, where PGM is produced as a by product (Cramer, 2000).

PGM is the collective term for all platinum group elements (PGE) and their accessory minerals. The main PGE are platinum (Pt), palladium (Pd), iridium (Ir), osmium (Os), rhodium (Rh) and ruthenium (Ru). The PGE vary in physical properties (table 1) but are chemically very similar. PGE are very rare in the earth's crust and usually occur in base metal sulphide minerals, for example pyrrhotite, pentlandite and chalcopyrite or in PGM. Common PGM are alloys, sulphides, arsenides, antimonides, bismuthides and tellurides (BGS, 2009; Viljoen and Schürmann, 1998).

Table 1. Table showing physical properties of the six PGE as well as Au (Source: modified from BGS, 2009).

Physical properties of PGE and Au							
	Pt	Pd	Rh	Ir	Ru	Os	Au
Atomic weight	195.08	106.42	102.91	192.22	101.07	190.23	196.97
Density (g/cm^3)	21.45	12.02	12.41	22.65	12.45	22.61	19.3
Melting point (°C)	1769	1554	1960	2443	2310	3050	1064
Electrical resistivity (micro-ohm cm at 0℃)	9.85	9.93	4.33	4.71	6.8	8.12	2.15
Hardness (Mohs)	4-4.5	4.75	5.5	6.5	6.5	7	2.5-3

Pt, Pd and Rh are the most significant commercial PGE. Their main use is for industrial applications, as they act as catalysts when alloyed with other metals (BGS, 2009).

There are many other uses for Pt and Pd these include medical applications (anti-cancer drugs, pacemakers, catheters and dental alloys); jewellery; auto catalysts in the automotive industry; industrial uses (LCD glass, hard disks in laptops, GPS devices); and chemical applications such as manufacturing of silicone and petrochemical intermediates as well as use in fuel cells (Anglo Platinum Annual Report, 2010).

Cawthorn (2010) stated that '*there are enough platinum group element deposits in the Bushveld Complex in South Africa to supply world demands for many decades..'* in his recent review of the reporting of PGE resources and reserves.

Pt production from the Bushveld Complex (BC) is currently around 5 million ounces per year and the BC is estimated to represent 75 % of the world's Pt resources (Cawthorn, 2010).

PGE resources also occur in other parts of the world such as in the Great Dyke in Zimbabwe, the Stillwater complex in Montana (USA), and as by-products in nickel-copper deposits like the Sudbury area in Canada and the Noril'sk-Talnakh district in Russia (BGS, 2009).

Pt concentrations vary significantly between ore bodies, as for example SA ores are typically 0.1 oz/ton; Stillwater 6 oz/ton; and Norilsk up to 50 oz/ton (Cramer, 2000). The Pt to Pd ratios also vary between ore bodies and deposits from the Northern Hemisphere typically have 1:3 Pt to Pd ratios, the Great Dyke and Western Bushveld have ratios of 2.5:1 while the Eastern Bushveld show 1:1 ratios. PGM associated with nickel-copper ores are usually in very low concentrations (Cawthorn et al., 2002; Cramer, 2000).

2.2. PLATINUM MARKET TRENDS

The demand for a metal is the main driving force to carry out exploration and find more ore deposits. An imbalance between supply and demand will cause the price of the metal to rise significantly, as the metal becomes ever increasingly scarce. This forces new engineering initiatives to produce a suitable alternative to the now ‘scarce resource’.

A good example of this cycle is the drive for a cleaner environment, by implementing legislation to ensure cleaner vehicle exhaust emissions. The demand by vehicle producers for auto catalysts moved the PGM industry into a mature and stable market, but as the Pt prices rose, Pd was being used as a cheaper substitute. This resulted in a big demand for Pd, and caused a serious imbalance between the supply and demand ratios for Pt and Pd. The result was a shortage in the supply of Pd and the price increased dramatically during 1999 and 2000 (Cramer, 2000 and van den Berg, 2008). Major automotive manufacturers began shifting back to Pt as the catalytic agent because of the short supply and high price of Pd (BGS, 2009). Figure 3 shows the world production of Pt and Pd and the massive drop in Pd production as a result of the high price and greater demand for Pt during 2001 - 2002 (BGS, 2009).

Figure 3. World mine production of Pt and Pd from 1997 to 2007. (Source: BGS, 2009).

The Pt supply and demand statistics for 1989 and 2009 are shown in figures 4 – 7 below (data obtained from Cramer, 2000 and Anglo Platinum Annual Report 2009). In the last 20 years the main contributing producers remained the same, with South Africa being the largest contributor to the global Pt market. North America's contribution is declining while Russia is still the second largest Pt producer. The 'other' smaller deposits increased their contribution from 2 % in 1989 to 6 % in 2009.

The global demand for Pt is mainly from the auto catalyst and jewellery sectors, while the industrial market demand has stayed fairly consistent. The investment sector has grown dramatically in the last 20 years, from 5 % in 1989 to 12 % in 2009.

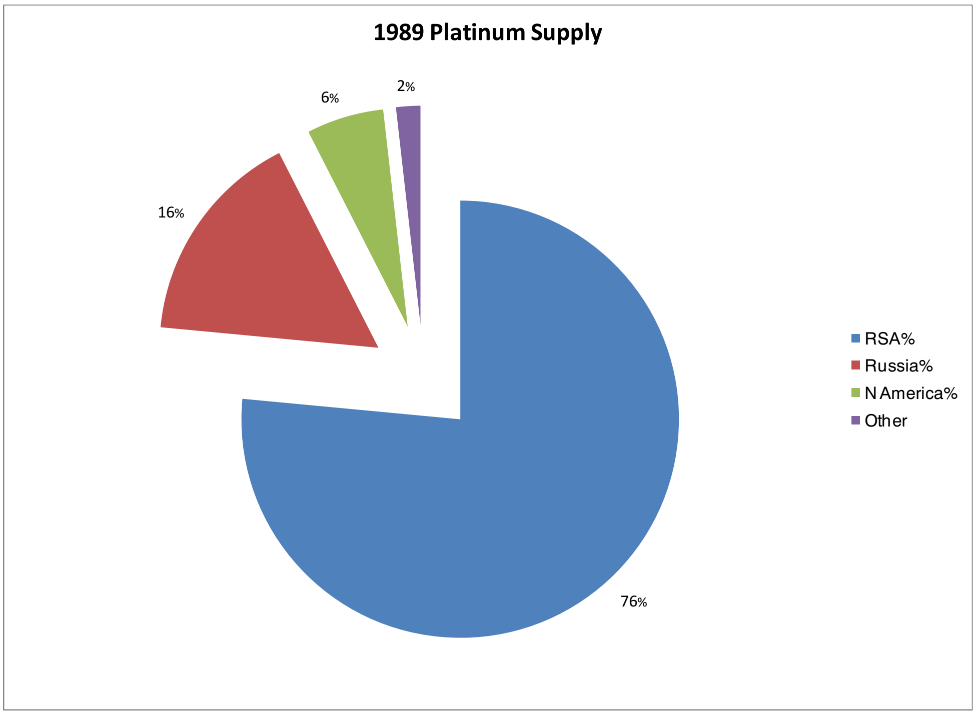

Figure 4. Pie chart showing the breakdown of global Pt supply during 1989 (data from Cramer, 2000).

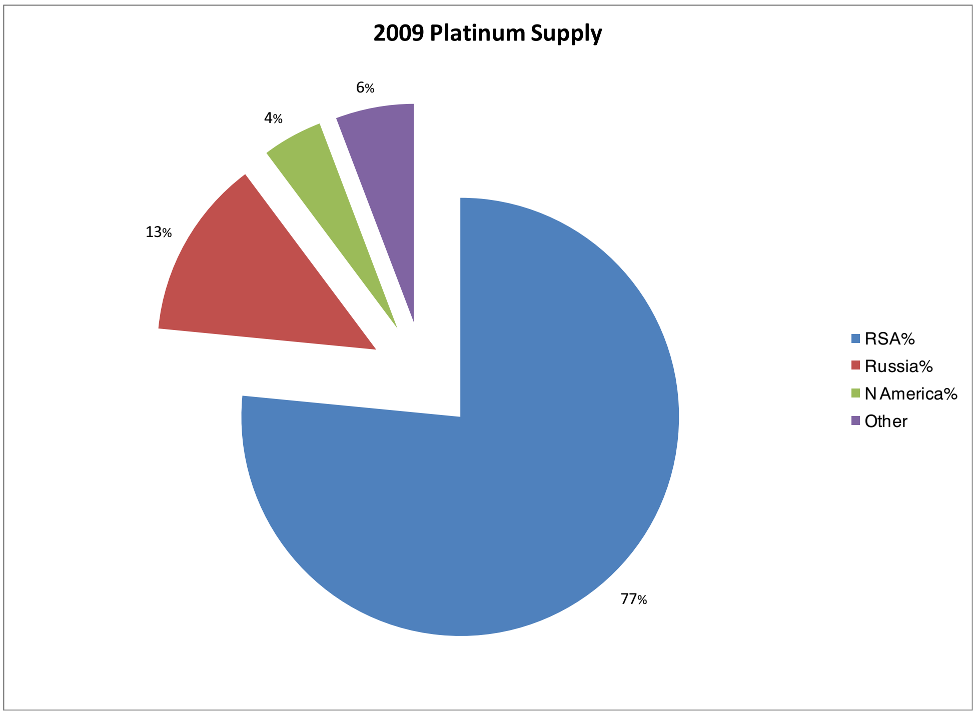

Figure 5. Pie chart showing the breakdown of global Pt supply during 2009 (data from Anglo Platinum Annual Report 2009).

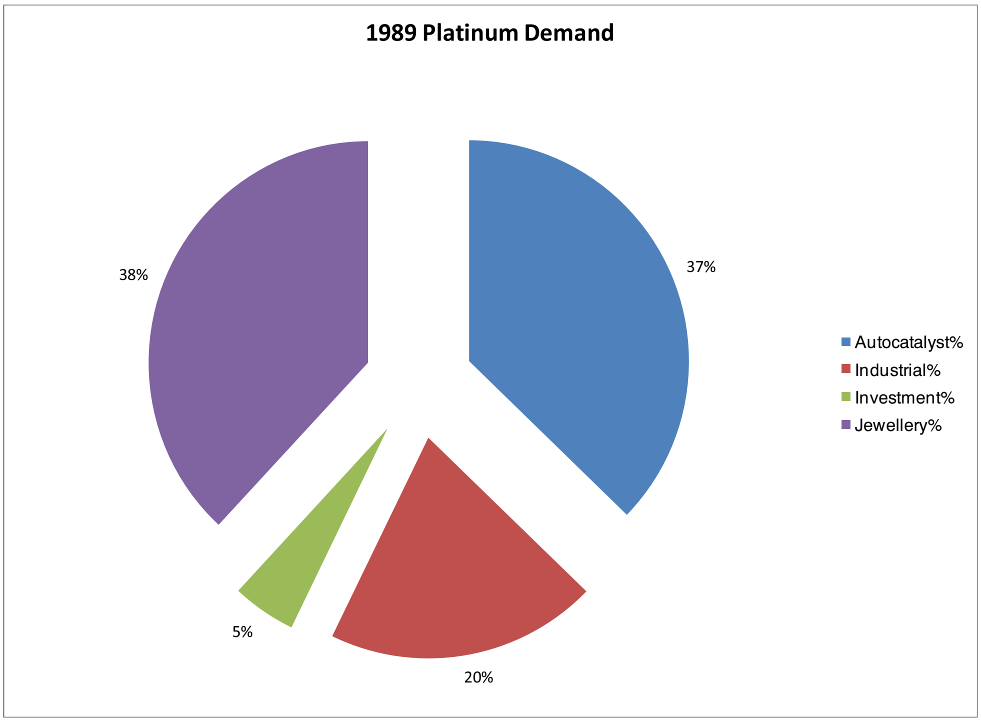

Figure 6. Pie chart showing the breakdown of global Pt demand during 1989 (data from Cramer, 2000).

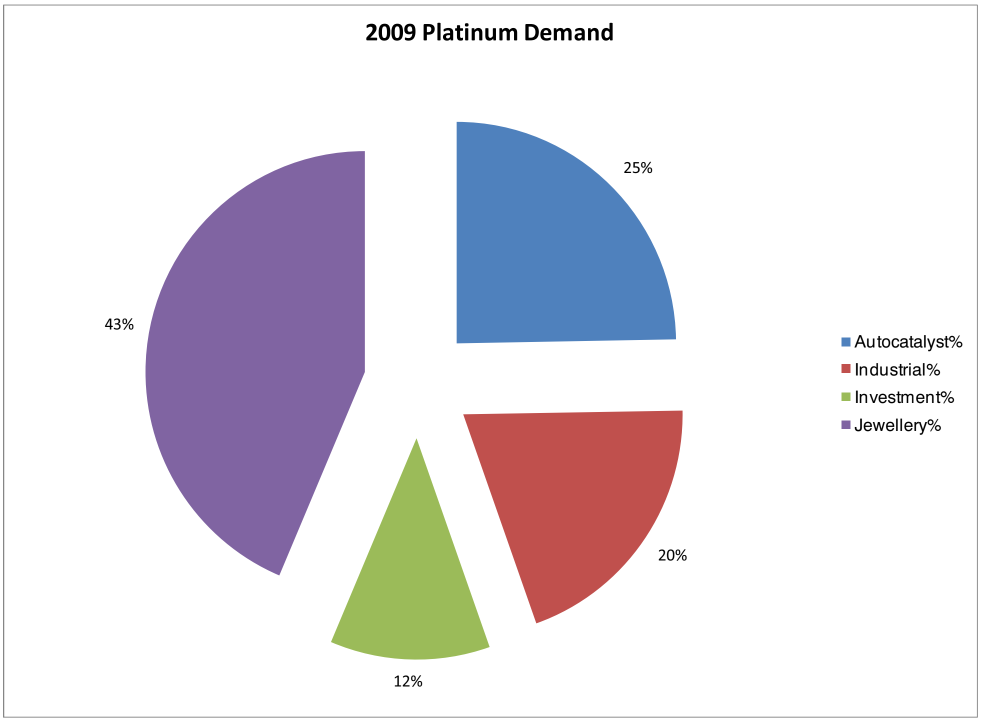

Figure 7. Pie chart showing the breakdown of global Pt demand during 2009 (data from Anglo Platinum Annual Report 2009).

The global supply and demand trends were plotted in figure 8 to show the comparison. In the 1970's and 1980's supply and demand were well balanced, but in the ten years between 1989 and 1999 the demand for Pt rose much faster than the supply. The next ten years show that the Pt supply was increased, but the demand has stabilized. It would seem that an oversupply of Pt can occur if the demand does not improve.

The next/future Pt demand is expected to come from the development of the polymer electrolyte membrane (PEM) fuel cell (Cramer, 2000). Motor manufacturers are developing new commercial fuel cell (electrical) cars, in the continuous drive for a cleaner environment. In light of more stringent legislation on carbon emissions worldwide, the fuel cell is a clean and efficient producer of energy (Swan et al., 1994). According to the 2010 Anglo Platinum Annual Report, the demand for Pt has nearly doubled from this sector during 2009 - 2010.

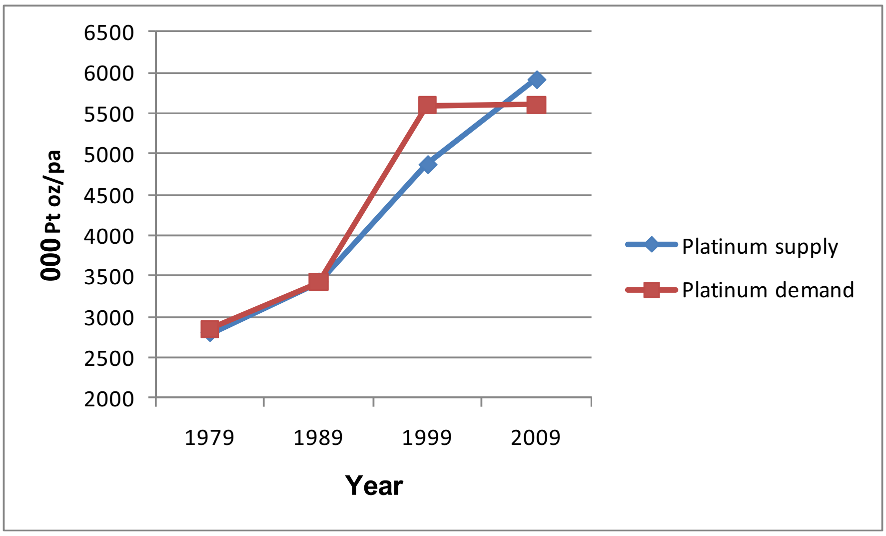

Figure 8. Graph showing the comparison of global Pt supply and demand between 1979 and 2009 (data from Cramer, 2000 and Anglo Platinum Annual Report 2009).

Global supply and demand trends are significant as it relates closely and influences other economic trends for example, metal or commodity prices and the currency exchange rates.

The following figures (9 - 12) show the historical trends for the Pt, Pd, Rh and Au commodity prices. What are significant are the fluctuations evident in these graphs. The two recent economic downturns are also clearly shown in the commodity prices, the 2000 - 2001 economy slow-downs, and the recent 2007 - 2008 global economic crisis.

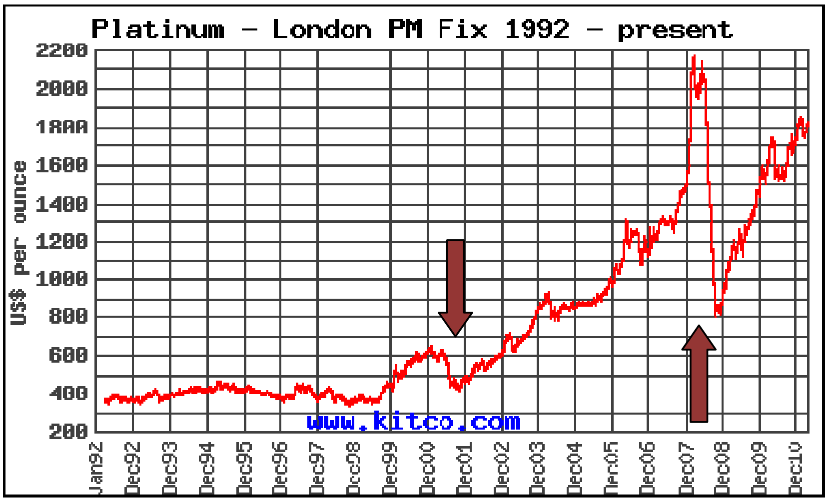

Figure 9. Graph showing the Pt price in US$ per ounce from 1992 until December 2010 (Source: www.kitco.com).

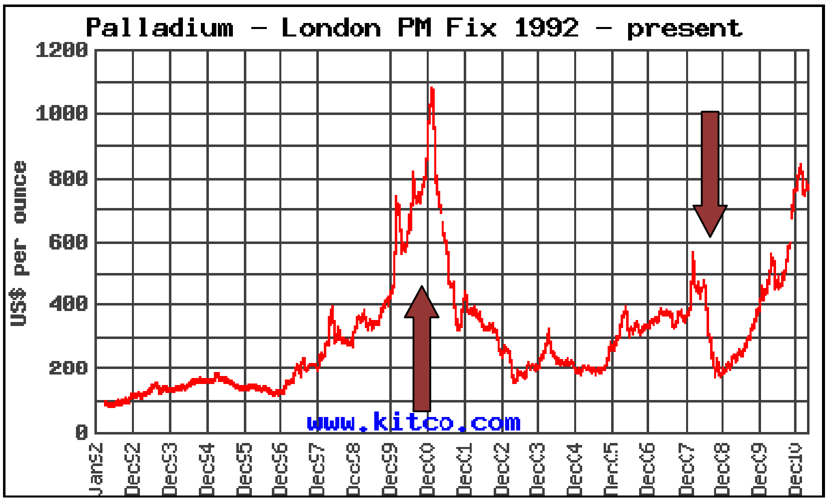

Figure 10. Graph showing the Pd price in US$ per ounce from 1992 until December 2010 (Source: www.kitco.com).

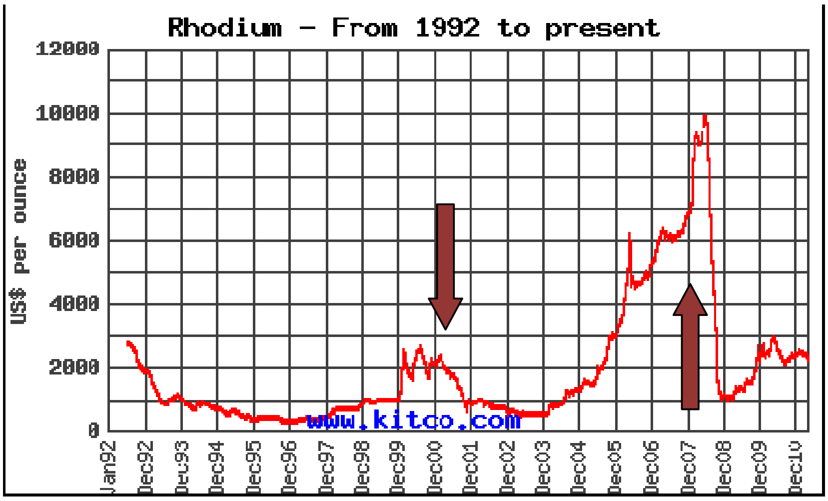

Figure 11. Graph showing the Rh price in US$ per ounce from 1992 until December 2010 (Source: www.kitco.com).

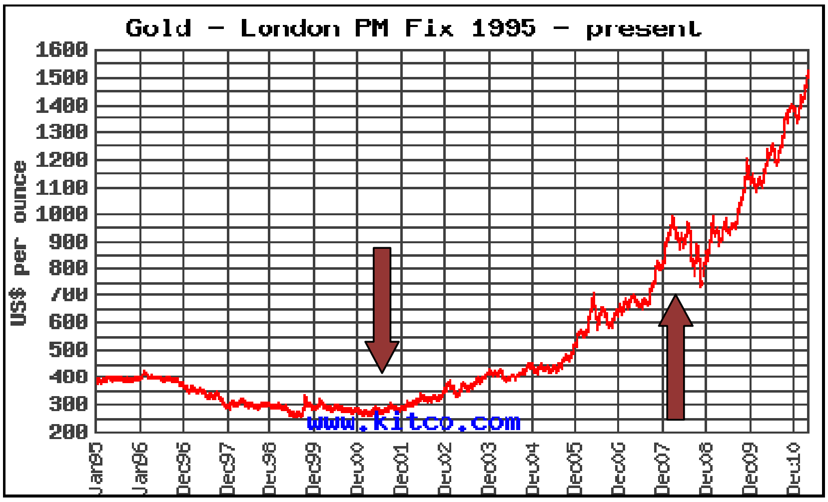

Figure 12. Graph showing the Au price in US$ per ounce from 1995 until December 2010 (Source: www.kitco.com).

The Pt price (figure 9) has shown a variation from US$ 415 to US$ 2,273 in the past ten years. The most noticeable spike is created by the major drop in price during 2008. The Pd price (figure 10) has shown a range of US$ 148 to US$ 858 in the past 10 years, with the most noticeable drop in price during 2001. There was also a downward trend in price during 2008. Both Pt and Pd seems to be showing a general upward trend in price after the major drop in 2008, with the average Pt price around US$ 1700 and the Pd price currently at US$ 740.

The Rh price (figure 11) shows some instability during the 2000 - 2001 period, and a major drop during 2008. The price reached a record high (US$ 9,745) in July 2008 and dropped down to US$ 991 in January 2009. The average Rh price has now stabilised at around US$ 2,300. Rh prices have varied between US$ 444 and US$ 9,745 in the past 10 years.

The Au price (figure 12) has shown a very gradual increase in the last 10 years, and it is on a steady upward trend at the moment. Prices have varied between US$ 265 and US$ 1,541. The influence of the global economic crisis during 2008 is also evident in the Au price, but does not seem as dramatic as with the other commodities discussed.

The yearly averages of the US$/ZAR exchange rate have been plotted in figure 13. From the graph it can be seen that the ZAR gradually weakened to the US$ from 1992 to 2000, when the ZAR plummeted (crashed) from around R6 to R13 to the US$ during 2001-2002. It recovered during 2003 back to between R6 and R7 per US$ and has since stabilised to an average of R7.2 to the US$.

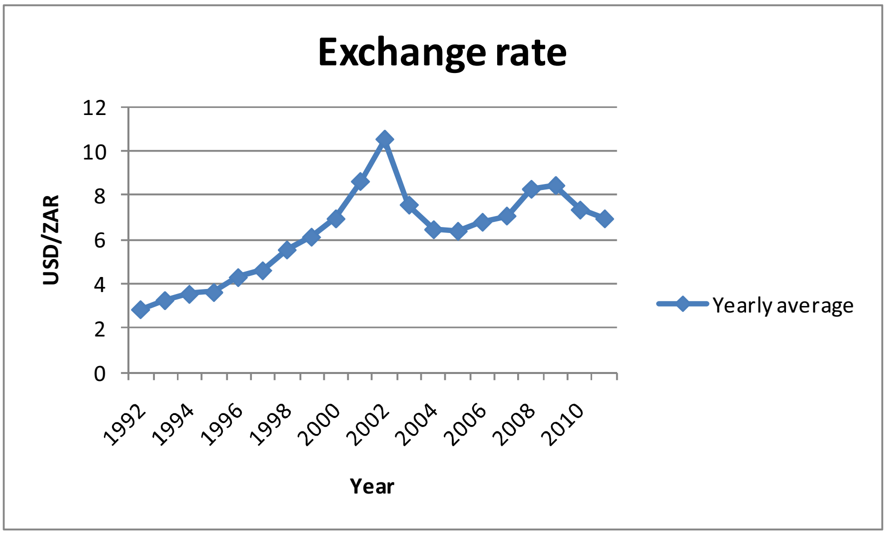

Figure 13. Graph showing the yearly average exchange rate (USD/ZAR) from 1992 until May 2011 (Source: data from www.oanda.com).

Inflation refers to the reduction in the value of money or the general rise in price levels. Reasons for prices getting higher could be as a result of an increased demand for the service or goods; because of higher taxes; higher costs of raw materials; or production costs.

Governments and central banks aim to maintain a low inflation rate as it could seriously impact on the country's economy. Low inflation tends to create better conditions for borrowing money from institutions and promotes consumer spending. High inflation has the opposite effect and usually hampers the growth of the economy by discouraging foreign investment and could also affect the confidence in the country's currency. High inflation is managed by raising the interest rates, but the consequence could be that no new money is created (www.global-rates.com, 2011).

The interest rate is the rate at which banks can borrow money from the central bank. The current interest rate (June 2011) in South Africa is 5.5 % (www.global-rates.com, 2011).

The consumer price index (CPI) is an important inflation figure that most countries calculate and publish annually. It is therefore comparable between countries and can be used to adjust salaries and monitor prices. The CPI is given as a percentage and is a measure of the average price that consumers spend on goods and services. The prices of goods and services are collected and weighted on the share in average consumer spending.

The current rate of inflation (CPI) in South Africa calculated for the month of June 2011 is 0.518 %. Figure 14 below show the change in yearly inflation rates (CPI) for South Africa as calculated from May 2005 to May 2011. The global economic crisis (2007 - 2008) is clearly evident in the inflation rates.

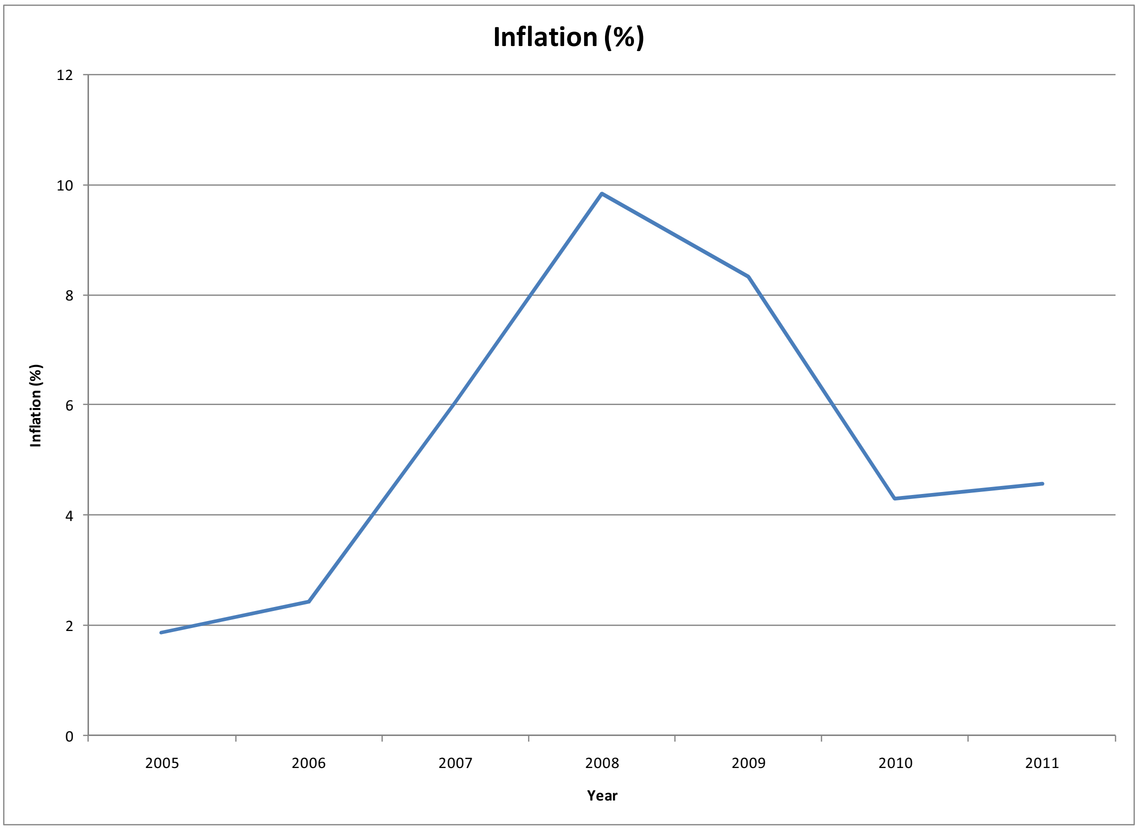

Figure 14. Graph showing the yearly inflation rate percentages (CPI) for South Africa from May 2005 to May 2011 (Source: data from www.global-rates.com).

From the above discussion it can be seen that all these market and economic factors are linked. Trends can be followed through when comparing the different economical aspects. Market analysts use these trends and history to make predictions for the future. These are very important assumptions that are then used by financial evaluators and companies to plan their activities and predict risks as well as profits.

3. PLATINUM MINING IN SOUTH AFRICA

3.1. HISTORY AND CHALLENGES

Mining in RSA has a long history, starting in the mid-1800's with copper mining in Namaqualand. The discovery of diamonds in 1867 near Kimberley and the discovery of gold in the Witwatersrand during 1886 ensured that RSA rapidly became a top mineral supplier to the world. This growth in the mining industry was accelerated with the discovery of coal, and then platinum in the Bushveld Complex during 1924 (Minerals Bureau, 1998).

Initially PGM mining in RSA was based on the extensive knowledge of traditional narrow reef gold mining techniques and many already skilled workers. Even the experience of gold processing and gravity concentration was just applied to the PGM, but as recoveries were generally low the process was quickly improved to a milling and floatation operation. At first all the concentrate was sent overseas for refining, but from the 1980's all base metals and platinum ores were refined locally (Minerals Bureau, 1998; Cramer, 2000).

Platinum-bearing dunite pipes in the Eastern Limb (Wagner, 1926) marked the start of PGM mining, while mining on the Western Limb (Rustenburg-Union-Amandelbult) started with the MR. The shallow MR resource is now becoming depleted on the Western Limb and the shallow UG2 resource is being exploited as replacement ore.

Cramer (2000) mentioned some future challenges (risks) that could potentially face the PGM mining industry in RSA, and his main concern was the high operating costs and low extraction efficiencies.

He (Cramer, 2000) elaborates by mentioning the additional costs that will be incurred with mining deeper to access the UG2 as well as the remaining MR resources on the Western Limb. The refrigeration and ventilation costs will increase as well as the increased safety risk with greater rock pressures and virgin rock temperatures. He also discusses the fact that profitable MR ore reserves are being depleted and that mining will have to focus on the UG2 and the Eastern Limb.

The Eastern Limb generally lacks infrastructure and will result in more capital intensive projects, which could limit the profitability of the company while these projects are being developed. Cramer (2000) also mentioned the fact that the MR on the Eastern Limb has a lower grade and is generally less attractive than the UG2. The Eastern Limb UG2 has lower Pt to Pd ratios therefore with the historical commodity price structure, development on the Western Limb is more favourable.

His other concerns included the labour intensiveness of the RSA mining industry and the fact that new technology is available, but not being utilized to its full potential because of the historically large amounts of un-skilled or semi-skilled labour available. He also mentioned that training programs and technology will have to be implemented to increase the productivity of labour and the potential impact on companies' labour force with the HIV/AIDS crisis must be considered.

Van den Berg (2008) also discussed some key challenges that face the RSA platinum mining industry. These include the challenge of producing at the lowest costs; building a high performance culture with employees; implementing new technologies; the lacking infrastructure on the Eastern Limb; managing the ore mix at current operations and keeping the costs in line while deepening the mines; and the importance of building the companies' investment brand name.

Smith et al. (2008a) also mentions the fact that mining operations on the Western Limb are maturing, and that the focus is shifting to the previously under developed Eastern Limb. The reason for the delay with mining on the Eastern Limb is because of the lower average grades and higher palladium to platinum ratios in the UG2, as well as more extensive potholing in the MR.

While this shift is taking place, it has become evident that the existing regional infrastructure in the Eastern Limb is insufficient to support a long-term mining industry investment similar to the Rustenburg area.

Infrastructure is considered to be the structural elements of the economy, including the road and rail transport systems; water supply and resources; electrical power generation and transmission; telecommunications; educational facilities; and solid waste treatment and disposal. The other functions associated with infrastructural development include the operating procedures; management practices; and developmental policies that will facilitate the effective utilization and development as per the social demand (Smith et al., 2008a).

Smith et al. (2008a) further relates a lack of infrastructure to ineffective social and economic community development resulting in areas with a high level of poverty. Rectifying the legacy of corruption and inefficient service delivery in these old homeland provinces is an ongoing challenge. The majority of the people are unemployed, unskilled or semi-skilled and living in poverty, which has significant socio-economic challenges to the sustainable provision of services and infrastructure development.

The platinum mining industry and the provincial governments are jointly working on addressing these challenges. One initiative is to promote development clusters by improving education and skills; providing the essential infrastructure; opening access to capital; building capacity in technology; and improving institutional efficiency.

One of the main challenges for communities and industry in the Eastern Limb is the availability of water. This challenge is being addressed through the establishment of the Lebalelo Water Users Association (LWUA). They are working in conjunction with the Department of Water Affairs and Forestry (DWAF). Plans are being executed to produce a sustainable water supply for the area, which will benefit the mining operations as well as the local population. The construction of the De Hoop dam is part of this scheme (Smith et al., 2008a).

Spatial development through the establishment of integrated and effective development at a municipal level is critical to the success of any area. The Steelpoort Valley Producers Forum have been established to co-ordinate and develop joint strategies between the government and mining sector that will ensure sustainable local economic development. They will assist mainly with housing development and service provision (Smith et al., 2008a).

3.2. COUNTRY RISK

A potential high risk in any investment or new project is the country risk. This includes the applicable laws and regulations that will have to be complied with as well as the political stability and local government relations.

Various institutions conduct surveys and annually publish their findings on countries' risks rankings. This can be compiled by any country or financial institution for example the World Bank. Figure 15 is an example of the world risk ratings per country. This example is from Euromoney (ECR, 2011) and their country risk analysis was based on economic, political and structural assessments as well as the access to capital, credit ratings and debt indicators. The full methodology is discussed on their website (www.euromoney.com).

This update was compiled on 8 March 2011 and Canada and Australia is top (very low risk) at 86.35 and 85.36 points respectively (Tier 1) and the United States of America (USA) scored 81.78 (Tier 2). Comparatively South Africa scored 59.2 and Russia 56.98 (Tier 3), but other African countries scored much lower and are therefore considered higher risk. The Democratic Republic of Congo (DRC) scored 22.51; and Zimbabwe is very low (high risk) on 16.87 (Tier 4). The highest risk countries (Tier 5) in the World according to this survey are Burundi and the Central African Republic with 8.35 and 7.55 respectively (ECR, 2011).

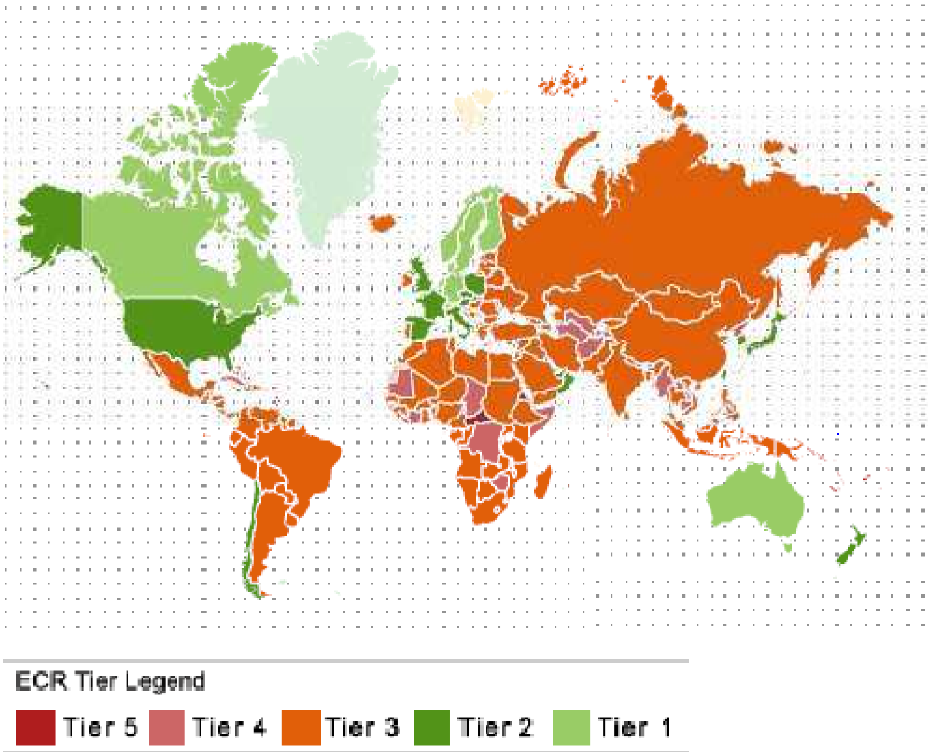

Figure 15. Map showing World Risk Ratings per country as calculated on 8 March 2011. (Source: www.euromoneycountryrisk.com).

The economies of the world are so interdependent that all countries felt the effects of the 2008 - 2009 global financial crises. The South African economy showed a significant reduction in growth during 2009.

The International Monetary Fund (IMF) reported in January 2010 that there are already signs of the recovery of the GDP in SA. International analysts seem to be certain that SA is *'well-placed to bounce back from the crises'*. They mention that SA has a '*large economy with solid fundamentals and sound financial systems'* (WEF, 2009).

The World Economic Forum (WEF, 2009) suggests that SA should address labour-related issues such as the uneducated workforce; labour market efficiency; health; and poor labour-employer relations. They identified that the biggest barrier to SA's competitiveness are '*the perceptions around the business costs of crime and violence*'.

The WEF ranked SA with regards to their Global Competitive Index as the '*highest ranked country in sub-Saharan Africa'* (45^{th} out of 133 countries, September 2009). SA performed strongly in the categories for innovation and the countries' reporting and auditing systems.

The Heritage Foundation (January 2009) complemented SA on their advanced financial systems and scored high in the economic freedom category, but scored SA lowest in the fiscal freedom category. They specifically mentioned that '*the judicial system is slow, and race laws and unclear regulations hamper foreign investment.*' (WEF, 2009).

There is a general development cooperation agreement in place between SA and Canada since 2006 (CIDA, 2011). The goals of this agreement have been aligned with the SA Government's main priority areas, which includes the strengthening of regulatory and public administration by sharing of relevant experience between the Canadian International Development Agency (CIDA) and SA. The other main goal is to deliver better health services to people and specifically CIDA assistance with the treatment, care, support and prevention of HIV/AIDS in the country.

CIDA (2011) reports that SA is the biggest and most advanced economy in Africa, they feel that SA has '*sound constitutional and legal policies but lacks the capacity to implement them effectively'*. They suggest that the main reason for this is sub-standard education resulting is a skills shortage and high unemployment rates.

3.3. TAXATION

Taxation in South Africa is managed and collected by the South African Revenue Services (SARS). The main taxes and levies that will apply to a company doing business or managing a business in South Africa are: income tax; value added tax (VAT); capital gains tax; secondary tax on companies (STC); pay-as-you-earn (PAYE); Unemployment Insurance Fund (UIF); and the skills development levy (SDL).

These taxes and levies are fully explained on the SARS website (www.sars.gov.za). Normal income tax, PAYE, UIF and the SDL will be applicable to the employees of the company while the VAT (currently 14 %) is South Africa's main source of indirect taxation revenue and is payable on all goods or products that are purchased, or services delivered in the country.

Mining companies registered and trading in South Africa pay normal income tax on their taxable income currently calculated at a flat rate of 28 %. Companies are also liable for STC which is taxed at a rate of 10 % on the excess of dividends paid to shareholders in a twelve month period (Thornton, 2010).

Capital gains tax is currently levied at 14 % on the disposal of fixed assets and the gains or losses made on the sale of shares held for more than three years (Thornton, 2010).

The Price Waterhouse Cooper fourth annual total tax contribution survey that was released in June 2011 showed that South Africa's large companies contributed 23.51 % of the total government tax receipts for 2010. The average total tax rate contributed by these companies was 33.24 % and was calculated by looking at the business taxes compared to the profit before tax. The survey also determined that the mining industry contributed with a total tax rate of 38.36 %, which was higher than the manufacturing industry contributions (Business LIVE, 2011).

SARS also makes provision for capital expenditure allowances and deductable operating expenditure. Capital expenditure includes factory plant and buildings; vehicles; computers and research; prospecting and capital development expenditure. Deductible expenditure includes rehabilitation expenses actually incurred as well as contributions to an approved rehabilitation trust (Thornton, 2010).

3.4. MINING LEGISLATION IN SOUTH AFRICA

The Mineral and Petroleum Resources Development Act (MPRDA) of 2002 is the current governing mining legislation in South Africa. The purpose of this Act is '*to make provision for equitable access to and sustainable development of the nation's mineral and petroleum resources…*'. The MPRDA (2002) clearly states that the State is the custodian of all the mineral and petroleum resources in South Africa and that the State will manage this to the benefit of all the people of South Africa.

The Minister is also tasked to ensure that the national environmental policies (as set out in NEMA, 1998), and standards are utilized and adhered to, in order to promote economic and social development, and ultimately sustainable development of the natural resources (MPRDA, 2002).

Chapter 2 of the Act discusses the fundamental principles and mentions the nine main objectives of the Act. The mineral and environmental regulations are set out in chapter 4. Section 23 (6) states that a granted mining right is valid for a period that may not exceed 30 years. A mining right may be renewed by lodging an application to the Minister, a mining right may be renewed for further periods but each period may not exceed 30 years (section 24 (4), MPRDA, 2002).

Section 25 deals with the obligations and rights of a mining rights holder, 2 (f) mentions the social and labour plan must be complied to, and (g) that the holder of the mining right must pay the State royalties (MPRDA, 2002).

The Mineral and Petroleum Resources Royalties Act (2008) came into effect 1 March 2010 and the main purpose is *'to impose a royalty on the transfer of mineral resources and to provide for matters connected therewith.'*

Any person that recovers a mineral resource from the Republic must pay a royalty for the benefit of the National Revenue Fund (section 2, Royalties Act, 2008). The amount that must be paid will be calculated according to a formula in section 4 (1) or (2) depending on the type of resource – refined or unrefined. For Pt, concentrate is classified as unrefined and Pt that has been refined and smelted to 99.9 % purity is classified as refined, as per the definition in section 3 of the Royalties Act (2008).

The MPRDA also discusses the Minister's rights and powers. The Minister is appointed by the State and acts on behalf of the State regarding matters pertaining to the MPRDA (sections 47 and 49). In chapter 7, where general and miscellaneous provisions are discussed, section 91 and 92 refers to the permission (power) to enter any mining area and conduct routine inspections. Section 93 sets out the steps that must be followed if any contravention of the Act or conditions of a permit is not complied with. These include the immediate rectifying of the contravention or immediate suspension or termination of activities until instructions are complied with (1 (b)(i) and (ii), MPRDA, 2002).

Section 100 of the MPRDA refers to transformation of the minerals industry and mentions that the Minister must '*develop a broad-based social-economic empowerment Charter that will ensure the entry of historically disadvantaged South Africans into the mining industry and thus allow them to benefit from the exploitation of mining and mineral resources.*'

This Charter was first published in the Government Gazette on 13 August 2004. The Mining Charter set out the '*scorecard for the broad based socio-economic empowerment charter for the South African mining industry*' where the scope, objectives and all targets was set out and described.

The Mining Charter was updated in 2010 (amendment of the broad based socio-economic empowerment charter for the South African mining industry, 2010) after an audit revealed that the first target that was set out was not reached. According to Susan Shabangu (Mineral Resources Minister) companies were supposed to sell 15 % of their South African assets to black investors by 2009, but black-economic empowerment (BEE) ownership is currently only 8.9 % (Shabangu, 2010; Mail & Guardian online, 2010).

The revised Charter requires 26 % of mining assets to be BEE owned (same as original targets) by 2014. Companies must procure at least 40 % of capital goods from BEE owned businesses. 70 % of services and 50 % of consumer goods must also be obtained from BEE companies by 2014. A company that does not comply will face penalties that could include revoking of a company's mining license (Shabangu, 2010; Mail & Guardian online, 2010).

The Department of Mineral Resources (DMR) has been tasked to administer the Mine Health and Safety Act (MHSA, 1996). The MHSA has been specifically drafted for regulating health and safety in the mining industry.

The MHSA main objectives are '*to provide for protection of the health and safety of employees or other persons at mines*…..' The MHSA describes the duties and responsibilities of the employer (owner or appointed manager, chapter 2), as well as the obligations and rights of the employee (section 22 and 23).

The MHSA also requires each mine with 20 or more employees to have health and safety representatives on each shift and at each working place. The MHSA mentions procedures regarding negotiations between the employer and trade unions that represent the employees to discuss matters concerning the general health and safety at the mine (chapter 3).

Chapter 5 of the MHSA deals with the Inspectorate's appointment/ establishment as well as the duties and powers assigned to the appointed Chief Inspector of Mines. The main function is to ensure that the MHSA is complied with and enforced. Other duties include the appointment of other inspectors and a medical inspector; to determine and implement policies that will promote health and safety at mines and to compile necessary progress and annual reports.

The appointed Inspector may enter any mine at any time with or without notice and conduct an inspection of the activities, facilities or documentation on any matter related to the MHSA (section 50). Section 54 gives the Inspector the right to give any instruction to protect the health or safety of persons at the mine if he has reason to believe the health or safety might be endangered. These instructions include suspending certain activities or part of a mine until the risk has been adequately dealt with or the management provides sufficient proof that the risk has been addressed.

The Inspector may also recommend an administrative fine or if the breach is specific to certain sections of the MHSA the responsible employee can be criminally charged (chapter 7).

Chapter 8 discusses general provisions of the MHSA applicable to the Minister with regards to delegation of power to other persons as well as the power to (after consultation) make regulations regarding matters dealt with in the MHSA. Important definitions for words and phrases used in the MHSA are given in section 102.

Environmental regulations are contained in the National Environmental Management Act (the NEMA, 1998). NEMA addresses three main environmental areas of concern, namely resource conservation and exploitation; pollution control and waste management; and land use planning and development. These regulations are underpinned by the globally accepted concept of sustainable development (Glazewski, 2005).

4. STRATEGIC PLANNING AND PROJECT EVALUATION

4.1. STRATEGIC PLANNING

There are various elements that must be considered in project initiation and selection to assist executives and managers to determine in which mining project to invest. Managers of companies are under constant pressure to ensure projects are successful and profitable to maximize shareholder value.

The corporate management of a company must develop an objective statement that can guide the planning of the company in terms of the expectations of shareholders as well as market demand and long term goals. This is called the company's strategic business plan (SBP) or strategic long term plan (LTP).

Smith and Ballington (2005) define two types of mine planning – strategic and tactical mine planning. Strategic mine planning incorporates decisions and components that affect the long term value of the business. The main feature of this is the development of the business model for the company, in which the plan for extracting the mineral resource is outlined. This plan will include all the fundamental factors associated with extracting and optimizing the company's resources. Starting with the exploration strategy; extraction method; mining sequence; cut off grades; and moving to the scale of the operations; metallurgical processing; social and labour plans; environmental philosophy; sustainable development; deciding on the marketing and sales strategy to sell the product.

This business model and strategy must be continuously revised and updated by the company to ensure that changes in external environment and new information is incorporated and to make sure of the alignment with the long term strategic objectives. The strategic mine plan then forms the base line to assess new options and projects (replacement or expansion opportunities) to ensure comparability and alignment with the company's strategic objectives and long term goals (Smith and Ballington, 2005).

Tactical mine planning is the '*routine planning activities*' or short term planning, needed to carry on with existing operations or successfully implement the approved new projects. This will include the annual budgets; production schedule; and mineral processing results. Each of these routine activities must be assessed and measured against the long term strategic objectives (Smith and Ballington, 2005).

Griesel (2004) groups mining projects based on the type of project – expansion capital projects; ongoing capital projects; health and safety related projects; risk related projects; and environmental projects.

The selection of the correct 'mix' of projects will then depend on the company's SBP as defined by the top management. The onus is then placed on the company's operational level for the detailed planning and meeting of these objectives. All projects then proposed by the project team will undergo lengthy reviews and scrutiny before any capital is approved for project initiation (Griesel, 2004).

The main challenge facing management is to ensure that the investments are aligned with the SBP and to ensure the viability of current operations are not compromised (Smith et al., 2006).

Smith et al. (2006) discusses the processes, tools and techniques applied by Anglo Platinum Limited to determine project value and selecting the correct investment that guarantee alignment with the SBP.

The first step in the strategic long term planning process is the compilation of the mine extraction strategy (MES). This MES addresses the basic rules that will set the foundation of the LTP. The MES must show how the mineral resource will be exploited within the mining right area, the time period in which this will occur and what the cost (capital and operating) will be (Smith et al., 2006 and 2009).

The main issues that must be determined include the optimal scale of the potential operation; tonnage sources and splits if multiple reef horizons are planned; technology and mining layouts; any critical constraints to the proposed operation; and potential influences of the existing asset base to the proposed operation.

The MES will be used as the base for the development of the mining right plan (MRP). The MRP is the physical depletion plan that will cover the entire mining right of the project/operation. The emphasis is on planning the extraction of the available resource in the most optimal manner, using technically sound methodology; appropriate capital and operating cost estimates and general global planning parameters. No time constraints are applied, thus the optimal life span can be determined. It is not necessary for this plan to be economically viable across the entire mining right area, but rather to establish the optimal criteria for extraction of the resource. This MRP must be updated annually to incorporate updated global assumptions, technologies and costs (Smith et al., 2006 and 2009).

From the MRP the best exploitation option is selected and an optimized economic plan is created, the LTP. The LTP comprises production, operating cost and capital cost estimates for the first two 30 year periods of the life of operation. The 30 years is consistent with the granting of mining rights according to the MPRDA. This will provide the basis for establishing the concentrating, smelting and refining capacity requirements. The production profile will also determine the requirements of services and support infrastructure to successfully execute the LTP (Smith et al., 2006 and 2009).

The LTP from each operation and project forms the basis of the company's production and cost forecasts and is used for capital prioritization and value optimization process. In the capital prioritization process all operations and projects are categorized based on their confidence level (scoping to feasibility, figure 16) and strategic objective. Strategic objectives can include retention of mineral rights; critical path projects; sustainable development and targeted growth rates. The projects are also ranked according to their value forecast (value and rate of return) within their confidence category. Criteria based on market supply and demand trends; metal pricing forecasts; overall business returns; and debt/equity ratios are used to test each project (Smith et al., 2006 and 2009).

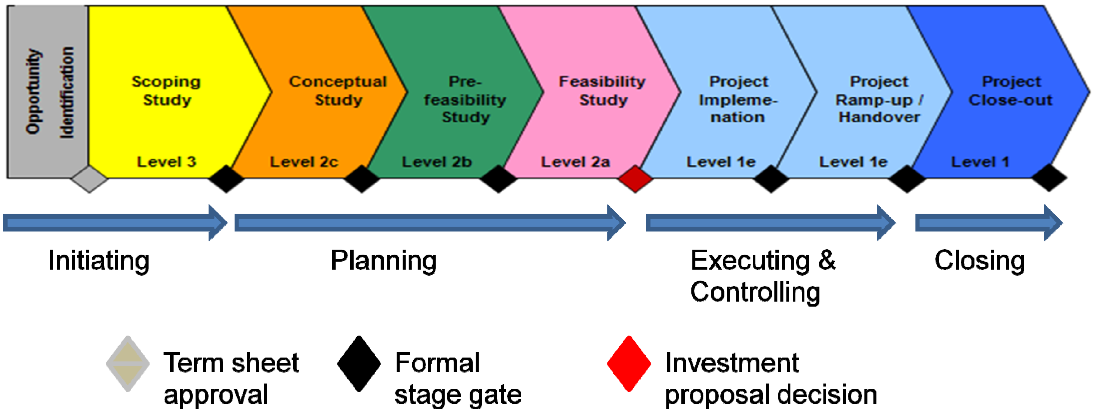

Figure 16. Schematic diagram showing the project timelines used by Anglo Platinum (Smith et al., 2006).

From the forecasted production outputs and timelines, the concentrating; smelting; converting and refining, projects are identified and motivated. This process now ensures that the best value projects, which comply with the strategic objectives, are aligned with the processing, and strategic targets can be achieved (Smith et al., 2006 and 2009).

Two types of capital costs are defined: firstly, projects or proposed capital investments of a 'once off' nature, that is mainly for expansion (increasing production output) or replacement (maintaining total business capacity) projects. Secondly, stay in business (SIB) capital that must maintain the existing assets and remain relatively stable from year to year, the normal run of business expenditure. SIB capital can be used for replacing individual assets like machinery, vehicles or other equipment; or business improvement initiatives; risk mitigation initiatives; for upgrading or developing shared infrastructure; and ore reserve development (Smith et al., 2009).

Data validation, auditing and comprehensive risk assessments are conducted on all input data to best ensure the overall integrity of the planning process. This is part of the requirements for due diligence and corporate governance. Good business practice, consistency within the company, quality assurance or correctness of data and the mitigation of risks are all required to ensure the LTP is of the best possible quality. The skills development and identification of skill shortages and training requirements will also be identified and can be planned or mitigated in the LTP process (Smith et al., 2009).

These audits are done by first engaging with each technical discipline that supplies input data, for example: mining; processing; geology; ventilation; rock engineering; human resources; and finance. These single disciplinary audits are conducted to ensure input data is viable and sound. This is followed by a multi-disciplinary audit assessing the consolidated LTP. The technical content, achievability, practicality, continuity, integrity and integration of the LTP are reviewed and a qualitative risk assessment is conducted to determine any potential threat to achieving the LTP (Smith et al., 2009).

4.2. PROJECT EVALUATION

Anglo Platinum uses Hyperion Strategic Finance (HSF) software that has been extensively customized to meet the company's requirements to evaluate and select investment options for their production profile (Smith et al., 2006). The HSF valuation package is based on a discounted cash flow (DCF) analysis/prediction, where all input data and general assumptions is stored on a centralized data warehouse for evaluation of all project or operation plans (Smith et al., 2006 and 2009).

A DCF is the process of finding the present value of future amounts by assuming an opportunity to earn a certain return on the money (Gitman, 2006). The present value is the current rand value of a future amount. The annual rate of return can also be referred to as the discount rate, cost of capital, opportunity cost or required return (Gitman, 2006).

A DCF analysis also takes into account the initial cash investment that will be required to purchase or develop the asset and compares it to the expected future cash flow from this asset (Smith et al., 2006 and 2009). This is done by calculating the net present value (NPV) and internal rate of return (IRR).

The NPV is found by subtracting the project's initial investment amount from the present value obtained from the cash inflows discounted at a rate equal to the company's cost of capital (Gitman, 2006). If the resultant NPV gives a positive rand amount, the project should be accepted as it generates more cash than needed to service its debt and to provide the required rate of return to the shareholders (Griesel, 2004). If the NPV is negative the project will not provide any profit and the risk will be higher.

The IRR is the discount rate that equals the NPV of an investment opportunity with no profit (ZAR 0) or where the present value of the cash inflows equals the initial investment (Gitman, 2006). The IRR also indicates the compound annual rate of return the company will earn if it invests in the project and receives the given cash inflows (Gitman, 2006). If the IRR exceeds the cost of funds used to finance the project, there will be a surplus amount after paying the capital and this will ultimately increase the wealth of the shareholders (Griesel, 2004).

The other factor that can be determined with a DCF is the payback period. The payback period is the amount of time required for a company to recover the initial investment and is calculated from the cash inflows (Gitman, 2006). A shorter payback period is advantageous as the company's exposure to risk on that project becomes less (Griesel, 2004).

Within a company (or mining area) it can happen that when a standalone project is evaluated a negative NPV is the outcome, but when evaluated with neighbouring projects, value is added with this project and gives an overall benefit to the company (Smith et al., 2006 and 2009), therefore rendering the project feasible to the company, in terms of the long term strategic plan.

It is essential that project cash flows are initially calculated in nominal (actual or stated) terms, due to the nature of the South African mining taxation and its treatment of capital investments (Smith et al., 2006 and 2009). This will assist with the accurate calculation of the tax prior to de-escalating to real terms for discounting to the present values. This is done by inputting the values in real money terms (regardless of the time value of money) and then calculating the nominal values by using escalation factors (Smith et al., 2006 and 2009). The tax is calculated and the after tax cash flow is then adjusted for inflation and the NPV calculated at the appropriate real discount rate (Smith et al., 2006 and 2009).

The discount rate or cost of capital is the rate of return that a company must earn on the projects in which it invests to maintain its market value and attract funds. The weighted average cost of capital (WACC) reflects the expected average future cost of funds over the long run. This can be determined by weighting the cost of each specific type of capital by its proportion in the company's capital structure (Gitman, 2006).

The determination of an appropriate discount rate must also take into account potential risk of an investment. Risk in capital budgeting is the chance that a project will prove unacceptable, or, the degree of variability of cash flows from the project. Therefore the worth of a capital expenditure and its impact on the company's value must be viewed in light of risk and return (Gitman, 2006).

A risk adjusted discount rate provides the rate of return that must be earned on a given project to compensate the company's owners adequately by maintaining or improving the company's share price (Gitman, 2006).

Capital refers to the long-term funds of a company and can be split into debt capital and equity capital. Debt capital includes all the long term borrowing, including bonds, which have been incurred by the company and equity capital, which consists of long term funds provided by the company's shareholders or owners (Gitman, 2006). The company can obtain equity capital internally or externally, internally by retained earnings rather than paying out dividends to the shareholders or externally by selling common or preferred stock (Gitman, 2006).

Equity investments in a project represent the risk capital (Griesel, 2004) and usually form the basis of the subscription price for common or preferred stock. The debt/equity ratio for a project is negotiated based on a combination of factors, for example market expectations, industry norms and risk. Lenders will require substantial equity investments in a project if creditworthy guarantors are not available and to ensure the project owners initiate and successfully operate the project to completion (Griesel, 2004).

Uncertainty is another risk factor that is part of a DCF, as the future is uncertain and prediction or forecasting of parameters is necessary when evaluating a project. This factor can be dealt with in the form of scenario planning and thereby producing a set of global assumptions or long term planning parameters (Smith et al., 2008b).

'*When investment decisions are made in the context of these global assumptions, these decisions are positioned with the expectation that this future world view would play out.*' (Smith et al., 2006) It is therefore very important that possible drivers of future change is identified and well understood by the team compiling these global assumptions (Smith et al., 2008b).

Estimations are made in terms of costs (capital and operating), tonnages, grade and recoveries with relative accuracy by technical experts through in-depth studies and analysis. Assumptions with regards to commodity prices, exchange rates, inflation and escalation are more difficult as various external factors can influence these elements. There is also the risk of value destruction if the long term view is too pessimistic, and projects are deemed not viable as a result. This can however be addressed through scenario evaluation and risk profiling (Smith et al., 2009).

Sensitivity and risk analysis is important when there are variables associated with the project evaluation process. The extent and nature of the uncertainty associated with the variables must be considered and form part of the final decision (Griesel, 2004). Changing the variables to determine the projects sensitivity to external parameters will give an indication of the robustness of the project to changing parameters, as well as provide the company with 'leading indicators' that could provide an advance warning that conditions are turning towards a particular scenario (Smith et al., 2008b). This process is equally important for identifying opportunities as it is for avoiding loss exposure (McGill and Theart, 2006).

Project value tracking is an important proactive step to ensure the investment is performing as expected after the decision to proceed with a project has been made. Within Anglo Platinum the project value tracking analysis is in the form of a waterfall chart, which illustrates the important changes to the NPV since the original decision was taken. The internal and external factors that caused the changes are clearly shown (see example in figure 17). This tool assists with providing continuous feedback of the investment performance, as well as assists the top management with decision making, value optimization and capital prioritization (Smith et al., 2006 and 2009).

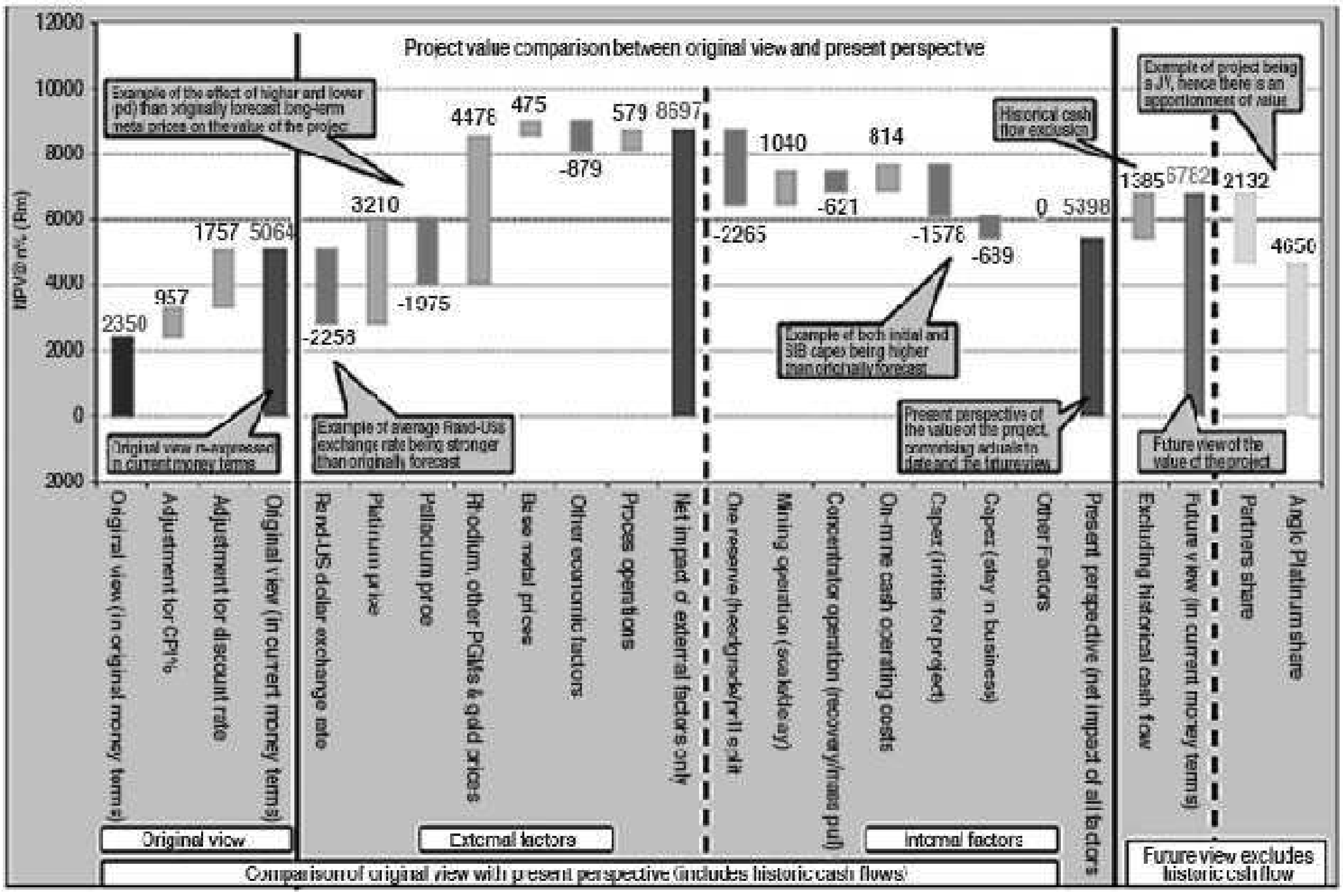

Figure 17. An example of the waterfall chart used to track the project (Smith et al. 2006).

5. GEOLOGY OF THE PROJECT AREA

The TPM Project is located in the north eastern portion of the Eastern Limb of the Bushveld Complex. Norite and gabbronorite rocks of the Main Zone outcrop in the west of the project whereas the anorthosite-norite-pyroxenite cyclic unit of the Upper Critical Zone outcrops to the east of the project area. Both the MR and the UG2 sub-outcrop in the area, with the UG2 exposed on surface on Hackney Hill and Serafa Hill in the north. The reefs strike roughly north east and dip approximately 14° - 16° to the south west. The total strike length on the project is 11 km. Both reefs occur as narrow tabular ore bodies that extend laterally over hundreds of square kilometres, their continuity established from years of exploration and mining in the Eastern Limb.

Figure 18 shows the locality plan for the TPM Project, indicating the sub-outcrops for the MR (red) and UG2 (green), as well as the outline of the areas under investigation for this evaluation. The two polygons represent the proposed first phase of the project, where the down dip limit represents the boundary where temperatures underground will be below 32.5 ℃ and no (artificia l) refrigeration will be required.

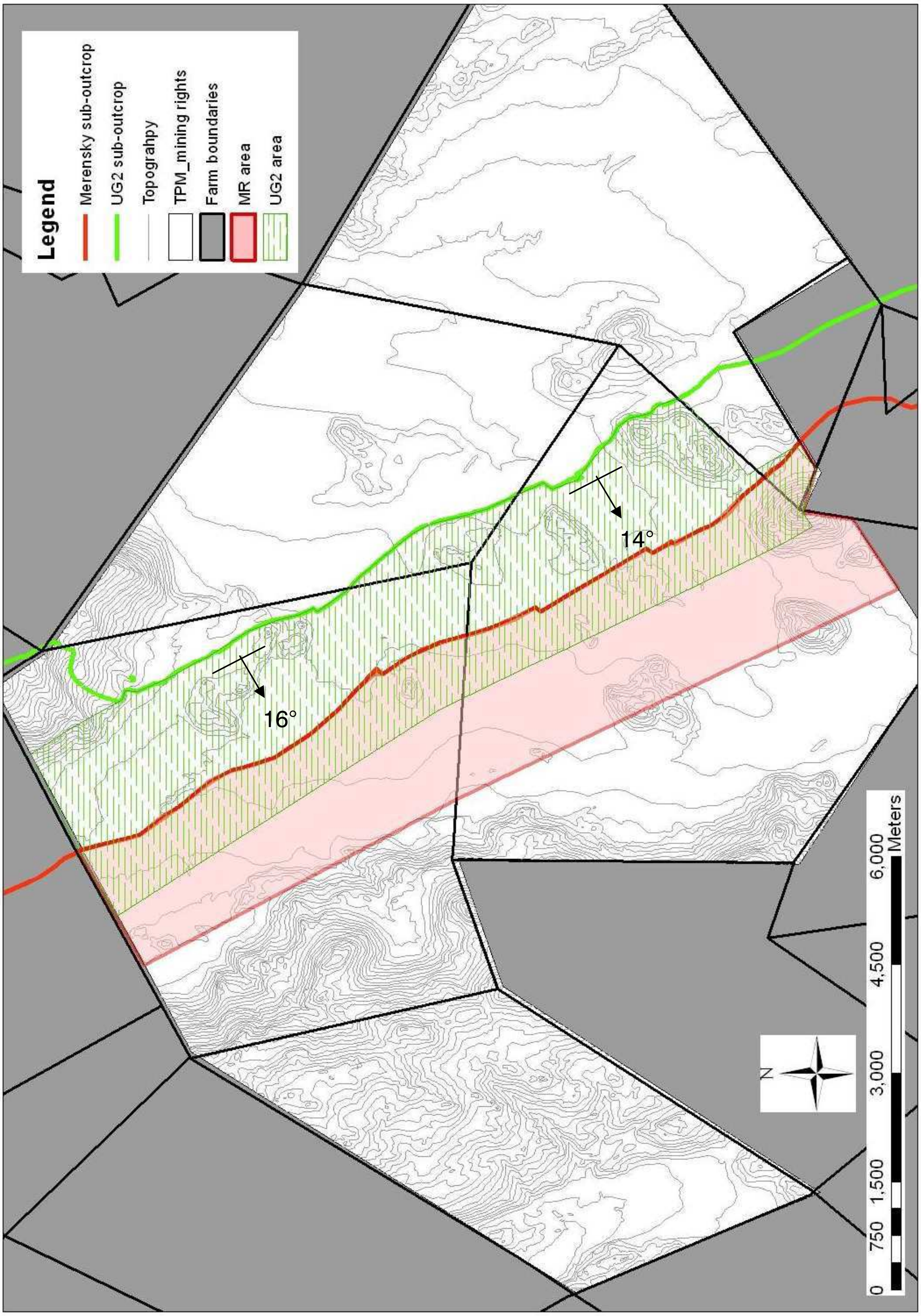

Figure 18. TPM Project locality plan.

The regional structure of the North Eastern limb is characterized by steeply dipping NNE and EW trending dykes and faults/fractures. These structural trends can be clearly identified on the aeromagnetic survey as well as from surface outcrop and underground mapping.

Potholes and replacement pegmatites are found across the area. These features have been intersected by surface boreholes as well as confirmed in the underground mines.

The structural plan for the TPM Project is shown in figure 19 and clearly shows the distribution of dykes, faults, potholes and replacement pegmatites. These are also the main features that are taken into account when geological loss is calculated and the structural model is the main platform for the mine design and planning of the underground mine.

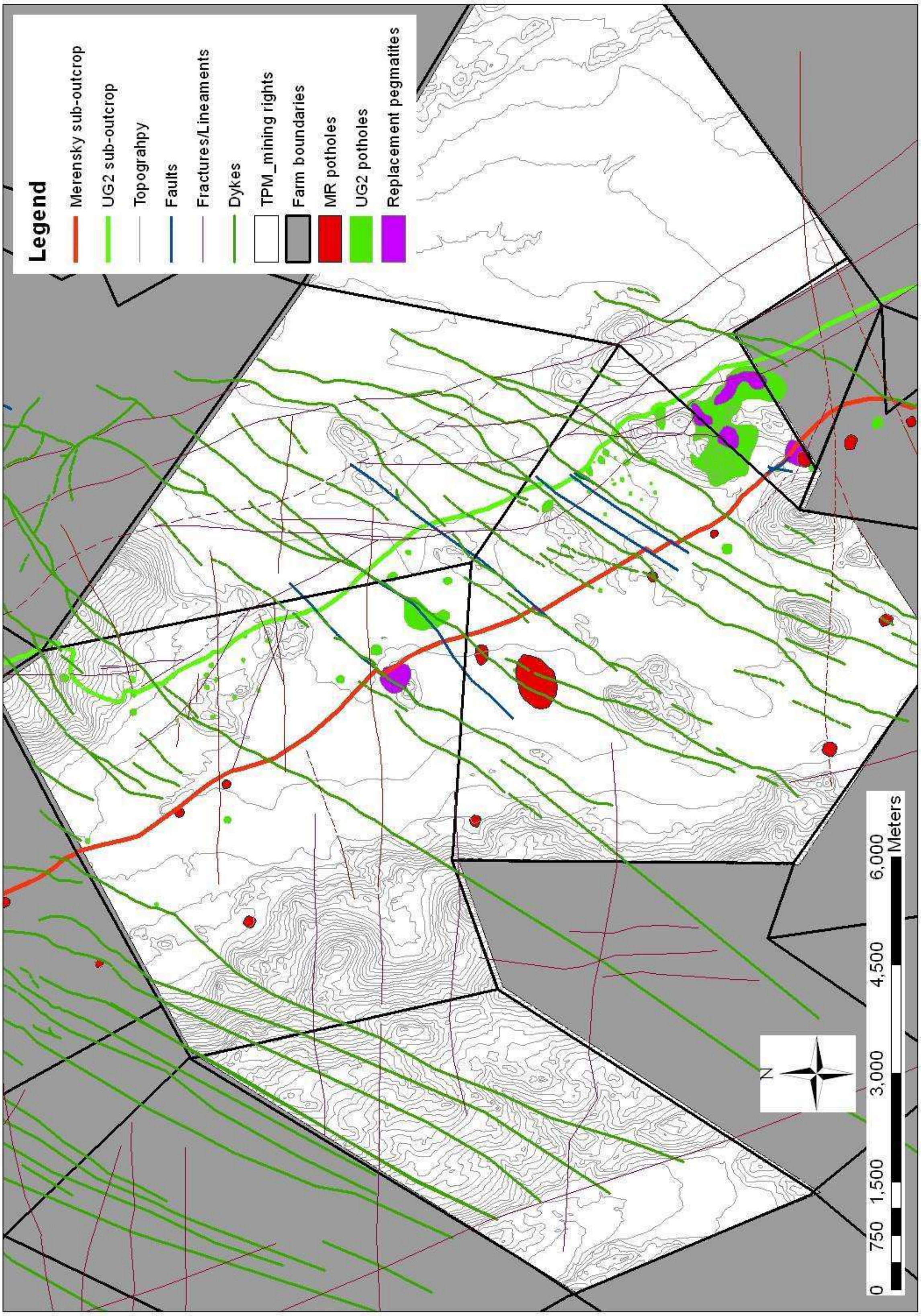

Figure 19. Generalised structural plan for the TPM project.

5.1. STRATIGRAPHY

The general stratigraphy of the TPM Project is shown in figure 20. The top of the Upper Critical Zone of the Rustenburg Layered Suite is marked by a 35 - 60 m wide poikilitic anorthosite, known as the giant poikilitic anorthosite or GPA, which gradationally becomes a norite and then grades into a poikilitic plagioclase pyroxenite. This first pyroxenite unit is approximately 4 – 5 m wide and is also referred to as the Bastard pyroxenite as it is commonly confused with the MR. There may be a thin chromitite stringer on the bottom contact (Cameron, 1982).

The footwall of the Bastard pyroxenite is a poikilitic pyroxene anorthosite that grades into a norite. The second poikilitic plagioclase pyroxenite unit varies in width from about 2 m to 10 m wide and grades into a pegmatoidal plagioclase pyroxenite which can be up to 0.5 m wide. This unit is mineralized and referred to as the Merensky pyroxenite. The MR that can be mined for PGM is only a 60 – 100 cm portion within this unit. The main mineralization occurs about 50 - 70 cm below the top of the pyroxenite and is usually bound by two thin (0.5 cm wide) chromitite stringers. The pegmatoidal plagioclase pyroxenite commonly marks the footwall of the MR.

Below the Merensky pyroxenite unit, a 390 m wide gabbronorite-norite-anorthosite sequence separates the MR with the UG2. The bottom of this unit is usually demarcated with a sharp contact between norite and poikilitic plagioclase pyroxenite, a thin chrome stringer is sometimes present at this contact.

This poikilitic plagioclase pyroxenite unit is about 15 m wide and hosts three separate chromitite bands. The UG3B and UG3A are 10 – 15 cm wide poorly developed/disseminated chromite bands, approximately 5 m below the start of the unit and 0.5 m apart. About 10 m below these two chromitite bands is another, usually well developed 15 - 20 cm wide chromitite band referred to as the UG3. The UG3 is underlain by a poikilitic anorthosite about 1 m wide, and serves as a regional marker horizon. A 25 m wide norite grades into a poikilitic plagioclase pyroxenite characterized by various thin (less than 1 cm wide) chromitite stringers. This 5 m wide pyroxenite is the hanging wall of the UG2.

The UG2 is a 61 cm wide chromitite band, with a characteristic pegmatoidal plagioclase pyroxenite of about 0.7 - 1 m wide below it. The PGM mineralization occurs within the chromitite and occasionally extends into the footwall. The pegmatoidal plagioclase pyroxenite grades into a poikilitic plagioclase pyroxenite and extends for about 7 – 15 m, where it then stops abruptly. The next unit is a poikilitic pyroxene anorthosite known as the 'Footwall Marker'. It is about 2 m wide and grades into a 70 – 80 m wide norite sequence.

The norite grades into a 20 m wide poikilitic plagioclase pyroxenite, which is the immediate hanging wall of the Upper Group 1 (UG1) chromitite. The UG1 is characterized by a 1 m wide chromitite band with numerous bifurcating chromitite stringers into the anorthosite footwall. This zone of chromitite blebs, lenses, bifurcations and stringers in the anorthosite is usually about 5 m wide. The UG1 is not well mineralized and not mined for PGM.

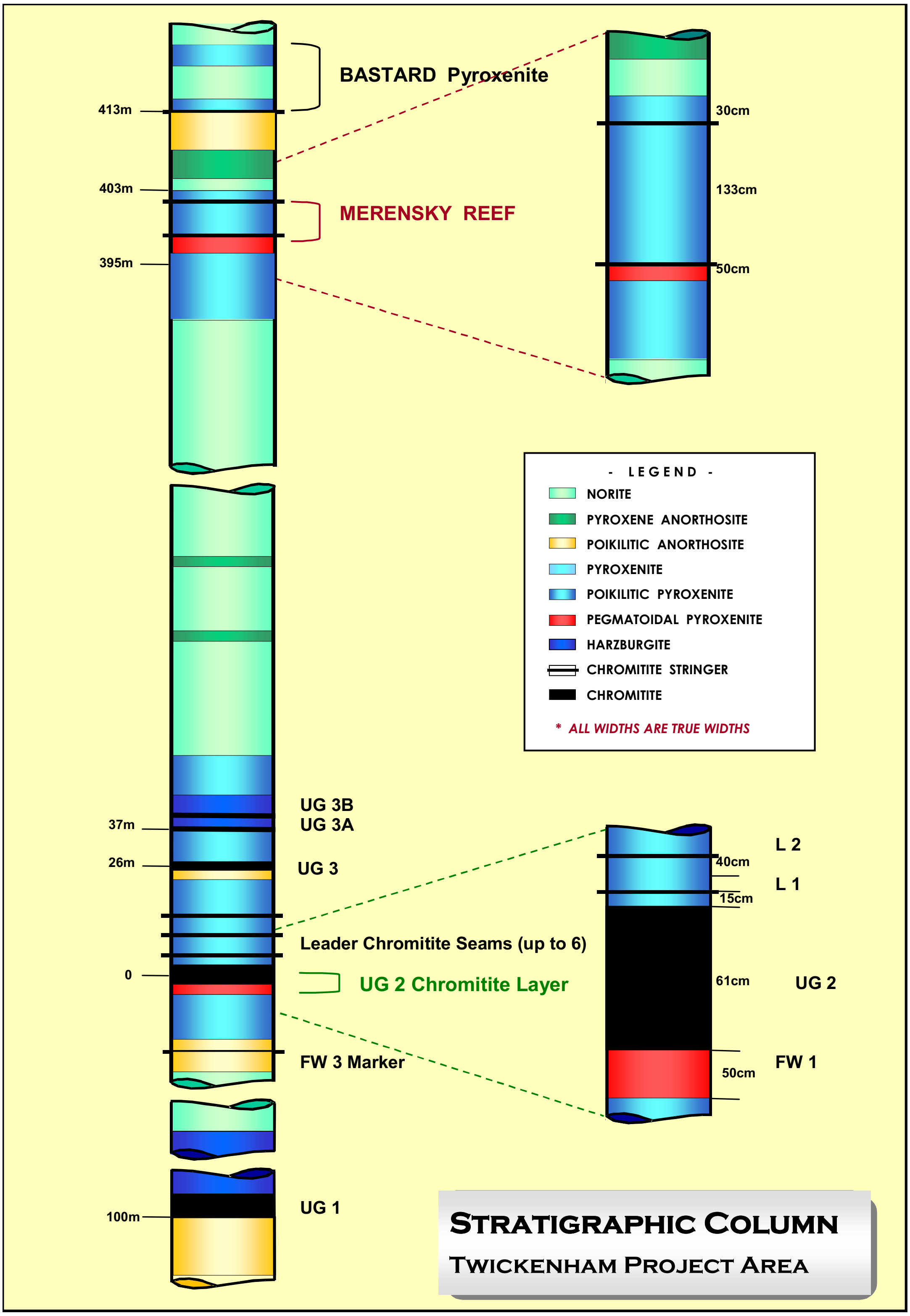

Figure 20. Generalised stratigraphic column for the TPM Project.

The Eastern Limb stratigraphy from the GPA to the UG1, as described above, is similar to that of the Western Limb. However, there are distinct differences in thickness and middling between these units. There are also subtle differences in the distribution and location of the PGM mineralization (figure 21).

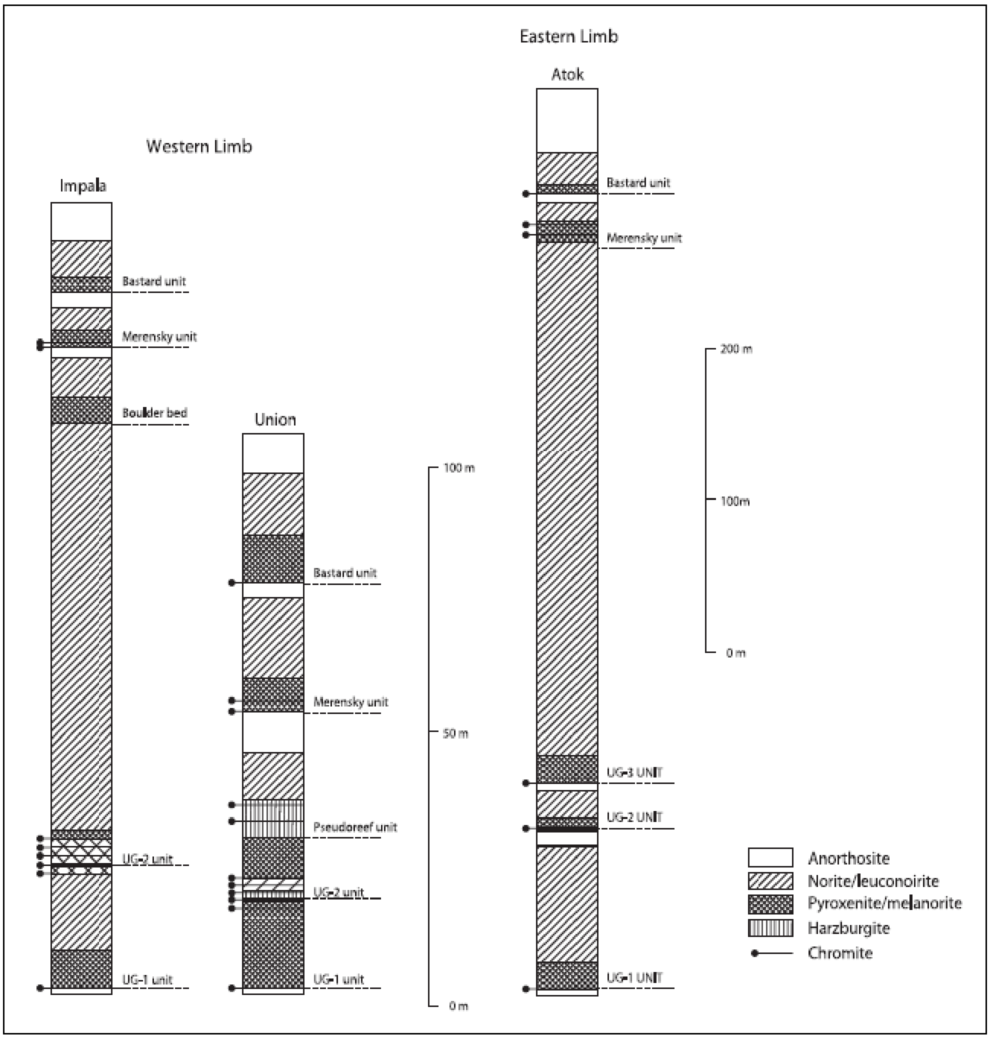

Figure 21. Stratigraphic differences between the Eastern Limb and Western Limb of the Bushveld Complex (after Barnes and Maier, 2002).

For example, at Union Mine situated in the north of the Western Limb of the Bushveld Complex, the entire sequence from GPA to UG1, occupies only 100 m in the stratigraphy (Barnes and Maier, 2002). Figure 21 shows the comparison between the Eastern Limb and Western Limb stratigraphy.

The top of the Critical Zone is marked by a poikilitic pyroxene anorthosite (GPA) and then grades into a 10 m wide norite. The norite grades into a 5 m wide poikilitic plagioclase pyroxenite, the Bastard pyroxene. The bottom of this unit is marked with a thin chrome stringer, creating a sharp contact with the anorthosite footwall. The anorthosite is about 2 – 3 m wide and grades into a norite and then into the second poikilitic plagioclase pyroxenite unit. The Merensky pyroxenite is up to 5 m wide.

The MR is characterized by a top (0.1 cm wide) chromitite stringer, 60 cm of pegmatoidal plagioclase pyroxenite and harzburgite and bound at the bottom by a second thin chromitite stinger (Smith et al., 2003; Kruger and Marsh, 1985; Lee and Butcher, 1990). The immediate footwall of the MR is a 5 m wide poikilitic pyroxene anorthosite that grades into another 15 m wide norite sequence. The schematic diagrams in figure 22 show the MR in the North Eastern Limb (a) compared to the MR in the Western Limb (b).

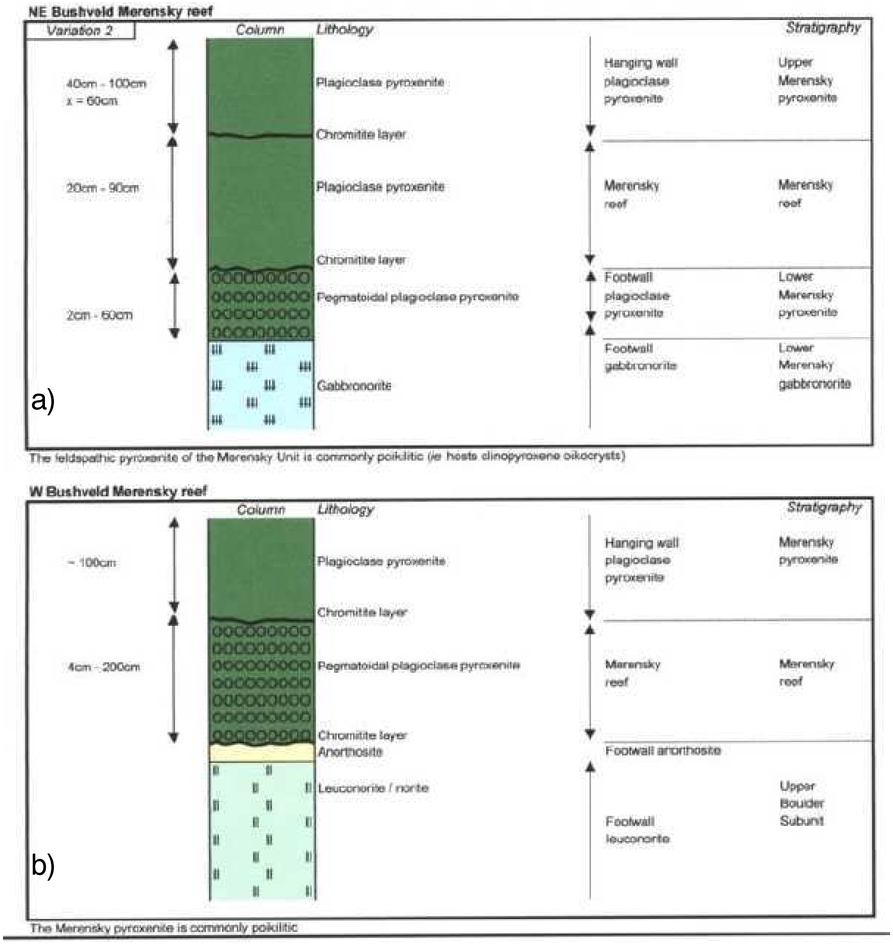

Figure 22. Difference between the MR in the eastern limb (a) to the western limb (b) (Brown, 2005).

This norite sequence grades into an 8 m wide feldspathic harzburgite unit, locally referred to as the Pseudoreef (Smith et al., 2003). There are some thin chromitite stringers present, as well as minor PGM mineralization, but are not currently being mined. It then grades into a 10 m wide poikilitic plagioclase pyroxenite, the hanging wall of the UG2. The UG2 unit consists of two to three 10 – 15 cm wide chromitite bands with alternating pyroxenite bands and then the main 60 - 80 cm wide chromitite, the total width of the UG2 reef package is usually 1.2 - 1.5 m. The footwall of the UG2 is usually a 70 cm wide pegmatoidal plagioclase pyroxenite and it then grades into a poikilitic plagioclase pyroxenite. The UG2 schematic is shown in figure 23, the main difference between the North Eastern and Western Limbs is the width of the poikilitic plagioclase pyroxenite between the chromitite leaders above the UG2 reef main band. The North Eastern Limb has the UG3 and no leaders above the UG2 main band as with the Western Limb.

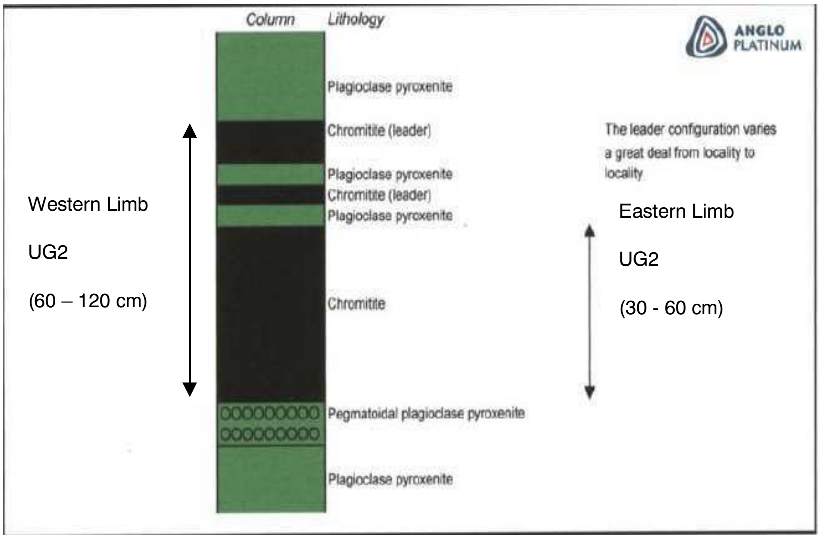

Figure 23. Difference between the UG2 in the Eastern Limb to the Western Limb (modified from Brown, 2005).

The footwall poikilitic plagioclase pyroxenite is about 15 to 20 m wide and is also the hanging wall of the UG1 chromitite. The UG1 is a 1.5 m wide chromitite band that show similar bifurcation (Eastern and Western Limbs) of the chromitite into the 3 m wide anorthosite footwall unit (Cawthorn et al., 2002).

5.2. STRUCTURE

Potholes are thermochemical erosional structural features between two layers. Formed while the cumulus pile is in a semi-solid state, and mainly caused by defluidisation and degassing of magmas, as well as, convection and movement of fluids in magmas (Carr et al., 1994). They can be described as circular or elliptical areas in which a portion of the footwall succession of the reef is absent, so that the reef and its hangingwall layers transgress into this area with an inward or centripetal dip. Variations in diameter, amplitude, and reef dip are common and the grade profiles of the reef are often affected (Brown and Lee, 1987; Cawthorn et al., 2002).

Merensky reef potholes, including those at Lebowa Platinum Mines (now Bokoni), have been well documented by, for example: Buntin et al. (1985); Campbell (1986); Fleming and Lee (1986); Mossom (1986); Brown and Lee (1987); Boudreau (1992); and Carr et al. (1994).

Some of the findings by Brown and Lee (1987) in their study of the nature and characteristics of MR potholes at Atok Platinum Mine (also known as Lebowa or Bokoni), are summarized as follows:

a) Potholes are generally ovoid in shape and have long axis dimensions of between 5 m and 145 m, with the average diameter being 23 m (skewed distribution, with the number of small diameter potholes greatly exceeding those of large diameter). Pothole depths vary from just a few metres below normal MR elevation, to in excess of 30 m below.

b) Pothole morphology is commonly of a ‘soup-bowl’ profile, with no particular layer acting as an ‘arresting layer’, as is commonly seen in parts of the Western Bushveld Complex. The stratigraphy is commonly severely disrupted on entry to, and within potholes. The MR within potholes is commonly abnormally thickened (and therefore of uneconomical grade), commonly disrupted, and of continuously variable elevation.

c) Statistical analysis of the distribution of mapped potholes showed them to be of random distribution.

d) No association with ultramafic or mafic replacement pegmatites was found. This observation explains the lack of detection by the magnetic surveys. Isolated and small felsic pegmatite occurrences, were however found spatially associated with potholes in a few cases, and some with late-stage anorthosite lenses in them. These features are not Fe-rich and would thus not have been detected by magnetometers.

e) No evidence of any relationship between pothole occurrence and structural phenomena such as joint density or orientation, faulting, or ground conditions was found.

UG2 potholes also show characteristic features and there are some markers that could indicate that there is a pothole approaching (for mining) or some markers to consider when only borehole core information is available. Some of these are summarised by Langwieder (2004) below:

- The increase or decrease in distance/middling between the MR and the UG2;
- The increase or decrease in distance/middling between UG3 and the UG2;
- Decreased distance/middling between UG2 and UG1;
- No reef intersection in borehole, this means UG2 is missing but other chromitite layers (UG1 and/or UG3) or 'markers' were identified;
- Thickened or thinned out UG2 chromitite (from general 61 cm width);
- Lack of chromitite stringers in the immediate hangingwall pyroxenite of the UG2;
- Duplication of UG2 intersections with no signs of faulting in between;
- Steeper than normal dip (top contact) of the UG2 chromitite layer;
- Replacement pegmatite occurrence within close proximity to the UG2;
- In underground excavations a change (usually steepening) in the dip of the reef as well as thinning of the UG2 chromitite (from average width) is indicative of a potential pothole or slump feature approaching;
- An abrupt end to the UG2 chromitite has also been encountered – with no signs of faulting present.

Undulations (often wrongly referred to as "rolling reef") are frequently observed on both reef horizons. Undulations represent small scale variations on one of the contacts (hangingwall or footwall) in relation to the general strike and dip of the ore body. Undulating reef is generally minable, as opposed to potholes, which are generally considered un-mineable or as geological loss.

Rolling reef refers to a reef horizon which, as an entity, deviates from its planar nature, and noticeably shows small scale changes in dip and strike. The dimensions, specifically the amplitudes, are such that extraction of rolling reef is possible.

All of the above can be classified as slump features, but depending on the scale of the feature, different terminology is often applied (e.g. regional slump or mega-pothole).

The TPM Project seems to be remarkably uncomplicated with regards to faulting. Minor faulting is expected to occur, consisting of dextral and sinistral strike-slip faults, normal and reverse dip-slip faults, as well as faults of both components. Displacements have generally been, and are expected to be, small at approximately 1 m. The only substantial fault that has been identified is a 20 m down throw fault to the north-west, in the middle of the TPM Project area.

Thrust faulting is synonymous to adverse ground conditions for any underground operation, although the actual displacements are usually very small. These features exist on most project areas in the Eastern Bushveld, but the occurrence and frequency are still under investigation (Brown, 2005).

An example of a known thrust fault occurs within the UG2 footwall stratigraphy. The so called 'Footwall 3 Shear' is found between 7 and 15 m below the UG2 and occurs in the poikilitic anorthosite ('Footwall 3'). This poikilitic anorthosite (also called the 'Footwall Marker') often contains chromitite stringers and these are associated with the thrust fault. It is regarded as a layer parallel to sub parallel to the undulating low-angle thrust fault (10 - 30 cm wide affected zone) and sometimes transgressing into the feldspathic pyroxenite ('Footwall 2'). Shearing, brecciation and even mylonitisation as well as, often intense alteration are commonly observed in the proximity and within the affected zone.

The 'Footwall 3 Shear' commonly occurs between 7 and 15 m below the UG2 and is an important mine design consideration for the layout of the haulages and cross cuts. This specific fault zone does not affect the MR horizon.

The major joint directions measured from strong macro-lineament features evident from the LandSat TM Spacemap, and underground excavations, show the dominant strike directions to be NNE –SSW and steeply dipping (75° – 90°).

A series of NNE-SSW striking dykes of post- Karoo age have been identified by the aeromagnetic survey. The dykes are primarily of dolerite composition and tend to be fine-grained. Areas of dense jointing and alteration are associated with the dyke edges, and the ground conditions underground tend to be very blocky.

Several of these dykes have also been mapped in outcrop and were exposed by means of trenching. Dips vary between 62 and 89 degrees to the SE, however some do dip to the NW.

Field mapping has indicated dyke widths as little as a few centimetres up to above 30 m, occurring either as a single entity or as dyke swarms, often showing en echelon type displacement. Sills could also be present, but none has been confirmed on the project area.

Replacement pegmatites are cross cutting, irregularly shaped rocks, which can be ultramafic, mafic, intermediate or felsic (Brown, 2004). Felsic pegmatites generally occur in the project area as white, sub-vertically orientated veins, which seldom exceed 10 cm in width. The occasional irregularly shaped pegmatite mass also does occur. These pegmatites seldom exceed 2 m across. These veins also typically show myrmekitic textures.

Felsic pegmatites naturally follow one of the major joint/fracture directions but most commonly trend NNE-SSW. These pegmatite occurrences are minor in abundance and extent when compared to the better-known and often very destructive mafic and ultramafic pegmatites that occur elsewhere on the Eastern Limb (Brown, 2005). They are thought to represent late-stage Bushveld features which have formed by metasomatic replacement of the original rock, hence forming discrete bodies enclosed by the “host cumulates”, and disturbing the original stratigraphy (Langwieder, 2004). In some cases the reef may be intensely disturbed or even absent, and therefore is called replaced. Iron-rich replacement pegmatites are fairly easy to identify by means of aeromagnetic surveys.

In instances where iron rich ultramafic replacement pegmatite affects the MR, its associated alteration does not replace the PGE, but merely influences the metallurgical recovery processes of the minerals from the reef zone negatively due to the change in PGM mineralogy. Recoveries are typically reduced to approximately 50 – 60 % (Roberts and Malysiak, 2004). Other effects related to replacement pegmatites can be strike and dip changes, slumping, as well as partial or total replacement of the Merensky pyroxenite unit, including the reef. The MR tends to be more susceptible to replacement than the UG2.

5.3. GEOLOGICAL LOSSES

The geological loss refers to a percentage of the total resource area lost to mining and reef extraction due to geological features. These features are an important factor for mine planning and resource estimation. A forecast, where these features occur and how frequently they occur, is virtually impossible. By extrapolation from exploration results, the most important loss features, in terms of their impact on the reefs, can be identified. Identifying these features helps to increase the confidence in mine planning, scheduling, and project evaluation (Langwieder, 2010).

Estimated geological losses around dykes are based on the width, jointing and alteration in the immediate vicinity, the magnetic response, as well as the landscaping effect of the feature concerned. An average zone of influence can be applied unless detailed information on geometry and ground conditions is available from trenching or outcrop mapping and is subject to continuous revision.

Potholes are the other major source of geological loss. As pothole dimensions are impossible to determine before it is actually mined, it is usually estimated from actual mining history as well as exploration activities. An estimate of the percentage potential potholes can be made by looking at the amount of pothole intersections compared to the total amount of reef intersections obtained from borehole core drilling. This can also be benchmarked with surrounding mines and or other exploration activities.

Geological losses are updated annually as part of the strategic planning cycle, but if a major structure is discovered the structural model and geological losses will be updated to ensure that the latest and most correct information is used for decision making.

5.4. RESOURCES AND RESERVES

Mineral resources and mineral reserves must be classified, verified, and reported in accordance with stock exchange, industry and professional guidelines. The classifications are based on the South African Code for the reporting of exploration results (SAMREC, 2007).

Reporting is done by professionals with appropriate experience in the estimation, economic evaluation, exploitation, and reporting of mineral resources and mineral reserves relevant to the various styles of mineralization under consideration.

Where mineral resources and mineral reserves are quoted, resources are usually in addition to reserves. Resources are, by definition, exclusive of any diluting materials that might arise as a consequence of the mining method and specific geological circumstances applicable to the mining of that resource. Mineral resources do however exclude the appropriate known and unknown geological losses. Mineral reserves on the other hand, include all expected mining related dilution.

Mineral resources are reported over a minimum practical mining width (SAMREC, clause 21). Because the widths of the MR and the UG2 reefs are generally less than 70 cm, the resource cut for the MR and UG2 is quoted over a practical minimum mining cut, suitable for the deposit and is referred to as the resource cut. The reported resources are based on a minimum width of 90 cm as investigations have confirmed that this is practical and safe. The resource cut includes geotechnical aspects in the hangingwall or footwall of the reef. Chromitite stringers above or below the UG2 or any ‘geotechnical weak zones’ are included in the resource cut. The minimum beam height regarding the geotechnical aspect depends on the mining method. The conversion of the resource cut to an appropriate reserve width would include additional dilution incurred as the result of mining considerations.

The SAMREC code (2007) defines mineral resources and reserves as follow:
Mineral Resources: “*A mineral resource is a concentration or occurrence of material of economic interest in or on the earth’s crust, in such form and quantity that there are reasonable and realistic prospects for eventual economic extraction. The location, quantity, grade, continuity and other geological characteristics of a mineral resource are known or estimated from specific geological evidence, sampling and knowledge interpreted from an appropriately constrained and portrayed geological*

model. Mineral resources are subdivided in order of increasing confidence in respect of geoscientific evidence into 'Inferred', 'Indicated' and 'Measured' categories, and must be so reported." (SAMREC, 2007).

Ore Reserves: "*An ore reserve is the economically mineable material derived from a measured and/or an indicated mineral resource. It includes diluting materials and allows for losses that are expected to occur when the material is mined. Appropriate assessments to a minimum of a pre-feasibility study for a project, or of a life-of-mine plan for an operation, must have been carried out, including consideration of, and modification by, realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors (the modifying factors)*." (SAMREC, 2007).

These assessments demonstrate, that extraction is justifiable, at the time of reporting. Ore reserves are subdivided in order of increasing confidence into probable ore reserves and proved ore reserves (SAMREC, 2007).

The resource modelling for the TPM Project is updated annually during the strategic planning cycle. Additional grade information from underground sampling and diamond drilling is compiled and quality control checks are performed. The data is passed and incorporated into the model with the updated structural model and geological losses. The updated resource classification (measured, indicated and inferred) information (tons, ounces and grade) is then available for mine planning and scheduling.

The applicable modifying factors (mining and pillar losses; additional dilution; re-development; mine call factor; and design limitations) are re-evaluated annually and applied to the scheduling of the extraction plan. This plan is then used to determine the reserve (tons, ounces and grade) that is going to be extracted and reported to the shareholders.

6. THE MERENSKY REEF (MR)

The Merensky Reef is mainly composed of poikilitic plagioclase orthopyroxenite and varies in thickness from more than 0.5 m but not more than 2 m. Texturally the MR varies from medium crystalline to patchy coarse to very coarse crystalline with a medium crystalline matrix. Visible sulphides occur in variable amounts within the reef horizon. The MR, under 'normal' reef conditions, is demarcated by a top chromitite stringer (generally less than 1 cm thick) and often by a bottom chromitite stringer (also less than 1 cm thick) and is underlain by a poikilitic plagioclase orthopyroxenite up to 10 m wide. The top 0.1 to 1.5 m of this unit is usually pegmatoidal (coarse to very coarse crystalline) which often is also mineralised to a variable degree.

The medium crystalline poikilitic plagioclase orthopyroxenite continues above the top chromitite stringer for between 40 and 70 cm before grading into a norite. This norite grades into a poikilitic pyroxene anorthosite. A second pyroxenite unit occurs above this anorthosite, the Bastard pyroxenite, which is sometimes confused with the MR (Cameron, 1982).

The occurrence of chromitite stringers (but specifically the upper chromitite-stringer) is often associated with the "value zones" of the MR and is variable, and in some cases the chromitite stringers may be completely absent. PGE mineralization often continues into the hangingwall and footwall units of the MR (Cawthorn and Boerst, 2006; Mathez et al., 1997).

The following lithological 'facies' variations has been identified in the project area, based on the occurrence of chromitite-stringers forming the top and bottom contacts of the MR, as well as the pegmatoidal plagioclase pyroxenite (FW1). The most common lithological 'facies' is the presence of two chromitite-stringers (top and bottom) with FW1 (pegmatoidal plagioclase pyroxenite), which represents approximately 39 % of the boreholes investigated. FW1 seems to be the main footwall lithology of the MR as only 29.1 % of intersections had no FW1 development. Langwieder (2004) investigated 212 boreholes across the project area with regards to the described lithological 'facies' variations. Table 2 refers to the individual percentages of each type identified, and figure 24 shows the four major lithological 'facies' based on chromitite stringer occurrences, irrespective of the association with the FW pegmatoidal plagioclase pyroxenite for the project area spatial relationships.

Table 2. All possible eight lithological 'facies' variations and its percentages with regards to occurrence in boreholes (Source: Langwieder, 2004).

Facies	Detail/Description	No. of boreholes	Percentage
2CP	upper and bottom chromitite stringer plus FW1 pegmatoid developed	83	39.2%
UCP	upper chromitite stringer only plus FW1 pegmatoid developed	31	14.6%
BCP	bottom chromitite stringer only plus FW1 pegmatoid developed	9	4.2%
NCP	no chromitite stringer but FW1 pegmatoid developed	23	10.9%
2C	upper and bottom chromitite stringer, but no FW1 pegmatoid developed	17	6.0%
UC	upper chromitite stringer only, but no FW1 pegmatoid developed	14	6.6%
BC	bottom chromitite stringer only, but no FW1 pegmatoid developed	6	2.8%
NC	neither chromitite stringer, nor FW1 pegmatoid developed	29	13.7%
total		***212***	100%

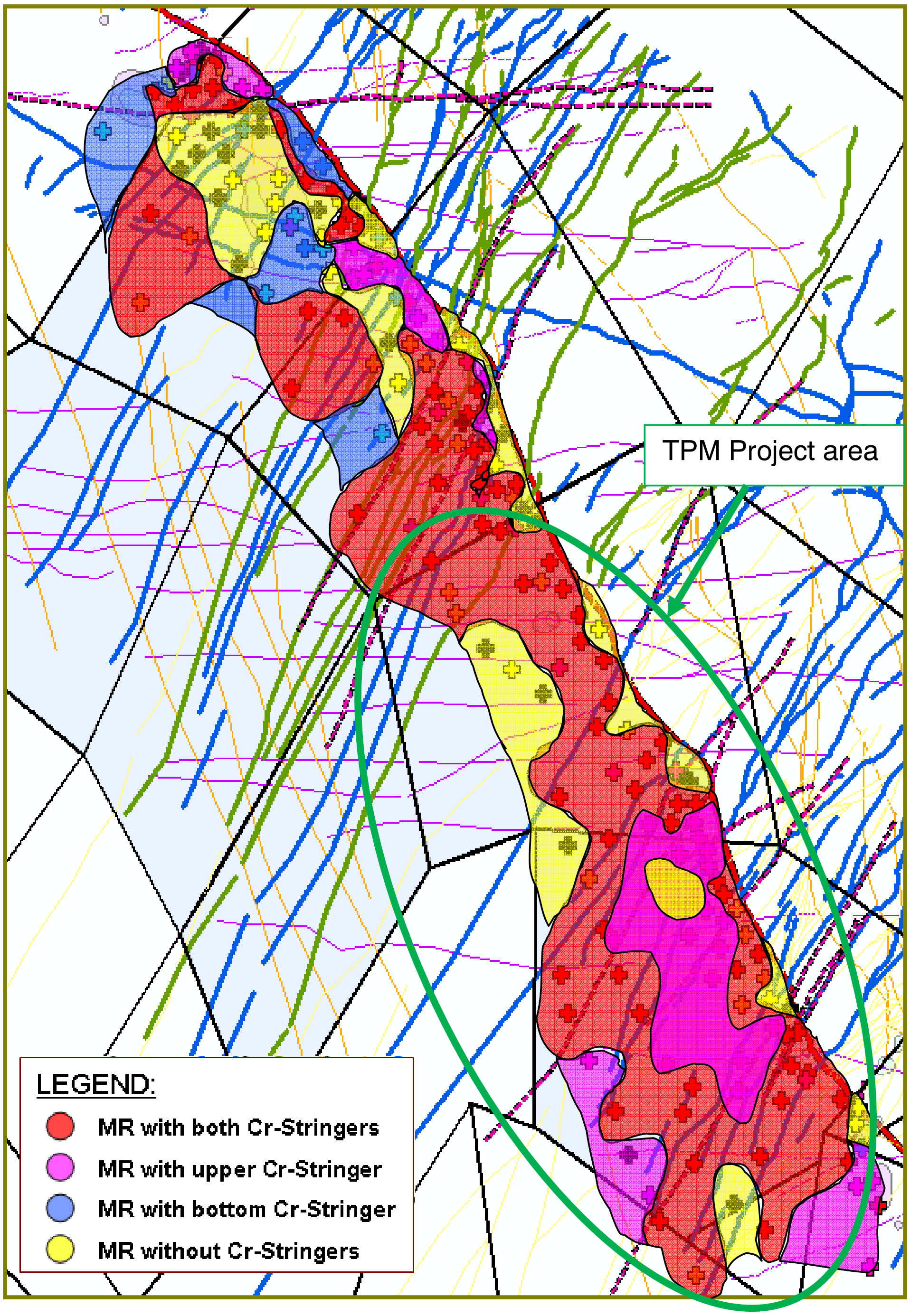

Figure 24. The four major lithological facies based on Cr-stringer occurrences, irrespective of the association with the FW pegmatoidal plagioclase pyroxenite for the project area spatial relationships (after Langwieder, 2004).

Chromite, base and precious metal sulphide accumulations are hosted within the MR plagioclase pyroxenite, with some minor occurrence in the immediate hangingwall and in the footwall rocks. Some relatively high PGE grades are known to occur down to 1 – 3 m below the MR basal contact, however these occurrences are highly erratic. The base metal sulphides (BMS) occur as discreet particles, sharing interstitial space with plagioclase feldspar, within a silicate framework of orthopyroxene. The main BMS are chalcopyrite, pentlandite, pyrrhotite and pyrite. PGE mineralization occurs as discrete platinum-group minerals (PGM) that is typically in close association with, and enclosed within, the BMS, and silicates. The PGM usually comprise PGE sulphides, sulpharsenides, arsenides, bismuthides, tellurides, bismuthotellurides and alloys. PGE and PGM are also strongly associated with the chromitite layers, as a function of BMS occurrence associated with these layers and PGE / PGM collection and enclosure by chromite (Brown, 2003; Naldrett et al., 2009).

The MR shows a typical bimodal PGE grade distribution (figure 25 and 26) and is largely related to the occurrence of the two chromitite layers that mark the top and bottom contact of the MR (Mathez et al., 1997; Brown and Lee, 1987; Barnes and Maier, 2002).

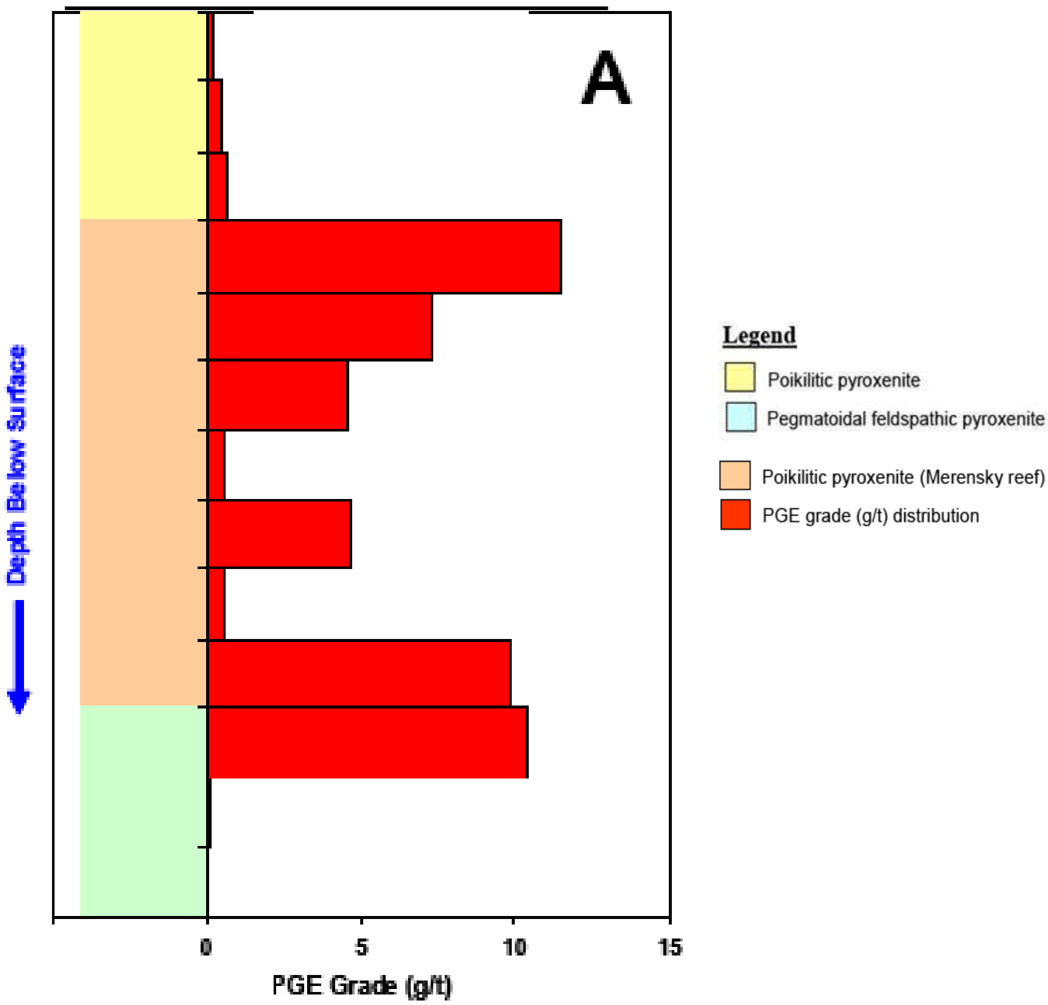

Figure 25. A schematic diagram showing the typical grade distribution of the MR (Brown, 2005).

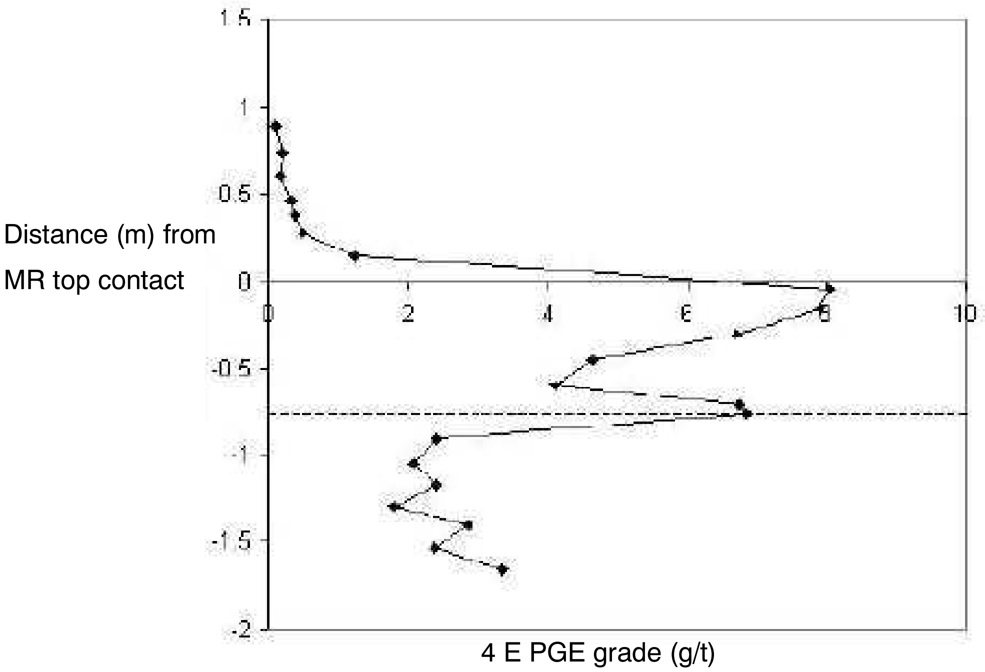

Figure 26. The average 4E PGE grade profile through the MR. Data from 90 MR intersections in the Lebowa (Bokoni) area. The solid horizontal line represents the MR top contact and the dashed line the bottom contact. Both contacts are defined by chromitite stringers (Brown, 2005).

The rock density for each unit has been calculated and the average for the MR is 3.4 g/cm^3. This comprises of the hanging wall pyroxenite (10 cm) at 3.32 g/cm^3; the approximate 70 cm of MR at 3.49 g/cm^3 (slightly higher due to the presence of chromitite) and the remaining 20 cm footwall pyroxenite at 3.4 g/cm^3. This will comprise the optimal mining cut of 100 cm.

The 100 cm takes into account the various lithological differences, as well as the value distribution within the MR. The hangingwall is mainly barren with regards to PGE mineralisation and with limited to no identified chromitite stringers that will influence the mining cut, 10 cm is considered adequate to ensure the value on the top contact chromitite stinger is removed during mining. The 20 cm footwall allowance is due to the sporadic grade occurrences below the bottom chromitite contact.

Mineralogical and metallurgical studies conducted on MR borehole intersections found that the MR varies substantially in width, and is top and bottom loaded in and around the chromitite layers. The hanging wall adjacent to the reef is generally barren of mineralisation, but PGE grades of >5 g/t are occasionally developed in the immediate pegmatoidal footwall, and are accompanied by coarse sulphides.

The samples have a 4E PGE grade of 3.2 to 9.5 g/t and a Pt:Pd ratio of 1.5 to 2.0. BMS comprise 0.3 to 2.7 %, with pentlandite and lesser chalcopyrite being the dominant liberated species.

PGE-sulphides are the major free PGM-type (figure 27) and PGE-tellurides the most locked. This implies that PGE that are situated within PGE-sulphide bonds are 'easier' to liberate and separate than PGE as PGE-tellurides. Figure 27 is an example of the percentage of each PGM-type that was identified in four borehole intersections that were analysed.

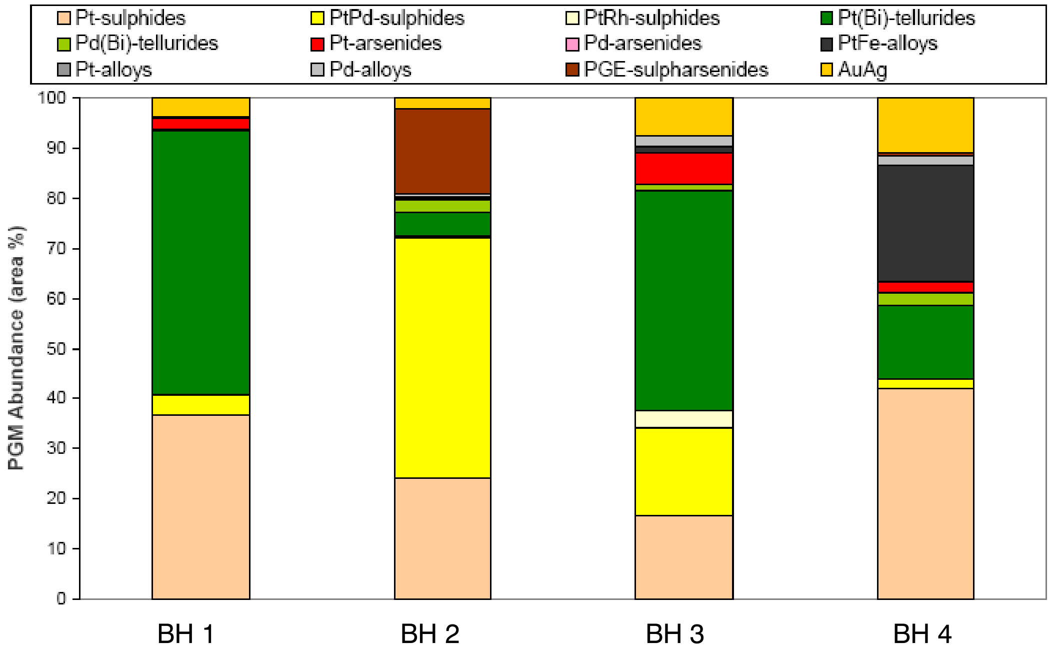

Figure 27. The modal abundance of the PGM-types (area %) as determined by the SEM-BPS for four boreholes analysed in the TPM Project area (Roberts and Shamaila, 2005).

Final recoveries of >92 % Pt and >95 % Pd are obtained for seven samples due to optimum liberation of the value minerals (figure 28) and the unaltered nature of the reef. Samples that are more altered, with oxidised sulphides and lower 4E PGE grades, produced lower recoveries (Roberts and Shamaila, 2005).

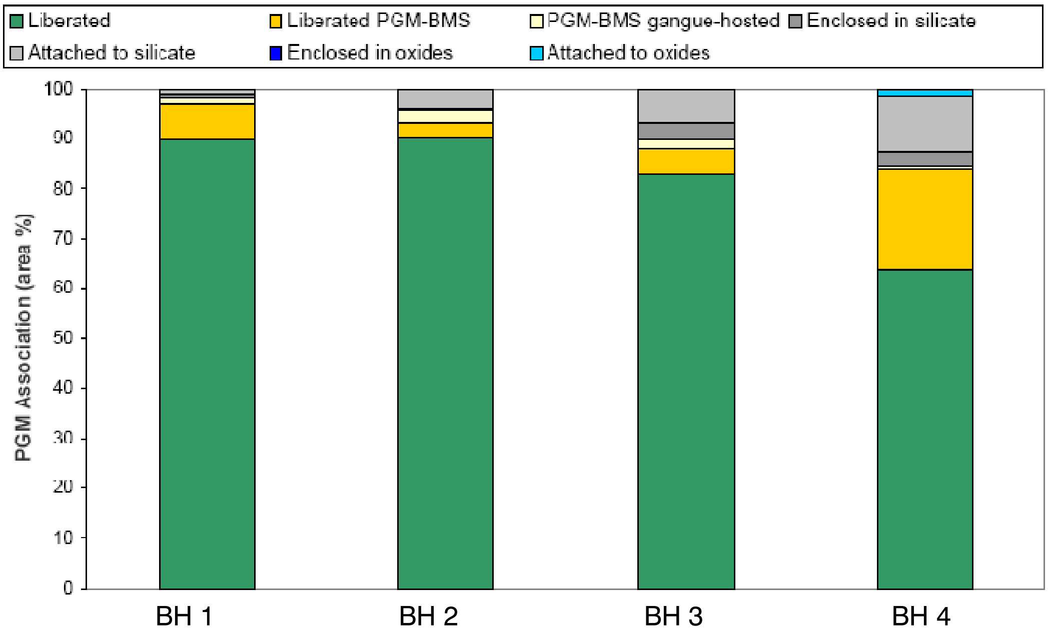

Figure 28. Liberation and association characteristics of the PGM (area %) as determined by the SEM-BPS for four boreholes analysed in the TPM Project area (Roberts and Shamaila, 2005).

The average 4E resource grade expected across a 100 cm stoping width is 5.02 g/t (Anglo Platinum Annual Report, 2010). This has been calculated in the resource estimation and takes into account the grade distribution in the MR to determine the best cut or value zone; the size and dip of the ore body; the various lithological types identified; and the density of each lithology.

The MR dimensions on the mining right area are calculated from the outcrop position along the mining right area boundary to give the area in metres squared (m^2). The average dip of the reef (16°) is then used to determine the dip corrected area. A dip corrected area of 25 Mm^2 will be used (calculated from plan) for the MR resource calculation.

The total geological loss for the TPM project MR is estimated at 25%. Of this, potholes are likely to account for 16 %, dykes 5 % and the remaining 4 % made up of losses from faults and alteration disturbances like replacement pegmatites.

The tonnage is calculated by multiplying the dip area (Mm^2) with the density (g/cm^3). The tonnage after geological loss is determined by subtracting the percentage of geological loss from the original tonnage.

The resource for the MR is shown in table 3. The prill split is the percentage of each metal (Pt, Pd, Rh, Au) that is present in the MR. The four percentages add up to 100 percent and it represents the composition of the 4E (3PGE + Au). Each of these elements also has a grade associated with it.

Table 3. TPM Project MR resource tabulation (calculated from parameters in Anglo Platinum Annual Report, 2009).

Twickenham Platinum Mine Project											
Merensky Reef	Dip area	Resource cut	Density	Geological	Tonnage after Geo Loss	4E grade	Content	Prill Split			
	(Mm2)	(m)	(g/cm^3)	Loss (%)	(Mtons)	(g/t)	(4E moz)	Pt%	Pd%	Rh%	Au%
Total	25	1	3.4	25	63.75	5.02	10.3	58	31.3	2.7	8

The 4E content in ounces (oz, where 1 gram is equal to 31.10348 troy ounces) is calculated by multiplying the tonnage (after geological loss) with the 4E grade (g/t) and converting the g/t to ounces. This represents the metal content that should be in the ore, and that is potentially available in the ore body (assuming 100 % extraction and 100 % recovery) for sale.

The proposed method of extraction for the MR will be based on the current UG2 mine design. The mine design entails two decline clusters, each comprising a material, conveyor and chairlift decline. The decline system will have an underground ore production capacity of 105 ktpm. The mining method will be conventional breast stoping, with tracked footwall haulages serviced by trackless footwall declines. Surface and underground engineering infrastructure will be required to facilitate production from seven operating half levels per decline system. Ore will be moved via conveyor belts to the on-site concentrator facility (De la Vergne, 2003).

7. THE UPPER GROUP 2 (UG2) CHROMITITE

The distance between the UG2 and UG1 at the TPM project is approximately 89 – 100 m. Above the UG1, a pyroxenite-norite sequence separates it from the UG2 unit. The 'Footwall 3 Shear' is found 7 to 15 m below the UG2 and occurs in the poikilitic anorthosite ('Footwall 3'). This poikilitic anorthosite (also called the 'Footwall Marker') often contains chromitite stringers that is associated with thrust faulting. The 'Footwall Marker' is overlain by a poikilitic feldspathic pyroxenite that is immediately overlain by a pegmatoidal pyroxenite. This pegmatoidal pyroxenite is highly variable in thickness and textural character, and it forms the immediate footwall of the main UG2 chromitite layer.

The main UG2 chromitite averages 61 cm in thickness at the TPM Project and is seldom thicker, but may pinch out to 10cm in disturbed or potholed areas. Occasionally pyroxenite lenses of limited lateral extent occur within the UG2 and are generally referred to as 'internal waste'. The UG2 is a chromite cumulate, and occurs either as a pure chromite or as a dense cumulate framework of chromite with fine crystalline interstitial plagioclase or othropyroxene. Interstitial silicates are seldom visible but interstitial sulphides can sometimes be seen with the naked eye (Brown, 2005; Cawthorn et al., 2006; Viljoen and Schürmann, 1998).

The top contact of the UG2 is sharp and the bottom contact may be sharp but generally tends to be diffuse or gradational in nature. Above the main UG2 layer, there is a poikilitic pyroxenite sequence of 3 to 5 m in thickness. The pyroxenite unit may contain up to 6 thin chromitite stringers between 0.1 and 2 cm in thickness. The separation between the main UG2 chromitite and the lowest leader is variable, but can be as little as 10 cm. The last chromitite stringer occurrence can be as far as 2 m above the UG2 top contact within the immediate hangingwall pyroxenite. These stringers in the hangingwall of the UG2 will play an important role in terms of hangingwall support and grade control (Mabuza, 2006). Anorthositic markers occasionally occur within the pyroxenitic hangingwall. The pyroxenite unit grades upwards into a gabbronorite-norite-anorthosite unit.

The UG2 – UG3 separation is approximately 26 m, with the UG3A situated approximately 11 m above the hanging wall contact of the UG3.

The mineralization within the UG2 occurs throughout the main chromitite band and shows a bimodal grade distribution, associated with the top and the bottom of the Chromitite (figures 29 and 30). Minor mineralization is often found in the footwall of the UG2, in the form of disseminated chromite and lenses within the pegmatoidal plagioclase pyroxenite. The first leader seam within the hangingwall pyroxenite contains limited PGM mineralization. The hangingwall pyroxenite is however considered barren of any PGE mineralisation. PGE mineralisation occurs as solid solution PGE in BMS, either associated the BMS or mainly as discrete PGM. The UG2 is typically less BMS-enriched than the MR. The BMS and PGM are also partially or wholly enclosed within the individual chromite crystals (Brown, 2005; Barnes and Maier, 2002).

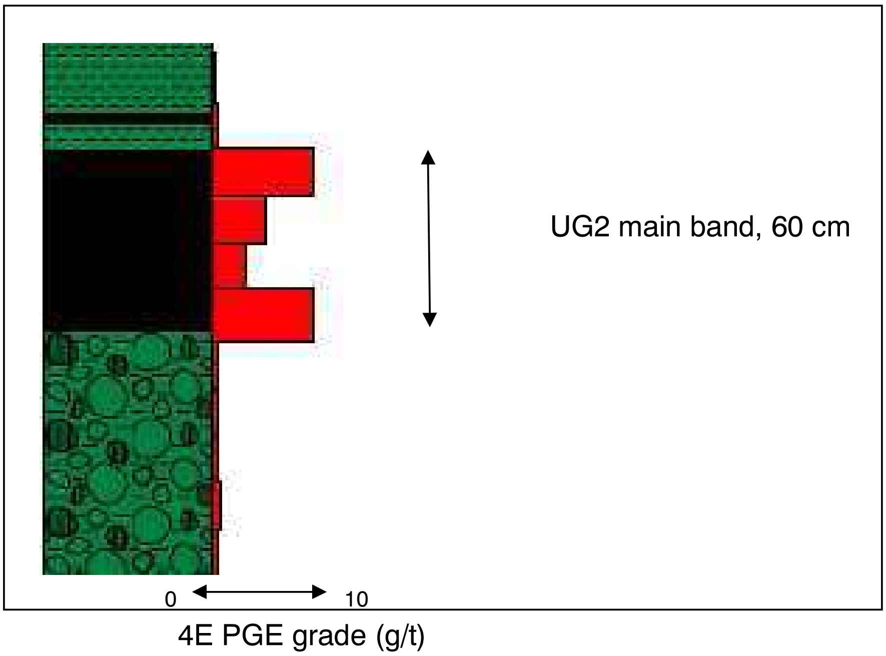

Figure 29. A schematic diagram showing the typical 4E PGE grade distribution of the UG2. Not to scale. The first hangingwall chromitite stringer has minor grade, the bimodal grade distribution within with main reef band; and the pegmatoidal pyroxenite footwall contains limited mineralisation.

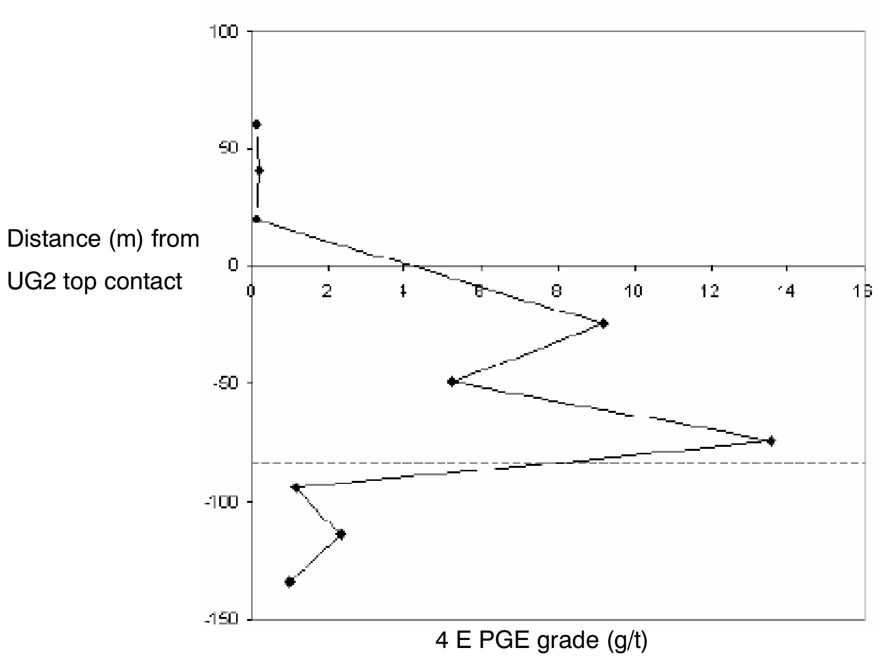

Figure 30. The average 4E PGE grade profile through the UG2. Data from UG2 intersections in the North Eastern Bushveld. The solid horizontal line represents the UG2 top contact and the dashed line the bottom contact. (Brown, 2005)

The average densities have been calculated for the UG2 succession comprising of 15 cm hangingwall pyroxenite 3.44 g/cm^3, 61 cm wide UG2 reef 4.17 g/cm^3, and approximately 17 cm footwall pyroxenite 3.42 g/cm^3. The average density for the stoping width is 3.98 g/cm^3. The 93 cm wide stoping width is based on the hangingwall geotechnical considerations (the thin leader chromite stingers); the reef main chromitite; and the rest made up of the pegmatoidal pyroxenite footwall.

Minerallogical and metallurgical studies conducted on the UG2 found that the base metal sulphides and PGM are very fine-grained. Sulphides are generally less than 100µm, while the PGM are less than 20µm in size. Despite being fine-grained the base metal sulphides are coarser in comparison to those of the western Bushveld UG2 (Roberts, August 2011, Personal communication).

The PGM consists mostly of cooperite (PtS), braggite (PtPd(Ni)S), and laurite (Ru(Os,Ir)S_2), PGE-tellurides and alloys, including gold and electrum constitute the balance of the PGM present (Roberts and Malysaik, 2004). Figure 31 is an example of the percentage of each PGM-type that was identified in six borehole intersections that was analysed. They have been grouped by reef characteristics and also give an indication of the main PGM present for each type, for example 'altered' and 'normal', based on lithology logs and the PGM-types present.

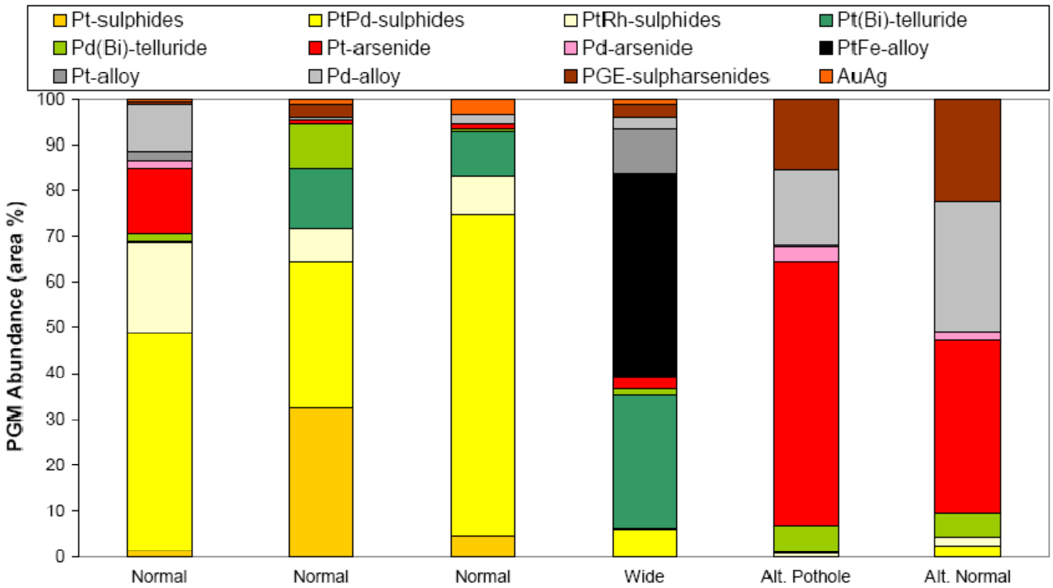

Figure 31. The modal abundance of the PGM-types (area %) as determined by the SEM-BPS for six boreholes analysed in the TPM Project area (Roberts and Malysaik, 2004).

The UG2 reef intersections that were analysed have a 4E PGE grade of 5.1-7.2 g/t and a Pt/Pd ratio of 0.8-1.1. Chromite comprises 54-59 % of each mining cut (93 cm). Pentlandite and chalcopyrite are the principal sulphide species.

Base metal sulphides (BMS) occur sporadically in the pegmatoidal feldspathic pyroxenite footwall and are principally composed of pentlandite and chalcopyrite.

In normal and wide reef, the BMS have a top size of 360 µm and are mainly composed of pentlandite and chalcopyrite, with varying amounts of millerite present in some of the intersections. About 80 % of the sulphides occur along grain boundaries and pyroxene cleavage planes and will most likely be liberated during the early stages of milling (Roberts and Malysaik, 2004).

Pentlandite, chalcopyrite and PGE-sulphide are the most common value minerals in these ores. The PGM are preferentially associated with BMS. Consequently, they are readily amenable to flotation and final recoveries in excess of 90 % Pt and Pd are achieved. Figure 32 is an example of the liberation and associated PGM characteristics found in six boreholes that were analysed.

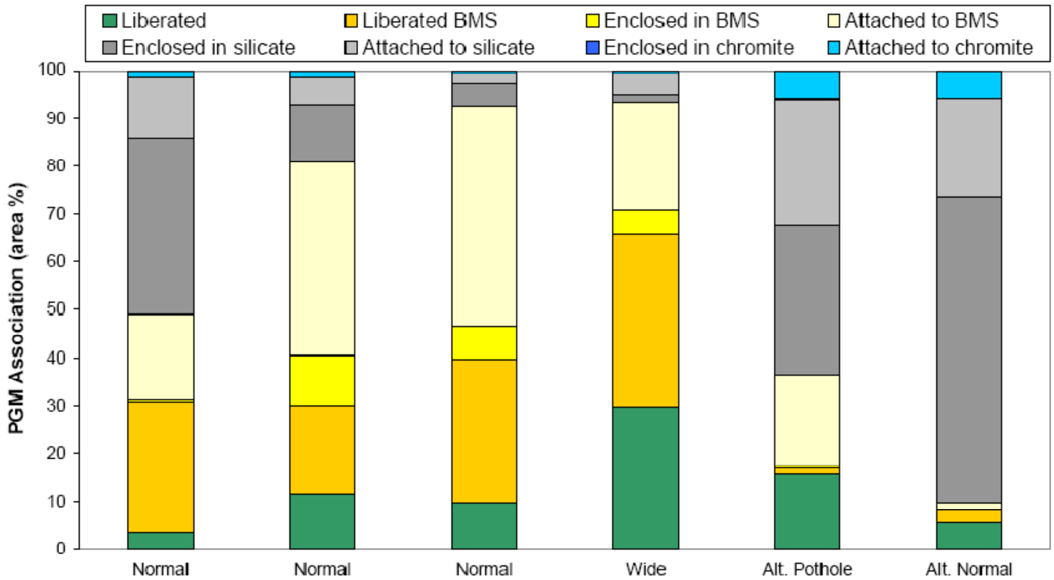

Figure 32. Liberation and association characteristics of the PGM (area %) as determined by the SEM-BPS for six boreholes analysed in the TPM Project area (Roberts and Malysaik, 2004).

Progressive alteration of normal and pothole reef affects the mineralogical characteristics of the BMS and PGM quite substantially and decreases final recovery quite considerably to 42 % Pt and 58 % Pd. The highly altered pothole and normal reef intersections contain much finer BMS. For normal reef, millerite is relatively common and half the PGM are hosted by silicate. This results in high liberation percentages and good recoveries. (Roberts and Malysaik, 2004).

The average 4E resource grade expected across the 93 cm stoping width is 6.34 g/t (AP 2010). This has been calculated in the resource estimation and takes into account the grade distribution within the UG2, the hangingwall geotechnical considerations, the minimum mining cut, the size and dip of the ore body, and the density of each lithology.

The UG2 ore body dimensions on the mining right area are calculated from the outcrop position along the mining right area boundary to give the area in metres squared (m^2). The average dip of the reef (16°) is then used to determine the dip corrected area. A dip corrected area of 27 Mm^2 will be used (calculated from plan) for the UG2 resource calculation.

The total geological loss for the TPM UG2 project area has been estimated at 23 %. Of this, 13 % is attributed to potholes, 7 % to dykes and 3 % to faulting and alteration disturbances like replacement pegmatites.

Tonnage is calculated by multiplying the dip area (Mm^2) with the density (g/cm^3) and the tonnage after geological loss is determined by subtracting the percentage of geological loss from the original tonnage.

The resource for the UG2 is shown in table 4. The prill split is the percentage of each metal (Pt, Pd, Rh, Au) that is present in the UG2. The four percentages add up to 100 percent and it represents the composition of the 4E (3PGE + Au). Each of these elements also has a grade associated with it.

Table 4. TPM Project UG2 resource tabulation (calculated from parameters in AP, 2009).

Twickenham Platinum Mine Project											
UG2 Reef	Dip area	Resource cut	Density	Geological	Tonnage after Geo Loss	4E grade	Content	Prill Split			
	(Mm2)	(m)	(g/cm^3)	Loss (%)	(Mtons)	(g/t)	(4E moz)	Pt%	Pd%	Rh%	Au%
Total	25	0.93	3.98	23	71.25	6.34	14.52	43.1	47.2	8.2	1.5

The 4E content in ounces (oz, where 31.10348 troy ounces is equal to1 gram) is calculated by multiplying the tonnage (after geological loss) with the 4E grade (g/t) and converting the g/t to ounces. This represents the metal content that should be in the ore and that is potentially available in the ore body (assuming 100 % extraction and 100 % recovery) for sale.

The proposed method of extraction for the UG2 is based to the current UG2 mine design and entails two decline clusters, each comprising a material, conveyor and chairlift decline. These decline system will have an underground ore production capacity of 250 ktpm. The mining method will be conventional breast stoping with tracked footwall haulages serviced by trackless footwall declines. Surface and underground engineering infrastructure will be required to facilitate production from seven operating half levels per decline system. Ore will be moved via conveyor belts to the on-site concentrator facility (De la Vergne, 2003).

8. FINANCIAL EVALUATION

The most common financial evaluation method used in industry is the discounted cash flow (DCF) model, where the net present value (NPV) and internal rate of return (IRR) are calculated and used to determine and compare potential value. This method of valuation is quite robust, but is largely dependent on the quality of the input data and the assumptions made. This method relies on numerous assumptions and forecasts, for example, exchange rates, metal prices, inflation, taxes, grade, and recoveries.

A number of specialized computer software packages have been designed to model the cash flows and determine the NPV and IRR. These programs allow for the input of different parameters, assumptions and forecasts, which is to be used in the analyses.

For this evaluation, Microsoft Excel was used to calculate the cash flow, NPV and IRR. A sensitivity analysis and risk assessment was conducted to investigate the effect changes in the input parameters will have on the NPV and IRR.

8.1. INPUT PARAMETERS

The following parameters were used in the construction of the DCF:

8.1.1. Metal Prices

The metals that are routinely assayed for are Pt, Pd, Rh, and Au (4PGE or 4E), as well as the associated base metals, Cu and Ni. For this evaluation, only the 4E metals will be considered. Chapter 2 showed that the metal prices display a cyclic nature and fluctuate, depending on supply and demand from the global economy. It is therefore very important for any company to continuously analyze these trends and come up with assumptions/fluctuations that they will then apply to any/all evaluations. This is important for consistency and comparability between projects or operations that are planned.

Table 5 shows the metal price assumptions that will be used for calculating the basket price of the MR and UG2 reefs for this evaluation. The metal price forecasts were modified from an article by Consensus Economics Inc. (EMCF, 2010) and values were compared to actual values as recorded by the website www.kitco.com.

Table 5. Metal price forecast in US$, only the 4E (Pt, Pd, Rh, Au) will be used. (Source: modified from EMCF, 2010).

Year	Metal prices (US$, real 2011)			
	Pt	Pd	Rh	Au
2011	1730	594	2383	1341
2012	1770	607	2390	1336
2013	1781	609	2285	1315
2014	1794	620	2017	1304
2015	1779	623	1886	1288
2016	1752	611	1500	1279
2017	1700	573	1500	1204

The metal price assumptions for 2017 was used for the remainder of the life of mine being evaluated. These forecasts are slightly lower than the July 2011 metal prices. These forecasts are considered to be representative of the average metal prices based on market trends over the past two years. The current outlook by economists also suggests that another economic slump is expected, and this could have an impact on the supply and demand for PGM. Spot prices recorded from the website www.kitco.com for 15 May 2010 and 17 May 2011 is shown in table 6 below, as an example of the changes that can occur within one year.

Table 6. Metal prices in US$ (source: www.kitco.com)

	Pt	Pd	Rh	Au
15-May-10	1741	543	3080	1233.6
17-May-11	1776	722	2200	1493.7

8.1.2. Exchange Rates

The expected exchange rate is important when the product will be sold in a different currency from where it is produced. The project under investigation is situated in South Africa, where the currency is Rand (ZAR), but the product is sold on the global market, where the accepted currency is United States Dollars (US$).

The fluctuation in the strength of a currency is dependent on various factors (discussed in section 2.2). Fluctuations in the currency can have a significant impact on the profitability of any project, and therefore the forecast of the expected exchange rate over a period of time is extremely important. Table 7 below gives the exchange rates that were used for this evaluation, in order to determine the basket price for the MR and UG2. These exchange rate forecasts were modified from an article by Dynamic Outcomes Inc. (Paynter, 2010) and compared to current trends that were recorded for this study. The exchange rate forecast for 2017 was used for the remainder of the life of mine being evaluated.

Table 7. Exchange rate forecasts (ZAR/US$) that will be used for the evaluation. (Source: modified from Paynter, 2010).

Year	ZAR/US$ (real 2011)
2011	6.8
2012	7.1
2013	6.9
2014	7
2015	7.2
2016	7.5
2017	7.5

8.1.3. Basket Price (MR and UG2)

In order to estimate/determine the revenue earned from selling the product a weighted metal price must be calculated. This calculation is done by using the prill split percentage for the 4E (Pt, Pd, Rh, Au) elements, the expected metal price, and the expected exchange rate. Table 8 shows the prill split percentages for the MR and UG2 (Anglo Platinum Annual Report, 2009).

Table 8. The prill split percentages for the MR and UG2 4E elements that will be used in the evaluation (Anglo Platinum Annual Report, 2009).

MR	Pt%	Pd%	Rh%	Au%
	58.0	31.3	2.7	8.0
UG2	Pt%	Pd%	Rh%	Au%
	43.1	47.2	8.2	1.5

The basket price was calculated as follows:

Step 1: Convert metal prices in US$ to ZAR by multiplying each metal price with the exchange rate.

Step 2: Use prill split percentage to determine ZAR value for each element by multiplying each ZAR value with the relevant prill split percentage.

Step 3: Add the ZAR values for all the elements for the final basket price.

The calculated basket prices for the MR and UG2 that will be used for the evaluation are shown in table 9 and 10 (MR and UG2 respectively) below.

Table 9. The 4E basket price (ZAR) calculated using the prill split, metal price forecasts and exchange rate forecasts for the MR.

Year	Pt	Pd	Rh	Au	ZAR/US$	4E Basket Price
	(US$, real 2011)				(real 2011)	(ZAR)
2011	1730	594	2383	1341	6.8	9,254.41
2012	1770	607	2390	1336	7.1	9,854.81
2013	1781	609	2285	1315	6.9	9,594.39
2014	1794	620	2017	1304	7	9,753.51
2015	1779	623	1886	1288	7.2	9,941.62
2016	1752	611	1500	1279	7.5	10,126.67
2017	1700	573	1500	1204	7.5	9,766.27
	Based on MR prill split					

Table 10. The 4E basket price (ZAR) calculated using the prill split, metal price forecasts and exchange rate forecasts for the UG2.

Year	Pt	Pd	Rh	Au	ZAR/US$	4E Basket Price
	(US$, real 2011)				(real 2011)	(ZAR)
2011	1730	594	2383	1341	6.8	8,442.33
2012	1770	607	2390	1336	7.1	8,984.30
2013	1781	609	2285	1315	6.9	8,708.86
2014	1794	620	2017	1304	7	8,755.66
2015	1779	623	1886	1288	7.2	8,890.39
2016	1752	611	1500	1279	7.5	8,892.67
2017	1700	573	1500	1204	7.5	8,581.62
	Based on UG2 prill split					

The difference in price between the two reefs is directly linked to the prill split percentages. The MR has a higher price with current market conditions. This is due to the higher prill split percentage (table 8) of Pt and Au, as compared to the UG2 which has a higher percentage Pd. The influence of the Pd price on the price of the UG2 has already been shown in section 1.2. Actual spot prices will be used to test the basket price changes in the sensitivity analysis.

The basket price calculated for 2017 (MR and UG2) was used for the remainder of the life of mine being evaluated.

8.1.4. Taxes and Royalties

The tax rate that will be used for this evaluation is the South African Ordinary Companies Tax rate of 28 %. Secondary tax on companies (STC) of 10 % must be calculated from the selling of dividends, as this evaluation will not calculate any dividends this tax will not be included.

Any company (or person) that extracts and sells minerals at a profit is responsible for paying a royalty to the benefit of the National Revenue Fund. The Mineral and Petroleum Resource Royalty Act (2008) gives the formula to calculate the percentage payable for refined and unrefined mineral resources.

This evaluation will assume that all metals extracted will be processed and therefore sold as a refined product. The formula to determine the royalty rate for refined products is as follow:

$$0.5 + \frac{\text{earnings before interest, taxes, depreciation and amortization}}{\text{aggregate gross sales for the assessment period} \times 12.5} \times 100 \qquad (1)$$

The Royalty Bill (2008) also specifies that the percentage determined must not exceed 5 %.

It will be assumed that for the life of mine being evaluated, that the tax rate as well as the formula to calculate the rate of royalties payable will remain constant.

8.1.5. Capital Requirements

The capital requirements for the UG2 have already been calculated by the project team. A total capital requirement of R 7.1 billion was requested and approved by Anglo Platinum during 2008 (Anglo Platinum Annual Report 2009).

It will be assumed that this amount will be sufficient to cover all the start up costs and bring the mine to full production (regardless of the time value of money).

The capital will be used for the establishment of the portal (mine entrance) as well as the shaft sinking development of the three declines down to level four. The top four levels' reef and waste tip development and equipping, as well as the levels' haulage development and equipping up to the second cross cut position in both the north and south directions will be funded from the capital.

Underground machinery necessary for the capital phase of mining (LHD, drill rig, bolter and dump trucks) will be acquired.

The capital will cover the establishment of temporary office and underground change house facilities and a fully equipped engineering workshop on surface. The surface portion of the conveyor belt will be constructed and the waste dump site established during the first three years. The capital will also be utilized to upgrade the existing Eskom power lines and substations, upgrade the roads in the vicinity of the mine and construct sufficient water and waste treatment facilities.

The construction of the concentrator plant, tailings dam and reef transport infrastructure has not been included in this capital estimate and will form part of a separate feasibility study. Additional capital will have to be secured for this construction. Any reef (ore) extracted during the pre-production phase of the project will be stockpiled and processed once the plant has been commissioned.

As discussed in Chapter 1, the mining method, mine design and extraction strategy for the MR will be the same as for the UG2. This is due to the similarities in reef thickness, minimum stoping width requirements and dip of the ore body. It will be assumed that the same capital amount (R 7.1 billion) will be required the MR project.

The capital will be spread out as per table 11 (Hartley, 2011, personal communication).

Table 11. The capital (ZAR) breakdown for the first six years of the project.

Year	Capital (ZAR)
2011	2,500,000,000.00
2012	2,400,000,000.00
2013	1,500,000,000.00
2014	530,000,000.00
2015	110,000,000.00
2016	60,000,000.00
Total	**7,100,000,000.00**

Another form of capital that must be taken into consideration is the Stay in Business (SIB) Capital. This is for the replacement and maintenance of permanent fixtures, for example the conveyor belt; trackless machinery; and ventilation shafts. As an estimate, the minimum SIB should be 12 % of your on mine costs (Hartley, 2011, personal communication). SIB capital will replace the Project Capital and must be taken into consideration for the life of mine (LOM) planning.

This evaluation will assume that the SIB percentage will remain constant over the life of mine being evaluated.

8.1.6. Operating Cost: mining and processing

Operating cost is the on-mine and off-mine (concentrator) cost of producing and processing a ton of ore. In the Anglo Platinum Annual Report (2009) the data for each operation with regards to the tons milled (‘000) as well as the on-mine cash cost (R/t) is given. This data was plotted to establish whether any trends exist within the company. The tons and cost comparison plot in figure 33 shows that as the tons increase (blue line), the cost (red line) decreases. The operation that produces only 130,000 tons requires R 1,200 per ton, while the operation that produces 5,500,000 tons only spends R 480 per ton.

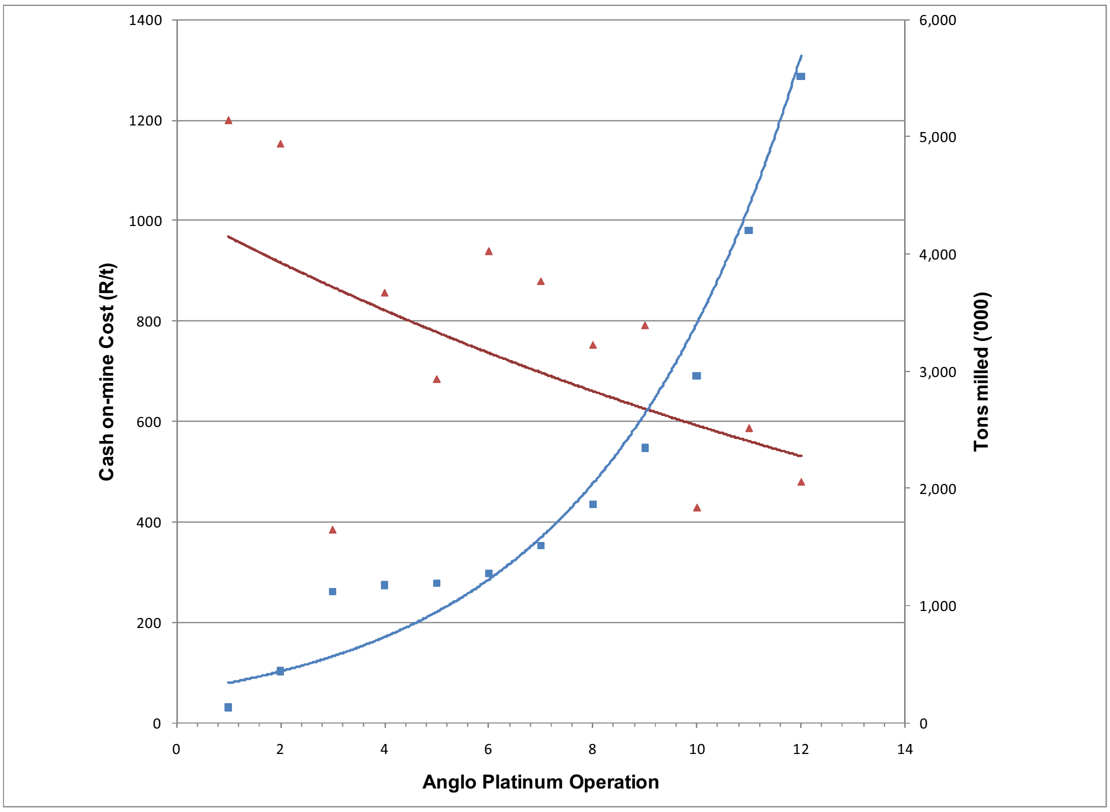

Figure 33. Graph showing the cash on-mine cost per ton (R/t) on the left axis (red line) against the tons milled (‘000) on the right axis (blue line). This is plotted per Anglo Platinum Operation (data from Anglo Platinum Annual Report 2009).

An operation that is starting production (for example the TPM Project) will therefore have a higher operating cost while in the build-up phase. Costs (R/t) will decrease when steady state is reached and more tons are produced.

The on-mine operating cost is made up of the expenses for the day-to-day running of the operation. The general breakdown of the on-mine cash costs for the TPM Project is shown in figure 34. Labour and contractors are the main expense whereas stores, utilities and sundries expenses make up only 19 % of the yearly on-mine budget.

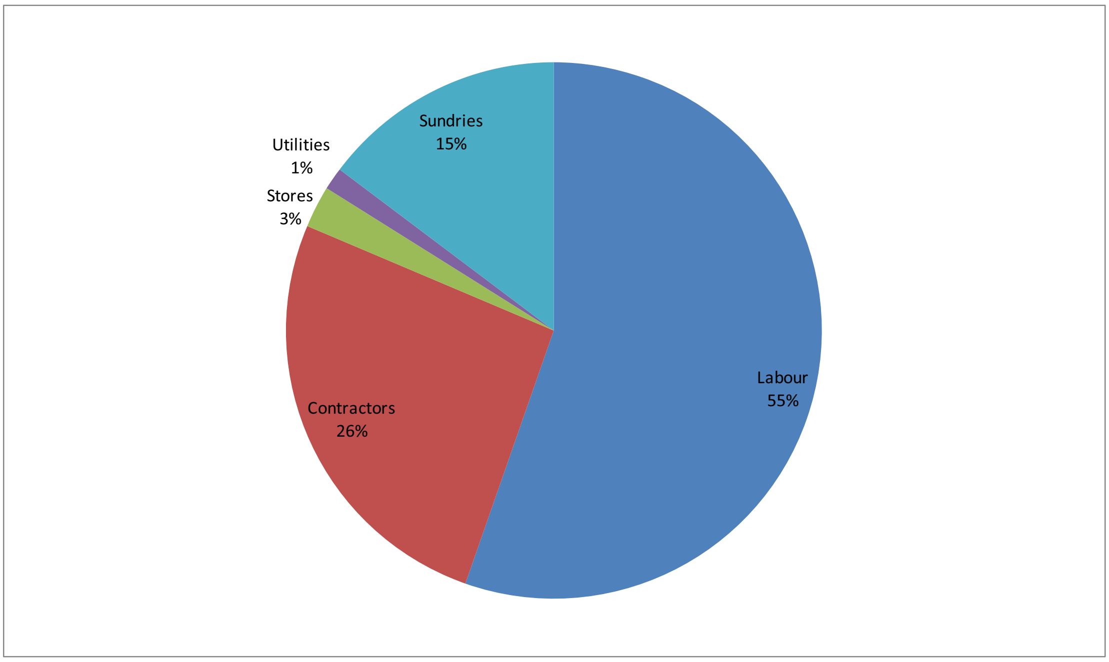

Figure 34. The breakdown of the cash on-mine cost for the TPM Project (Source: Hartley, 2011, personal communication).

The off-mine cost refers to the cost of processing a ton of ore (also in R/t). This will include the cost of ore transportation to the plant and the processing and concentrating of the ore (concentrator costs). The concentrate is then moved to the refinery for smelting and refining of the various metals. It is part of the mine or projects' budget to ensure that the total cost (on-mine and off-mine) is accounted for, from extraction to the refined product.

Indirect costs (OIC) refer to overhead costs that are necessary for the mine to function optimally, such as water, electricity, property or lease area rent, and the project team or any specialist consultants that might be required on occasion.

The operating cost that will be used for this evaluation has been calculated based on the current operating cost for Twickenham (Anglo Platinum Annual Report, 2009) and the average cost per ton milled for a steady state operation within the company. The cost breakdown for the build-up phase and the projected steady state phase is shown in table 12. All costs are in real money terms (no escalation or inflation added).

Table 12. Operating cost breakdown for the TPM Project evaluation showing expected costs during the build-up phase and expected costs during the steady state phase.

Build-up phase		Steady state	
Cost Description	R/t	Cost Description	R/t
On-mine cash cost (R/t)	682	On-mine cash cost (R/t)	345
fixed cost (60%)	*409*	*fixed cost (60%)*	*207*
variable cost (40%)	*273*	*variable cost (40%)*	*138*
Concentrator cost (R/t)	92	Concentrator cost (R/t)	92
Indirect costs (OIC) R/t	406	Indirect costs (OIC) R/t	129
Smelting and Refining cost (R/t)	119	Smelting and Refining cost (R/t)	119
Total working cost (R/t)	**1299**	**Total working cost (R/t)**	**685**

The same costs will be applied to both the MR and UG2 evaluations as the mine design, mining method and stope design is nearly identical.

Concentrator costs might vary due to the difference in reef characteristics, but it is assumed that only one project will be approved, and that the relevant concentrator will be designed and commissioned. Therefore it has been assumed that the cost allowance will be sufficient.

8.1.7. Discount rate

The discount rate is the companies' cost of capital. For the project to be considered economical, the resultant IRR must be higher than the discount rate. The discount rate is made up of various risk related percentages, including the cost of capital for the company, any country-related risk, and the type of project (project risk).

A new project will have a higher risk than an expansion project at an existing operation. According to Smith et al. (2009), the current appropriate real discount rate for mining projects in SA is 9 % to 12 %. Therefore if 9 % is assumed to be the rate for an existing operation (Hartley, 2011, personal communication), it can be argued that an additional 2 % can be added for a greenfields project (project risk) and another 1 % for the 'rural' location of the TPM project. This adds up to a risk adjusted discount rate of 12 %, which was used for this evaluation.

8.1.8. Recoveries

The expected recovery of metals from mineral processing and concentration processes must be considered to ensure the accuracy of ounce and revenue estimations. Mineralogical and metallurgical studies performed on drill core samples in the exploration phase (Roberts and Malysiak, 2004 and Roberts and Shamaila, 2005), as well as current data from adjacent operations (Coetzee, 2011, personal communication) indicate that a 90 % recovery of Pt and Pd can be achieved on the UG2; and 92 % Pt and Pd on the MR.

8.1.9. Grade

The 4E (Pt, Pd, Rh, Au) resource grade (geological loss included) for the UG2 in the area under investigation is 6.34 g/t (Anglo Platinum Annual Report, 2009). This is over a stoping width of 93 cm. The 4E resource grade for the MR is 5.02 g/t (Anglo Platinum Annual Report, 2009) over a stoping width of 100 cm.

The resource grade will differ from the expected head grade as no processing or mining dilution factors have been considered. The reserve grade, which accounts for the mining related dilution factors, will be used in the sensitivity analysis.

The resource grade reflects the maximum grade expected from the optimal extraction of the reef horizon, therefore it will be very unlikely for the grade to increase. Grades could increase if the parameters used in the resource estimation are changed. Parameters that could change include the geotechnical considerations; the minimum stoping width; or if additional grade information is used (closer borehole or sample grid spacing) in the resource estimation.

The reserve grade is usually lower than the resource grade, as factors related to the mining/extraction of the reef are taken into consideration. The reserve grade is usually more difficult to determine, especially if no mining or processing history is available to base assumptions on.

This evaluation will assume that the grade will remain consistent over the life of mine being evaluated.

8.1.10. Tonnage profiles (MR and UG2)

The tonnage profile is critical to the extraction plan of the resource. This profile is created based on the available resource tons (and subsequent reserve figures), as well as the optimal production/extraction rate. The life of mine can be calculated and various scenarios tested to ensure that the area is extracted to its full potential and to the best economical benefit/profit to the company.

The assumption is that neither project has commenced mining and will therefore show similar build-up and steady state profiles. The steady state tonnages are based on the calculated optimal extraction rates as designed by the project team for the reef properties (density and resource cut) and mining method selected.

The capacity for the MR has been calculated at 210 ktpm or 105 ktpm per shaft system (Rhodes, 2011, personal communication). This equals 2,520 kilotons per year at steady state. The tonnage profile for the MR is shown in figure 35, and with this profile the life of mine is 33 years.

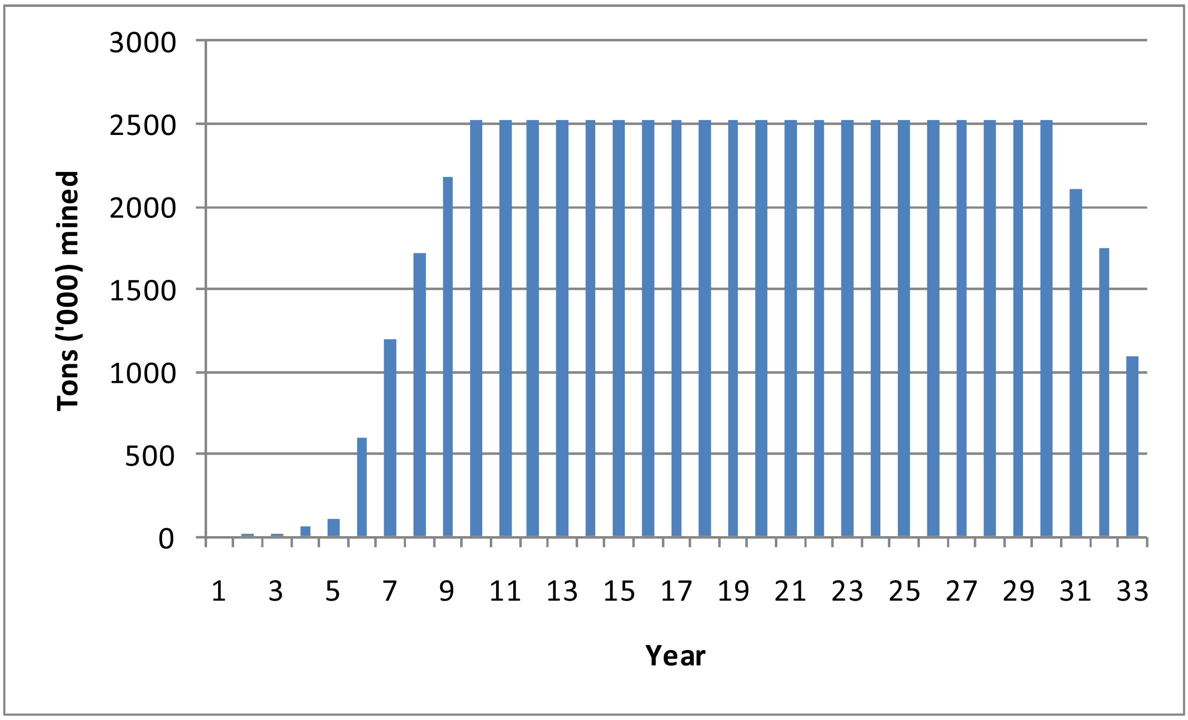

Figure 35. The tonnage profile for the MR at the TPM Project.

The capacity for the UG2 has been calculated at 250 ktpm or 125 ktpm per shaft system (Rhodes, 2011, personal communication). This equals 3,000 kilotons per year at steady state. The tonnage profile is shown in figure 36 and with this profile, the life of mine is 32 years.

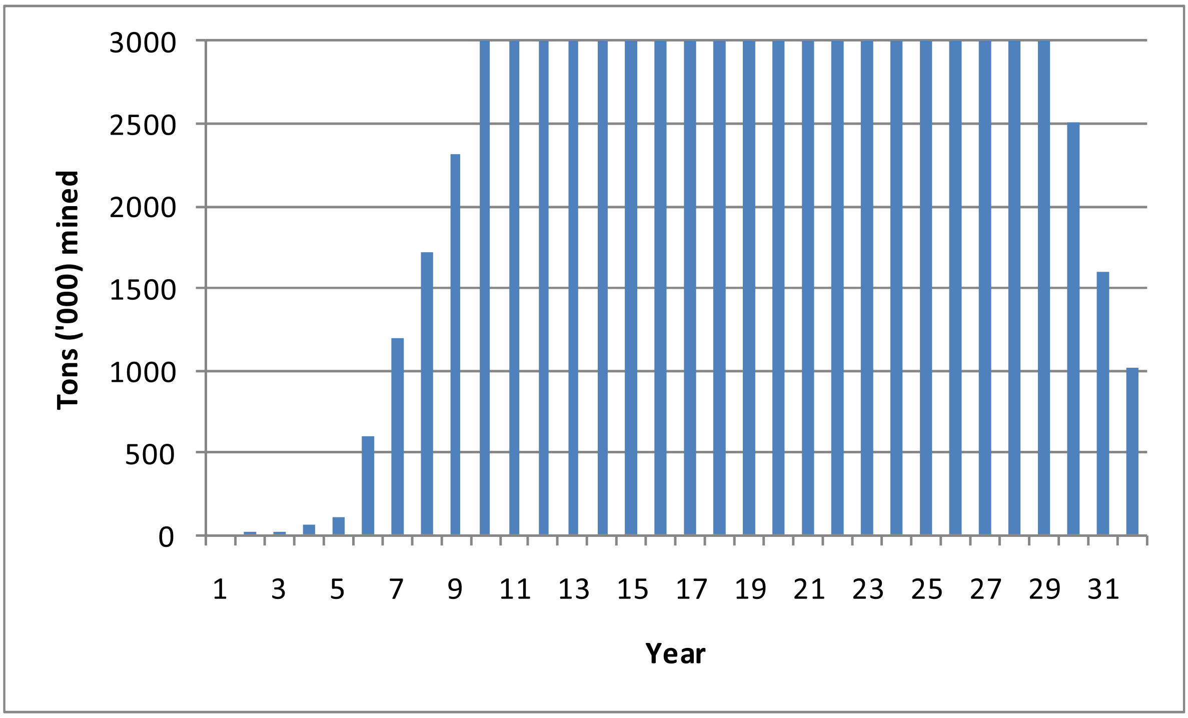

Figure 36. The tonnage profile for the UG2 at the TPM Project.

This evaluation will assume that the tons planned will be achieved yearly and that all tons mined will be milled and processed in the same year. No lag times have been accounted for.

8.2. ASSESSMENT

The input parameters, as discussed in section 8.1, were used to construct a DCF model. The main differences between the two evaluations were:

- the amount of tons mined during steady state (MR 2,520,000 t vs. UG2 3,000,000 t);
- the life of mine duration (MR 33 years vs. UG2 32 years);
- the grade (MR 5.02 g/t vs. UG2 6.34 g/t);
- the recovery percentage (MR 92 % vs. UG2 90 %); and
- the basket price (MR ~R 9,766 vs. UG2 ~R 8,581).

The parameters that were assumed to be the same for both evaluations were:

- the operating cost (R1,299 per ton for the build-up and R685 per ton at steady state);
- the capital amount of R7.1 billion;
- the SIB capital requirement of 12 %;
- the formula to determine the percentage royalties due (equation 1); and
- the South African Ordinary Companies tax rate of 28 %.

The DCF was constructed by using the tons as calculated for the build-up and steady state, to determine the expected 4E oz per annum. The basket price was then applied to determine the revenue that would be received from the sale of the 4E oz.

All costs (operating cost, capital and SIB capital) were subtracted to determine the 'cash flow before (interest and) tax' (EBIT). The royalties' formula (equation 1) was applied to calculate the annual royalties owed. The royalties' percentage ranged from 0.88 % to 4.49 % at steady state. The ZAR amount for the royalties was subtracted to determine the 'earnings before tax' (EBT). Ordinary companies' tax of 28 % was used over the full 33 years life of mine and the resultant 'nett cash flow' (ZAR) was calculated.

The formulas used for the MR DCF is shown in Appendix A (table A1) and the full DCF is shown in table A2 (Appendix A).

The 'nett cash flow' calculated for the MR shows that the cash flow becomes positive in the sixth year, the capital amounts in the first five years exceeds the revenue earned. These cash flows were used to determine the NPV (12 %), IRR and payback period for the MR.

The DCF for the UG2 was constructed in exactly the same manner, using the input parameters specific to the UG2 reef. Differences include the tonnage profile and life of mine (32 years for the UG2 and 33 years for the MR) as well as the grade and basket price. The operating cost per ton, capital requirements, SIB capital percentage, royalties' formula and tax rate are the same for both reefs. The calculated royalties' percentages ranged from 1.48 % to 4.81 % at steady state. The formulas used for the UG2 DCF is shown in table A3 (Appendix A) and the full DCF is shown in table A4 (Appendix A).

The 'nett cash flow' calculated for the UG2 also becomes positive in the sixth year. The cash flows were used to calculate the NPV (12 %), IRR and payback period for the UG2.

The comparison of the NPV, IRR and payback periods calculated for the MR and UG2 are shown in table 13.

Table 13. Summary of DCF assessment results for the MR and the UG2 at the TPM Project. Table show the NPV, IRR and payback period calculated for each reef.

MR		UG2	
NPV (12.0%)	**R -1,664,541,443.47**	NPV (12.0%)	**R -109,614,208.27**
IRR	9%	IRR	12%
Payback Period (yrs)	19.25	Payback Period (yrs)	16

The MR evaluation results show that the NPV is negative and the IRR is 9 %, well below the required discount rate of 12 %. The payback period is 19.25 years, more than half of the 33 years life of mine for the project area evaluated.

The UG2 evaluation also produced a negative NPV but the IRR is 12 %, equal to the required discount rate. The payback period is 16 years, half of the 32 years life of mine.

The DCF results shows that neither the MR nor the UG2 are economically viable as standalone projects with the forecasts and cost requirements used in this financial evaluation.

The UG2 evaluation results are more favourable than that of the MR. The project team could use this outcome as an opportunity to re-evaluate the input parameters to determine where value can be increased or costs saved.

A comprehensive sensitivity analysis will have to be conducted for the MR project in order to establish which factors will contribute to the project becoming economically viable. The outcome of the sensitivity analysis will determine the way forward for the MR project.

Input parameters that could be changed or controlled during mining must be evaluated to determine the impact on the financial viability to the life of mine. These could be from re-designing or altering the planning sequence to add value, such as spreading out the capital requirements without negatively influencing production, or controlling operating cost during steady state. If the range of operating cost, where a positive NPV and IRR can be achieved is known and deemed practical, plans can be put in place to achieve this.

The tonnage profile could potentially be altered to determine if there is an advantage to the life of mine. Factors not within the control of the project team or mine should also be tested, to produce a range in which a currently unfeasible project can become feasible, such as metal prices and exchange rates.

A sensitivity analysis on the grade and metal recoveries will also show what impact inefficiencies in mining or concentration will have on the financial viability of the project, this will provide a guideline for controls that needs to be in place to ensure optimal extraction, grade control and plant/concentrator efficiency. The necessity for additional studies or optimization of processes can also be determined.

9. SENSITIVITY ANALYSIS

A sensitivity analysis is done to determine the impact of changing variables on the NPV and final viability of the project. The sensitivity analysis is used to investigate and quantify some of the risks involved with a project. The main factors that will be evaluated for the MR and UG2 are grade, recovery, basket price, capital cost, and operating cost.

These factors were chosen because they are critical variables in any mining project. They also provide the potential to the project team to change or optimize the project, if the impact and limitations are fully understood.

The grade variances that were used for the sensitivity analysis were obtained from the 2009 / 2010 Anglo Platinum Annual Report. Some Anglo Platinum Eastern and Western Limb operations' 4E built-up head grade and reserve grade were used. This also provides a rough benchmark for the TPM Project with regards to grades actually achieved.

Metal recoveries are a very important factor, if the impact of changing recovery percentages is found to be significant this must be emphasized, and given the necessary attention by the project team.

The basket price is not within the control of the project team, but plays a vital part in the determination of the revenue for the company. The sensitivity of the MR and UG2 to changes in the basket price will be determined by using actual 'spot' prices, which have been collected over a one year period.

Basket price sensitivity information is useful when global market conditions or economic factors change, and a previously un-economic project becomes viable due to the change in commodity prices or exchange rates. This is also true for the opposite, where a marginal project becomes un-economic due to commodity price or exchange rate changes.

Underground mining projects' capital requirements are substantial, and therefore must be very well designed and motivated to assure the investor that all the expenditure is critical and relevant. The impact of savings and overruns in capital cost must be very well understood to assist with decision making.

Operating cost is the final factor that will be tested to determine the impact that overspending or cost saving could have on the final outcome of the project. This factor is within the control of the project team or mine management, and could be adjusted and controlled on site as an ongoing concern.

By changing only one parameter at a time, a range of NPV amounts was determined, and the impact (sensitivity) of the changed factors can be evaluated. The cash flow results from the DCF calculations using these changed factors are given in Appendix B.

9.1. MERENSKY REEF

The 4E resource grade for the MR evaluation was 5.02 g/t. The grade for the sensitivity analysis was changed by 1 g/t intervals to determine the potential impact to the projects NPV and IRR. The range analysed was from 3.02 g/t to 6.02 g/t.

The recovery percentage expected from the plant was varied with 0.5 % intervals; a range between 80 % and 99 % recovery was tested. The base case for the MR evaluation was 92 % recovery.

The basket price is critical as it has a direct influence on the revenue the company receives from the sale of the product. For the sensitivity analysis a variety of prices were used, three spot prices collect over a seven month period as well as random prices from previous years (table 14). This was to determine the impact on the projects NPV over a range of values between R 7,500 and R 10,900 per 4E ounce.

Table 14. Table showing the range of basket prices used for the MR sensitivity analysis.

Date	Metal prices (US$)				R/$	Basket Price
	Pt	Pd	Rh	Au		(ZAR)
18-Feb-11	1842	850	2630	1386	7.18	**10,889.31**
17-May-11	1776	722	2200	1493.7	6.98	**10,017.90**
20-Aug-11	1880	757	1950	1854.1	7.18	**10,973.33**
2009	1394	317	1923	927	7.34	**7,588.51**
2010	1455	331	1221	1070	8.15	**8,678.18**
Based on MR prill split						

The basket price for the financial evaluation was varied for the first seven years where after the same price was used for the remainder of the project life. To simplify the basket price sensitivity analysis it was decided to use a constant price for the entire analysis. The influence was first tested on the base case by using the price for year seven (R 9,766.27) from year one to year 33. The difference in NPV was -0.78 % (R 12,920,193.92) and the IRR did not change from 9 %.

The capital requirements for the project were altered by 10 % intervals. From the base case the capital for the first six years was reduced by 10 % and 20 % as well as increased by 10 % and 20 % (table 15).

Table 15. The changes to the capital requirements for the MR sensitivity analysis.

Project Capital Requirements - ZAR ('000)							
	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	TOTAL
-20%	1,500,000.00	1,200,000.00	950,000.00	830,000.00	800,000.00	400,000.00	**5,680,000.00**
-10%	2,000,000.00	1,900,000.00	1,300,000.00	700,000.00	390,000.00	100,000.00	**6,390,000.00**
Base Case	2,500,000.00	2,400,000.00	1,500,000.00	530,000.00	110,000.00	60,000.00	**7,100,000.00**
10%	3,000,000.00	2,300,000.00	1,400,000.00	850,000.00	150,000.00	110,000.00	**7,810,000.00**
20%	3,500,000.00	2,500,000.00	1,600,000.00	500,000.00	270,000.00	150,000.00	**8,520,000.00**

Operating cost was the final variable selected for the sensitivity analysis as this is potentially a factor that could be controlled or changed during the project life through optimization or changes implemented by the management team. Two scenarios were tested. Firstly the build-up cost per ton (R/t) was altered from the base case amount of R 1,299 to R 1,100 and R 900 while the steady state rate of R 685 was kept the same. Secondly the build-up cost (R 1,299 per ton) was kept constant and the steady state cost was altered from the base case R 685 to R 750 and R 600.

The resultant NPV (12 %) amounts for all the sensitivities (grade; recovery; basket price; capital; and operating cost) have been plotted to determine the potential impact to the viability to the MR project. Figure 37 shows the comparison of the results. The cash flows, NPV and IRR for each variable evaluated are given in table B1 (Appendix B).

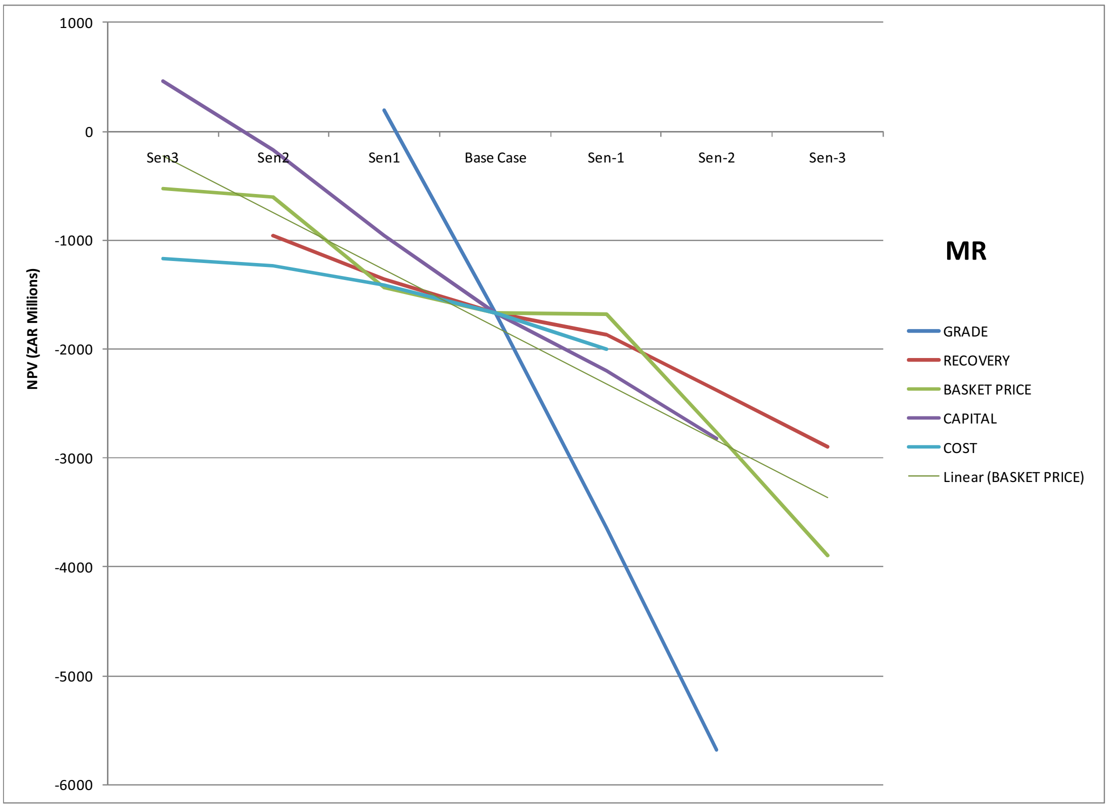

Figure 37. Graph showing the comparison of NPV (12 %) results obtained for the various factors tested for the MR sensitivity analysis.

The sensitivity analysis shows that grade is a critical factor and even small changes (1 g/t) will make a significant difference to the cash flow of the project. It also indicates that by improving/increasing the basket price, recovery percentage, and operating cost, this project does not become positive in terms of the NPV. The MR grade was increased to 6.02 g/t and did produce a positive NPV, but such a grade increase is highly unlikely, as the resource grade was used for the base case analysis (no dilution included). The only factor that produced a convincing positive NPV was a 20 - 30 % decrease in initial project capital requirements.

9.2. UG2 REEF

The 4E resource grade for the UG2 evaluation was 6.34 g/t. The grade for the sensitivity analysis was changed by 1 g/t intervals to determine the potential impact to the projects NPV and IRR. The range analysed was from 4.34 g/t to 7.34 g/t.

The recovery percentage expected from the plant was varied with 0.5 % intervals; a range between 80 % and 99 % recovery was tested. The base case for the UG2 evaluation was 90 % recovery.

For the sensitivity analysis of the basket price, the same methodology as discussed for the MR sensitivity analysis, and spot prices were used for the UG2 analysis. The UG2 prill splits were applied to these and the resultant basket prices are shown in table 16. The basket prices obtained varied over a range of values between R 6,600 and R 10,200 per 4E ounce.

Table 16. Table showing the range of basket prices used for the UG2 sensitivity analysis.

Date	Metal prices (US$)				R/$	Basket Price
	Pt	Pd	Rh	Au		(ZAR)
18-Feb-11	1842	850	2630	1386	7.18	**10,280.69**
17-May-11	1776	722	2200	1493.7	6.98	**9,138.84**
20-Aug-11	1880	757	1950	1854.1	7.18	**9,731.02**
2009	1394	317	1923	927	7.34	**6,672.89**
2010	1455	331	1221	1070	8.15	**7,211.26**
Based on UG2 prill split						

The influence of using a constant basket price over the life of mine instead of the variable price that was used in the initial financial evaluation was tested. The base case was altered by using the price for year seven (R 8,581.62) from year one to year 32. The difference in NPV was -15.76 % (R 17,273,835.58) and the IRR did not change from 12 %.

The capital requirements for the UG2 sensitivity analysis was altered in exactly the same way as for the MR analysis. The amounts were changed with 10 % intervals as per table 15.

The changes to the operating cost were also kept the same as for the MR analysis and two scenarios were tested. Firstly, the build-up cost per ton (ZAR/t) was altered from the base case amount of R 1,299 to R 1,100 and R 900 while the steady state rate of R 685 was kept the same. Secondly, the build-up cost (R 1,299 per ton) was kept constant and the steady state cost was altered from the base case R 685 to R 750 and R 600.

The resultant NPV (12 %) amounts for all the sensitivities (grade; recovery; basket price; capital; and operating cost) have been plotted to determine the potential impact to the viability to the UG2 project. Figure 38 shows the comparison of the results. The cash flows, NPV and IRR for each variable evaluated are given in table B2 (Appendix B).

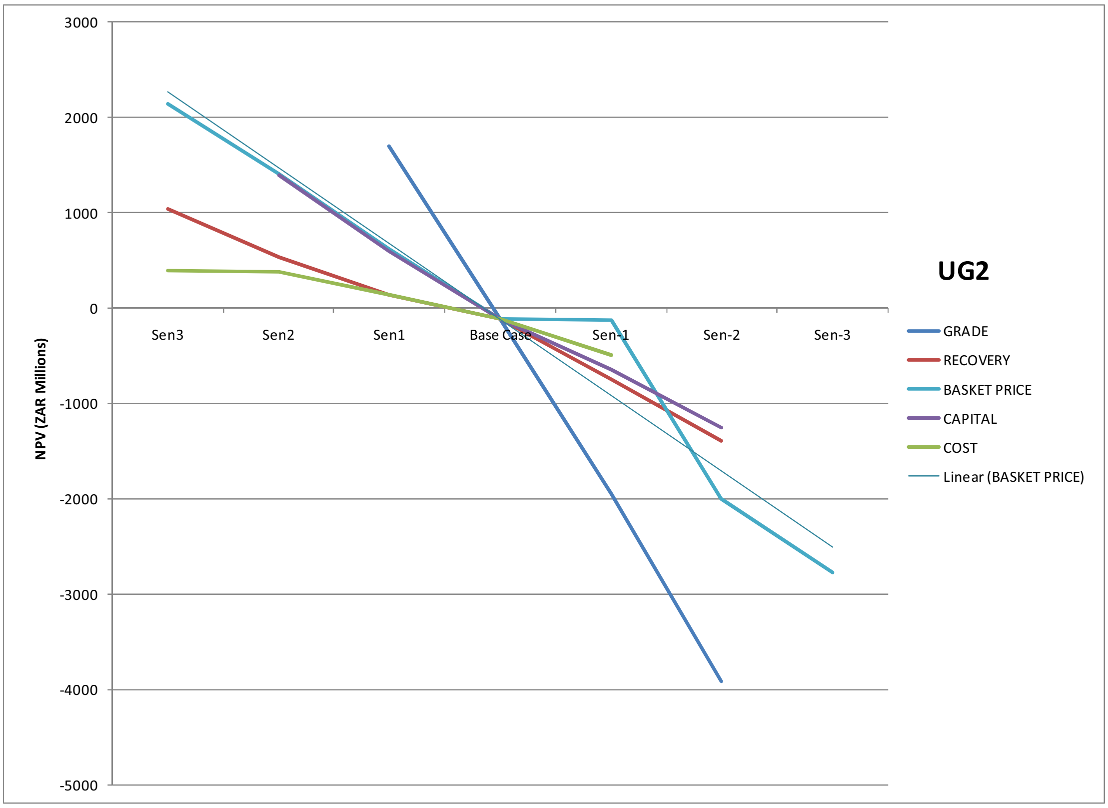

Figure 38. Graph showing the comparison of NPV (12 %) results obtained for the various factors tested for the UG2 sensitivity analysis.

The sensitivity analysis shows that even small changes to the recovery percentage; operating cost; and basket price will provide a positive NPV for the UG2 project. A 10 % reduction in capital requirements will also result in a positive NPV. The grade is the only factor that is unlikely to show an increase, as the resource grade was originally used for the evaluation, and any decrease in grade will result in the NPV dropping significantly.

10. RISK ASSESSMENT

Risk is defined as '*the possibility of a loss or gain as a result of uncertainties'* and the risk assessment uses probabilities to quantify the likelihood of changes in the variables used to forecast the results (Dowd, 1994).

The first step in conducting a risk assessment is to identify the risks or factors that can potentially influence the successful outcome of the project. These risks or factors must then be quantified to determine the impact it will have if it happens/occurs. Through this process a risk register is developed. These risks must then be analysed and suitable control measures identified and implemented. These controls can be of a deterring, preventative or corrective nature, as long as the initial risk is mitigated or reduced to an acceptable limit or completely eliminated.

There are many potential risks for a new project. The main ones that could influence the success of the TPM Project, regardless of which reef is being mined, will be discussed and quantified to give an indication of the projects risk profile. The rating of these risks will determine the high and significant risks, as well as the priority in which these risks should be addressed by the project team.

10.1. RISKS IDENTIFIED

The main risks identified for the TPM Project are legislation and government, economic factors, capital, infrastructure, labour, community risks, SHE related risks, build-up risks, ore body risks, and processing risks. The risk register was compiled according to Anglo Platinum procedure (AA SSDP, 2010) and is shown in table 17. Each risk is then discussed in more detail.

Table 17. Table showing the risk register compiled for the TPM Project.

	Risk Analysis TPM Project		
	Risk Register		
No.	Main Risk Identified	Risk Breakdown	Unwanted event
1	Legislation and Govermant		
		nationalisation of mines	loss of investors, job losses, safety, cost
		DMR	section 54, fines
		changes to MPRDA	new regulations, unplanned costs
		taxation	go up, change
		royalties	go up, change
		political	unstability
		local government	new demands
		mining rights	not renewed
2	Economic factors		
		supply and demand	over supply, less demand
		metal prices	drop
		exchange rates	loss of value
		inflation	raise
		cost of capital	go up
3	Capital		
		deferrement	loose value
		cost overruns	over spend
		company profile	changes, cant give us money
		planning	fatal flaws
4	Infrastructure		
		power	not in place, not sufficient
		roads	unsafe, non existant
		surface infrastructure on mine	late, not planned well
		underground infrastructure	late
		water	not sufficient
		training/education	very low, must invest
		stores/spares	not in place, not planned
		accessability	no airport, railways
		management practises and development policies	not implemented, monitored
		health care	low, inadequate
		housing	poverty, not in place
5	Labor		
		skills retention	low, high turnover
		renumeration	very high to keep skills
		health and safety	poverty, high TB, AIDS rates
		strikes/violence	disruption of operation, damages
		theft	disruption, damages, safety
		recruitment	difficult
		training	expensive, long

	Risk Analysis TPM Project		
	Risk Register		
No.	Main Risk Identified	Risk Breakdown	Unwanted event
6	Community		
		relationship	tentative-bad
		access to land/lease agreements	not in place
		upgrading local community	poverty, not delivering on promises
		poverty	major problem
		uprising/vandalism	unhappiness due to poverty
7	SHE		
	Safety, Health and	policies and procedures	not properly implemented, fatalities
	Environment	management	supervision lacking
		occupational health	managed, need good quality doctors
		environmental management	major spill/accident
		environmental conservation	historical/archilogical sites
8	Build-up		
		design	flawed
		construction	late
		commisioning	late, inadequate
		time, cost, quality	late, overspent, poor
		technology	not in place, not appropriate
9	Ore body		
		grade	lower than expected
		structure	more than expected
		dilution/losses	high due to structure/mining practises
		geotechnical	problems, safety, cost
10	Processing		
		recovery	low
		transport	expensive, safety
		management	skills
		production	down on production-cant fill plant

10.1.1. Legislation and Government

This risk is not within the control of the company or project team, but will form part of the project risk rating or country risk. This risk will include issues such as laws, taxation, royalties, and political stability.

The Department of Mineral Resources (DMR) has established a close working relationship with mines in RSA. Regular audits and inspections are conducted on the mines' compliance to Health and Safety, as well as to ensure the mines' standards and procedures are correctly implemented. The DMR is authorized (by the MHSA) to temporarily stop or fine any operation if it fails an audit, or is found to be endangering the safety of employees by any non-compliance.

The risk for the mine is loss of production; loss of revenue and reputational damage due to non-compliance.

The national and regional political stability is a concern to any company, especially with potential changes in local or regional governments. This could influence the process of negotiation and progress, as new political structures (and persons) usually re-evaluates previous agreements or decisions. Renegotiations because of changes to the local or regional governments could cause delays or additional unplanned costs to the project. Political uncertainty will also increase the risk for the project, for example, the recent speculation about nationalization of mines in SA.

Transparency of laws and taxation regimes are important, in order for companies to understand their obligations to the government and for accurate planning. Uncertainty or continuously changing laws will result in a higher project risk. These regulations together with country risk play a crucial role for foreign investment into the country and obtaining additional capital to fund expansions or new projects.

10.1.2. Economic Factors

Economic factors are not in the control of the company or project team, but have a major impact on any project or operation. This will include the global economic climate, as well as supply and demand for metals. The metal prices and exchange rate has a significant impact on the viability of a project.

Other economic factors that will influence the feasibility of a project will include inflation and the cost of capital for the company. If it is relatively easy to get investors or loans from institutions the discount rate for the company will be lower and therefore viability may be easier to achieve.

10.1.3. Capital

The amount of capital required to construct a mine is substantial. Any cost overruns, regardless of the reasons, could be detrimental to the viability and success of the project. It is therefore critical that the designs and planning are done properly, and costed accurately.

There could be some external influences that are not within the control of the project or construction teams, for example the global economic market, or supply and demand changes that influence the availability of capital from the source.

For example, when the company reviews its strategy and long term planning, the new project may not be critical or necessary at present, and therefore the capital is deferred/ postponed to be available at a later stage. This could result in a short fall of capital (mainly due to time value of money and inflation) when the project is started again, or the entire project must be re-scoped to determine the amended capital requirements.

10.1.4. Infrastructure

A mine cannot function in isolation; it requires roads, power, water, and basic primary services to be in place. These primary services will include environmental and regional management practices and procedures, for example, sewage, storm water management, law enforcement, health care, housing, and education.

Secondary services that will have to be established or available include small and medium businesses that supply assistance to the construction and maintenance functions of the mines, storage facilities with spares and supplies, engineering functions, construction and maintenance services, catering, and housekeeping services.

These general services that should be available for the smooth running of a mine will go hand in hand with the actual construction and infrastructure development taking place on site. The project needs to establish the necessary surface and underground infrastructure to ensure the mine can be commissioned successfully.

The mine infrastructure includes conveyor belts, tips, underground development (shaft sinking, off reef development and access to reef horizons), engineering workshops, storage facilities, office and change house facilities, as well as the necessary safety and security measures.

Infrastructural development setbacks may put the project at risk of not meeting required standards or critical deadlines.

10.1.5. Labour

Labour is a critical requirement for any mining or construction project. Mining operations in SA is more labour intensive compared to developed countries. Historically, SA has benefitted from the high availability of relatively inexpensive labour. Current trends indicated that the cost of labour has increased dramatically. Financial pressures on the labour force are the main influence driving the demand for better salaries.

The Eastern Limb is relatively new to large scale mining and the local communities are unfamiliar with this line of business. Therefore the specialized skills are mainly unavailable locally and must be obtained from other regions. Skills retention is therefore a concern and the education and training of the local population is a long term investment. Recruitment will be a potential problem, as skilled labour will have to be sourced elsewhere and remuneration and retention incentives could be high.

Unskilled labour will have to be trained to the desirable level and this will increase training periods and costs. There is also a bigger risk with inexperienced labour to general health and safety, as well as production.

Other potential labour risks include strike action, theft, and violence that could result in losses of production, damages to property, and threats to general safety.

10.1.6. Community Risks

Every business is dependent on the relationship with the local community in which it is located. Therefore upliftment of the local community will be a major focus area as better education; housing; service delivery; and health care will have to be provided. Regular meetings for information sharing and training will have to be put in place to ensure that there is an understanding of the business and opportunities that are available for the community through employment or small business practices.

Not engaging the local communities or local government properly could be unfavourable to the project and the general relationship with the industry or company.

The new project will be reliant on this relationship with the community for additional (or initial) surface rights and lease agreements; surface access for exploration activities; research; and safety of employees and infrastructure.

10.1.7. SHE Related Risks

Safety, Health and Environmental policies and procedures must be in place, properly implemented, and managed to ensure compliance to laws and regulations. This will contribute to safe, healthy and productive employees.

Safety includes, safe working practices and procedures that are continuously monitored to reduce the amount of injuries (and fatalities) to employees; loss of production due to unsafe working conditions, and damages to property.

Occupational health risks must be monitored and controlled. Occupational health risks include, physical and mental health of employees; good housekeeping practices; dust control; availability of clean water; underground ventilation (including exposure to fumes and gasses; velocity of air movement; and acceptable temperatures); and sanitary ablution facilities (underground and surface).

Environmental management and conservation is extremely important, and is a legal requirement that must be adhered to. The management and monitoring of surface water dams and any water discharge that can cause pollution (used water from underground; acid mine drainage; sewage discharge) as well as seepages from waste rock or tailings dams, must be controlled. Pollution caused by hydrocarbon (oil) and chemical spills must be prevented, together with air pollution by dust or emissions from vehicles or concentrator/processing facilities. These issues pose a risk to the well being of the local communities and environment. The consequence of not managing this risk properly can result in a fine or the operation being shut down for non-compliance by the DMR.

10.1.8. Build-up Risks

This risk is related to the actual construction; commissioning; and mining at the new mine. If there are major problems with the designs and construction, it will cause an overrun in time and cost for the initial phase. A late handover to the next phase will cause similar problems. Quality of construction and decision making can be negatively affected if pressure is increased for production. A late commissioning of the concentrator and mining activities will affect the build-up profile as well as delay the planned steady state production.

This type of delay will affect the company in that costs (capital) could be higher than initially planned. The time delay could negatively affect the bigger production (supply) plan for the company and would require revising of the long term strategic plan.

Another factor that can contribute to a delay in the build-up at the project is the availability of the appropriate technology and services required to achieve the plan. Technologies such as specialized machinery ordered and imported from other countries or service delivery. The main cause for this risk could be poor planning or design by the project team, or external factors such as strikes, or a global economic crisis.

10.1.9. Ore Body Risks

Ore body risks will be related to the resource model; the structural model; geological loss estimations; grade; geotechnical considerations; and modifying factors used in the reserve conversion.

The SAMREC code requires an extensive risk assessment on the resource and reserve figures before it can be signed off. With the companies' long term planning process, the modifying factors and dilution (mining; pillars; and geological losses) percentages are thoroughly scrutinized.

There is still a risk factor involved if more potholes or faults disrupt the mining or creates unsafe ground conditions underground. Underground grade control and the managing of unnecessary dilution is important, as grade fluctuations will adversely influence recoveries and ultimately the revenue generated by the mine/company.

10.1.10. Processing Risks

The plant or concentrator is the first step in recovering the value within the extracted ore. The correct procedures and systems are required to optimize this process. A drop in recovery will result in a loss or reduction of revenue for the company and could render the project uneconomical.

Another factor for the successful and optimal operation of the concentrator is directly related to production from underground. If the production targets are not achieved, there could be problems filling the plant with ore and the cost effective running of the facility will be affected.

The transport of ore from the mine to the concentrator is another potential safety and cost concern. The method of transportation will be reliant on the distance of the concentrator from the mine. Methods currently being evaluated include road transport via ore trucks; overhead cable ways; and pumping via underground/surface tunnels.

10.2. RISK PROFILE

These main risks were tabulated and more specific concerns extracted and quantified. Each concern was then assigned a likelihood of occurrence, between 1 – 5, where 5 is almost certain to occur and 1 is rare. As well as a consequence, also between 1 – 5, where 5 refers to a major impact and 1 would be a minor impact. This table is shown in Appendix C (table C1). Each concern was plotted on the risk matrix table and the weighting read off from the template (table C2, Appendix C). The weightings for each risk were averaged and the resultant risk profile is shown below in figure 39, and plotted as a bar graph in figure 40.

		Consequence				
		Minor (1)	Low (2)	Medium (3)	High (4)	Major (5)
Likelihood	Almost certain (5)		Legislation and Goverment (16) and SHE	Infrastructure (20) and Labour	Build-up (23)	
	Likely (4)			Economic factors (17)	Community (21)	
	Possible (3)				Capital (18)	
	Unlikely (2)				Processing (14)	
	Rare (1)					Ore body (15)

Figure 39. Risk Matrix representation of the TPM Project risk profile. (Modified from AA SSDP, 2010).

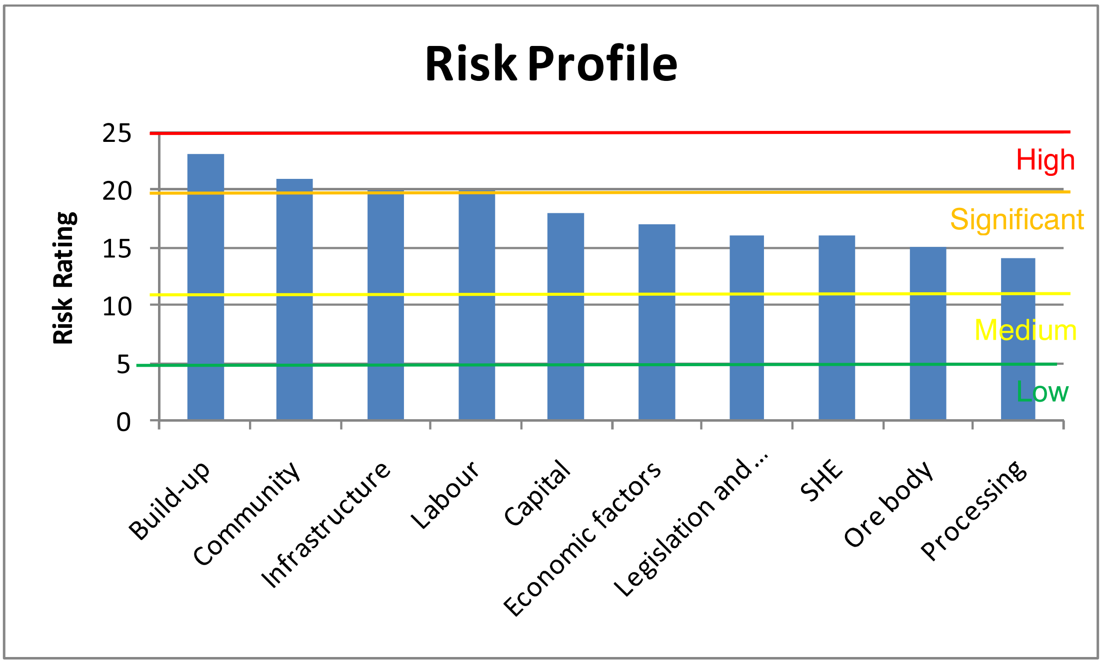

Figure 40. The TPM Project risk profile. Risks are plotted from high risk (red) to low (green) risk, according to the weighting assigned to each by the risk matrix.

The risk profile in figure 40 shows that the build-up phase of the project, as well as community relations are high risks and therefore critical for the success of the project. These risks should be addressed first. Proper controls need to be put in place to ensure these risks are properly monitored and controlled. The other risks discussed are all significant. These should be discussed by the project team and a proper risk management plan should be constructed. The risk management plan must be implemented and monitored to minimise potential impact. Risks such as the economic factors, legislation and government related risks are not in the control of the project team. These risks will have to be managed by monitoring them closely for signs of change.

11. SUMMARY OF RESULTS

The financial evaluation showed that both the MR and UG2 projects have negative NPV's, therefore making both projects unfeasible as standalone investments.

11.1. MERENSKY REEF

The MR evaluation resulted in an IRR of 9 %, which is well below the discount rate of 12 % required by the company to be viable. The sensitivity analysis conducted on the MR tested the influence of changes to various input factors. Changes in the grade resulted in a line with a very steep gradient, indicating a high sensitivity to this factor. The MR for the TPM project is only reported as a resource, no reserve has been declared. Reserve grades and head grades from other MR operations were obtained from the Anglo Platinum Annual Report (2009; 2010) and used for comparison.

The only other MR operation currently working on the Eastern Limb is Bokoni Platinum. The 2010 4E reserve grade for Bokoni was 4.14 g/t and they achieved a head grade of 4.07 g/t. Mines on the Western Limb achieved head grades of 4.51 g/t (Tumela) and 4.28 g/t (Khuseleka) according to the 2010 Anglo Platinum Annual Report. This means that when the mining and dilution factors are considered, the reserve grade for the TPM Project will be substantially reduced. The reserve grade should be in the region of 4.22 g/t.

The Bokoni reserve grade (4.14 g/t) was substituted into the MR DCF, and resulted in a NPV (12 %) of R -3,398,000,156.19 and an IRR of 5 %.

The recovery percentage was tested to determine if a more efficient recovery process could result in the project becoming viable. The results showed that even at a 99 % recovery the MR project gives a negative NPV (R -952,941,837.84) and an IRR of 10 %.

A range of basket prices were used in the sensitivity analysis to represent different economic or market conditions. The MR basket price is generally higher than that of the UG2 because of the difference in prill split percentages. The assumptions used to determine the basket price for the DCF was generally lower than the basket prices calculated for 2011. Even with the current high gold price

(US$ 1,854 on 20 August 2011), the NPV was negative (R -524,326,182.51) and the IRR 11 %.

The MR only becomes viable (positive NPV and IRR 12 %) when the basket price reaches R 11,600.

The operating cost is potentially a variable that can be changed or managed to produce better results or operate at a more efficient cost per ton. The two scenarios tested did not produce a positive NPV for the MR evaluation. The NPV became positive at a build-up rate of R 685 per ton with a steady state rate of R 450 per ton.

This is significantly lower than what is currently envisioned for the costs on the Eastern Limb. Mines on the Western Limb are achieving steady state rates in this vicinity, but with much higher tonnages produced (Anglo Platinum Annual Report, 2009; 2010).

The Western Limb operations have different challenges compared with a project on the Eastern Limb, mainly due to the lack of infrastructure, skilled labour, and support services. Another factor that contributes to higher costs will be the construction of a new mine, as opposed to the expansion or replacement area for an existing operation, as is the case in the Western Limb.

The final variable that was tested was the capital requirements for the project. This variable showed that a decrease of 20 – 30 % in the capital over the first six years of the project build-up, will give a positive NPV (R 457,051,834.11) and an IRR of 13 %.

A 30 % capital reduction equals an R 2.1 billion saving. To achieve this, the scope of the project will have to be revised to determine where the cost saving can be accomplished.

The risk assessment identified the build-up phase of the project as a high risk. The build-up is critical to the operation, for if this is delayed or not executed to standard, the entire project will be in jeopardy. The sensitivity analysis showed that the capital requirements and build-up costs will have to be reduced for the MR project to be economically viable.

The second high risk identified is the community. A good relationship with the local community will have to be established prior to, or very early in the project timeline. A good relationship with the local community will assist the operation in obtaining permission to conduct exploration activities as well as gain access to surface areas for critical infrastructure development.

A good community relationship goes hand in hand with the related significant risks of labour and infrastructure development.

The MR is not a viable option to pursue as a standalone project. The sensitivity analysis showed that only an increase in grade (which is unlikely to occur) or a significant decrease in costs (operating and capital) will render this project viable. The options available to extract the MR resource must be re-engineered, to include either a different mining method or a different extraction strategy. Possible examples include looking at an extended life of mine (past 30 years); utilizing common infrastructure with a neighbouring project (concentrator); or the evaluation of both the UG2 and MR as one project.

Ways of saving costs must be explored, for example using working cost or SIB to fund the initial phase of construction on the MR and in doing so reducing the capital requirements for this project.

There are eight significant and two high risks related to a new project on the Eastern Limb that must be addressed and adequately mitigated.

11.2. UG2 REEF

The UG2 achieved an IRR of 12 % which is equal to the discount rate required. Even with a negative NPV this implies that some re-designing or optimization could result in this project becoming viable.

The sensitivity analysis indicated that grade is a factor that will have to be controlled, as a slight decrease in grade makes a significant change to the NPV and IRR. The resource grade (6.34 g/t) was used for the initial evaluation, but the reserve grade has been calculated and was published in the 2010 Anglo Platinum Annual Report. The reserve grade for the TPM Project is 5.37 g/t, and if this is used in the evaluation, the result is a NPV of R -1,954,144,712.91 with an IRR of 9 %. This is a decrease of 1,683 % from the base case.

Reserve grades and head grades from other UG2 operations were obtained from the 2010 Anglo Platinum Annual Report for comparison. The neighbouring Bokoni Platinum published a 4E reserve grade of 5.25 g/t, and achieved a 4.74 g/t head grade for 2010.

The published 4E head grade for Modikwa Platinum Mine, to the south of the TPM Project, was 4.73 g/t and Mototolo indicated a head grade of 3.33 g/t. Tumela Mine on the Western Limb published a 4E head grade of 4.46 g/t, which is slightly lower than the Bokoni and Modikwa (Eastern Limb) grades.

The reserve and head grade figures mentioned, confirm that the resource grade is very high, and that it will be very unlikely for the TPM Project to achieve. The major drop in NPV and IRR when the current reserve grade is substituted into the evaluation is a concern, as the project becomes uneconomical.

The recovery percentage was adjusted to determine if a more efficient recovery process could result in the project becoming viable. The results from the sensitivity analysis showed that an increase to 92 % recovery will increase the NPV to R 145,775,588.28 and give an IRR of 12 %.

The change in NPV when the recovery percentage is increased by 2 % is encouraging. Additional work will have to be done to increase the efficiency of the UG2 recovery process or further metallurgical studies are required on the TPM Project UG2, to better understand the metallurgical characteristics. This factor is critical, as the UG2 concentrator plant is currently being planned and this information could have a significant impact on the design requirements for this plant, as well as potential cost implications.

A range of basket prices were used in the sensitivity analysis to represent different economic or market conditions. The UG2 basket price is generally lower than that of the MR, mainly because of the difference in prill split percentages. The assumptions used to determine the basket price for the DCF was generally lower than the basket prices calculated for 2011.

The basket price was calculated using the current high gold price (US$ 1,854 on 20 August 2011) which worked out to R 9,731 per 4E ounce. The resultant NPV equalled R 1,406,572,565.06 and the IRR 14 %. The sensitivity analysis showed that the UG2 project will be economically viable with a basket price above R 9,138 per 4E ounce.

The operating cost is potentially a variable that can be changed or managed to produce better results or operate at a more efficient cost per ton. The two scenarios tested in the sensitivity analysis showed that by reducing any of the costs (build-up or steady state) a positive NPV can be achieved. This variable should definitely be pursued by the project team and with efficient planning; systems can be put in place to ensure costs are kept within acceptable limits.

The last variable tested in the sensitivity analysis was the capital requirements for the project. The NPV increased to R 596, 616,577.82 and produced an IRR of 13 % with a 10 % decrease in capital. A 10 % reduction in capital represents an R710,000.00 saving. This type of saving can be achieved through careful planning and re-scheduling of activities. The project scope should be re-assessed to identify potential non-critical components that could be deferred or funded from other sources.

The risk assessment identified the build-up phase of a project on the Eastern Limb and the general relationship with the local community as the two highest risks. The build-up phase for the UG2 project is critical, as the infrastructure must be in place for the operation to reach steady state. The other risk factors that is linked with this, is the training of the people that will work at the operation and the availability of adequate working areas underground to produce the required tons at steady state.

The relationship with the local community is very important as the future of the operation is dependent on this relationship. The mine will require additional surface lease areas in order to expand and will have to conduct further studies and exploration activities as the mining progresses. This can include upgrading of the resources; or structural information for geological loss estimations; or geotechnical information for dams, silos or ventilation shafts.

The sensitivity analysis for the UG2 indicated that this project can be economically viable if any one of the following factors is changed: the recovery percentage is improved; the capital and operating cost is decreased, if the basket price stays at approximately R 9,138.00 per 4E ounce, which has been the approximate price since May 2011.

The only concern identified from the sensitivity analysis is the major drop in viability when the grade is decreased. A drop in grade is almost certain to occur. The current published reserve grade caused a drop in the NPV of 1,683 %, for this evaluation. The grade must be considered a major risk to this project. The UG2 project design and input parameters to the financial evaluation will have to be scrutinized, to construct an optimised revised plan.

In summary, the UG2 is not a viable option to pursue as a standalone project. The UG2 however show much more potential to become viable with some adjustments and re-evaluation of input parameters, when compared to the MR. However, a potential fatal flaw in this analysis is the high sensitivity to changes in grade. As grade is not a factor that can be readily controlled or changed, it must be regarded as a high risk. Dilution control underground when stoping commences, as well as strict control when ore is moved to the plant will be critical. Improvement of metal recovery at the plant will make a difference when head grade is calculated, but these factors might not be adequate to render the project viable.

12. CONCLUSION

Comparing the economic viability of the MR to that of the UG2, on the TPM Project area, with the input parameters and assumptions used in this evaluation showed that the UG2 has more potential to be extracted as an economically viable project. The MR project will require additional engineering and re-designing to dramatically reduce the capital and operating costs involved.

The MR evaluation results indicate that this project is not economically viable with the input parameters and assumptions used in this study. The NPV is negative and the IRR well below the required discount rate of 12 %. In terms of the payback period, the initial project capital will only be repaid 19 years into the 33 years life of mine. The sensitivity analysis showed that by reducing the initial capital requirements by 30 % or R 2.1 billion, the project produces a positive NPV. The operating cost can also be reduced to result in a positive NPV, but the ZAR per ton values required, seem unrealistic for a project on the Eastern Limb of the Bushveld Complex.

The Eastern Limb of the Bushveld Complex has unique challenges in that there is limited infrastructure development, and the area largely consists of rural villages and communities, with an associated high level of poverty. The challenges described for the Eastern Limb have a direct implication on mining development, as support structures required to assist and supply services to an operation are not in place or must be sourced from a substantial distance, thus increasing the risk, as well as the costs involved. Required infrastructure such as, electricity, water, waste removal, and transport systems is important for mining and processing activities. With these challenges in mind, planning an operating cost similar to that achieved by existing operations on the Western Limb, seems unlikely.

The UG2 evaluation produced a negative NPV and an IRR equal to the required discount rate. The sensitivity analysis gave encouraging results as a positive NPV was produced with minor changes to the input parameters tested. By increasing the recoveries by 2 % and reducing the initial capital cost by 10 %, the NPV was positive. The UG2 project input parameters must be refined and re-evaluated, as this project shows potential to be extracted as an economically viable project. The only concern is the sensitivity to changes in grade, which must be further investigated to ensure the modifying factors used for the resource to reserve conversion, is optimal.

The revenue, for both the MR and UG2, was calculated using the 4E (Pt, Pd, Rh, and Au). However, the 4E are not the only metals that are being recovered from the MR and UG2. The base metals Cu and Ni are also being refined and sold. Therefore including the base metals into the basket price, or accounting for these when calculating the revenue, could potentially increase the revenue and result in the NPV increasing.

The mining challenges that were described for the Eastern Limb are very similar to the risks that were identified in the project risk assessment. The concern is that these challenges have been identified and described on previous occasions but not resolved. The main challenges and risks are the lack of infrastructure; poverty; need for improved health care and education facilities; availability of water and electricity; local and provincial government not efficient; and the socio-economic development of the local communities.

The global market and economy are closely related to the success of mining operations in South Africa. The demand for Pt is likely to exceed the supply as environmental legislation to promote the reduction of emissions is being enforced in major developed countries. The main demand comes from the automotive sector and is expected to increase with the development of new technologies where Pt is a main component, like the PEM fuel cell.

The most recent development in the global economy is the renewed concerns for another economic downturn, as experienced during 2007-2008. The result could be crippling to the South African mining sector. Labour costs are rising dramatically as a result of increase living costs and inflation. Electricity costs are increasing because of an increased demand and limited supply. The access to capital for new or replacement projects can become limited as fewer investments are made due to the uncertainty in global markets. The political instability within the ruling party and the uncertainty currently prevailing in South Africa on speculation regarding the nationalization of mines, could also negatively affect the investment into the country.

The study has shown that the decision to commence mining on the UG2 instead of the MR was well founded. The DCF model of the UG2 did not show that the ore body is economically viable using the forecasts in this study. However, the sensitivity analysis did show that by reducing the capital or increasing the recoveries slightly, a positive NPV could be achieved at the required discount rate.

With the depletion of PGM resources on the Western Limb, mining companies will have to turn to the Eastern Limb. In order to extract the full potential of PGM wealth on the Eastern Limb, the fundamental issues highlighted in this study will have to be resolved. The MR and UG2 resources at the TPM Project are testament to this potential value in the Eastern Limb. However, current capital and operating cost requirements, together with the current economic climate, do not make this project viable. The project team will have to look at means to reduce these costs and address the challenges in the area, both infrastructural and socio-economic.

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Appendix A

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	1	2	3
Tons	Tons milled ('000)		0	5	10
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((D2*1000)*D3)	=((E2*1000)*E3)	=((F2*1000)*F3)
Oz Content	content 4E oz		=D5/31.10348	=E5/31.10348	=F5/31.10348
Recovery	4E oz recovered	0.92	=D6*0.92	=E6*0.92	=F6*0.92
Basket Price	Basket price (ZAR/4E oz)		9254.41	9854.81	9594.39
Revenue	**Revenue (ZAR)**		**=D6*D8**	**=E7*E8**	**=F7*F8**
	Cost per ton (ZAR/t)		1299	1299	1299
Minus cost3	Operating Cost (ZAR)		=(D2*1000)*D12	=(E2*1000)*E12	=(F2*1000)*F12
	SIB %	0.12			
Minus cost4	SIB Capital				
Equals	**Operating free cash flow**		**=D10-D13-D15**	**=E10-E13-E15**	**=F10-F13-F15**
Minus cost5	Project Capital (R'000)		2500000	2400000	1500000
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=D16-(D17*1000)**	**=E16-(E17*1000)**	**=F16-(F17*1000)**
	Royalties %				
minus	Royalties ZAR				
equals	**Earnings before Tax (EBT)**		**=D18-D21**	**=E18-E21**	**=F18-F21**
minus	Tax	0.28			
equals	**Nett Cash Flow (ZAR)**		**=D23-D25**	**=E23-E25**	**=F23-F25**

NPV (12.0%)	0.12	=NPV(E30,D27:AJ27)
IRR		=IRR(D27:AJ27)
Payback Period		19 yrs 3 months

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	4	5	6
Tons	Tons milled ('000)		60	106	600
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((G2*1000)*G3)	=((H2*1000)*H3)	=((I2*1000)*I3)
Oz Content	content 4E oz		=G5/31.10348	=H5/31.10348	=I5/31.10348
Recovery	4E oz recovered	0.92	=G6*0.92	=H6*0.92	=I6*0.92
Basket Price	Basket price (ZAR/4E oz)		9753.51	9941.62	10126.67
Revenue	**Revenue (ZAR)**		**=G7*G8**	**=H7*H8**	**=I7*I8**
	Cost per ton (ZAR/t)		1299	1299	1299
Minus cost3	Operating Cost (ZAR)		=(G2*1000)*G12	=(H2*1000)*H12	=(I2*1000)*I12
	SIB %	0.12			
Minus cost4	SIB Capital				
Equals	**Operating free cash flow**		**=G10-G13-G15**	**=H10-H13-H15**	**=I10-I13-I15**
Minus cost5	Project Capital (R'000)		530000	110000	60000
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=G16-(G17*1000)**	**=H16-(H17*1000)**	**=I16-(I17*1000)**
	Royalties %				=0.5+(I18/(I10*12.5))*100
minus	Royalties ZAR				=(I20%)*I18
equals	**Earnings before Tax (EBT)**		**=G18-G21**	**=H18-H21**	**=I18-I21**
minus	Tax	0.28			=I23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=G23-G25**	**=H23-H25**	**=I23-I25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	7	8	9
Tons	Tons milled ('000)		1200	1720	2180
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((J2*1000)*J3)	=((K2*1000)*K3)	=((L2*1000)*L3)
Oz Content	content 4E oz		=J5/31.10348	=K5/31.10348	=L5/31.10348
Recovery	4E oz recovered	0.92	=J6*0.92	=K6*0.92	=L6*0.92
Basket Price	Basket price (ZAR/4E oz)		9766.27	=J8	=K8
Revenue	**Revenue (ZAR)**		**=J7*J8**	**=K7*K8**	**=L7*L8**
	Cost per ton (ZAR/t)		1299	1299	685
Minus cost3	Operating Cost (ZAR)		=(J2*1000)*J12	=(K2*1000)*K12	=(L2*1000)*L12
	SIB %	0.12	=$C$14*682	=$C$14*682	=$C$14*345
Minus cost4	SIB Capital		=(J2*1000)*J14	=(K2*1000)*K14	=(L2*1000)*L14
Equals	**Operating free cash flow**		**=J10-J13-J15**	**=K10-K13-K15**	**=L10-L13-L15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=J16-(J17*1000)**	**=K16-(K17*1000)**	**=L16-(L17*1000)**
	Royalties %		=0.5+(J18/(J10*12.5))*100	=0.5+(K18/(K10*12.5))*100	=0.5+(L18/(L10*12.5))*100
minus	Royalties ZAR		=(J20%)*J18	=(K20%)*K18	=(L20%)*L18
equals	**Earnings before Tax (EBT)**		**=J18-J21**	**=K18-K21**	**=L18-L21**
minus	Tax	0.28	=J23*$C$25	=K23*$C$25	=L23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=J23-J25**	**=K23-K25**	**=L23-L25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	10	11	12
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((M2*1000)*M3)	=((N2*1000)*N3)	=((O2*1000)*O3)
Oz Content	content 4E oz		=M5/31.10348	=N5/31.10348	=O5/31.10348
Recovery	4E oz recovered	0.92	=M6*0.92	=N6*0.92	=O6*0.92
Basket Price	Basket price (ZAR/4E oz)		=L8	=M8	=N8
Revenue	**Revenue (ZAR)**		**=M7*M8**	**=N7*N8**	**=O7*O8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(M2*1000)*M12	=(N2*1000)*N12	=(O2*1000)*O12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(M2*1000)*M14	=(N2*1000)*N14	=(O2*1000)*O14
Equals	**Operating free cash flow**		**=M10-M13-M15**	**=N10-N13-N15**	**=O10-O13-O15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=M16-(M17*1000)**	**=N16-(N17*1000)**	**=O16-(O17*1000)**
	Royalties %		=0.5+(M18/(M10*12.5))*100	=0.5+(N18/(N10*12.5))*100	=0.5+(O18/(O10*12.5))*100
minus	Royalties ZAR		=(M20%)*M18	=(N20%)*N18	=(O20%)*O18
equals	**Earnings before Tax (EBT)**		**=M18-M21**	**=N18-N21**	**=O18-O21**
minus	Tax	0.28	=M23*$C$25	=N23*$C$25	=O23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=M23-M25**	**=N23-N25**	**=O23-O25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	13	14	15
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((P2*1000)*P3)	=((Q2*1000)*Q3)	=((R2*1000)*R3)
Oz Content	content 4E oz		=P5/31.10348	=Q5/31.10348	=R5/31.10348
Recovery	4E oz recovered	0.92	=P6*0.92	=Q6*0.92	=R6*0.92
Basket Price	Basket price (ZAR/4E oz)		=O8	=P8	=Q8
Revenue	**Revenue (ZAR)**		**=P7*P8**	**=Q7*Q8**	**=R7*R8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(P2*1000)*P12	=(Q2*1000)*Q12	=(R2*1000)*R12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(P2*1000)*P14	=(Q2*1000)*Q14	=(R2*1000)*R14
Equals	**Operating free cash flow**		**=P10-P13-P15**	**=Q10-Q13-Q15**	**=R10-R13-R15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=P16-(P17*1000)**	**=Q16-(Q17*1000)**	**=R16-(R17*1000)**
	Royalties %		=0.5+(P18/(P10*12.5))*100	=0.5+(Q18/(Q10*12.5))*100	=0.5+(R18/(R10*12.5))*100
minus	Royalties ZAR		=(P20%)*P18	=(Q20%)*Q18	=(R20%)*R18
equals	**Earnings before Tax (EBT)**		**=P18-P21**	**=Q18-Q21**	**=R18-R21**
minus	Tax	0.28	=P23*$C$25	=Q23*$C$25	=R23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=P23-P25**	**=Q23-Q25**	**=R23-R25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	16	17	18
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((S2*1000)*S3)	=((T2*1000)*T3)	=((U2*1000)*U3)
Oz Content	content 4E oz		=S5/31.10348	=T5/31.10348	=U5/31.10348
Recovery	4E oz recovered	0.92	=S6*0.92	=T6*0.92	=U6*0.92
Basket Price	Basket price (ZAR/4E oz)		=R8	=S8	=T8
Revenue	**Revenue (ZAR)**		**=S7*S8**	**=T7*T8**	**=U7*U8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(S2*1000)*S12	=(T2*1000)*T12	=(U2*1000)*U12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(S2*1000)*S14	=(T2*1000)*T14	=(U2*1000)*U14
Equals	**Operating free cash flow**		**=S10-S13-S15**	**=T10-T13-T15**	**=U10-U13-U15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=S16-(S17*1000)**	**=T16-(T17*1000)**	**=U16-(U17*1000)**
	Royalties %		=0.5+(S18/(S10*12.5))*100	=0.5+(T18/(T10*12.5))*100	=0.5+(U18/(U10*12.5))*100
minus	Royalties ZAR		=(S20%)*S18	=(T20%)*T18	=(U20%)*U18
equals	**Earnings before Tax (EBT)**		**=S18-S21**	**=T18-T21**	**=U18-U21**
minus	Tax	0.28	=S23*$C$25	=T23*$C$25	=U23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=S23-S25**	**=T23-T25**	**=U23-U25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	19	20	21
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((V2*1000)*V3)	=((W2*1000)*W3)	=((X2*1000)*X3)
Oz Content	content 4E oz		=V5/31.10348	=W5/31.10348	=X5/31.10348
Recovery	4E oz recovered	0.92	=V6*0.92	=W6*0.92	=X6*0.92
Basket Price	Basket price (ZAR/4E oz)		=U8	=V8	=W8
Revenue	**Revenue (ZAR)**		**=V7*V8**	**=W7*W8**	**=X7*X8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(V2*1000)*V12	=(W2*1000)*W12	=(X2*1000)*X12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(V2*1000)*V14	=(W2*1000)*W14	=(X2*1000)*X14
Equals	**Operating free cash flow**		**=V10-V13-V15**	**=W10-W13-W15**	**=X10-X13-X15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=V16-(V17*1000)**	**=W16-(W17*1000)**	**=X16-(X17*1000)**
	Royalties %		=0.5+(V18/(V10*12.5))*100	=0.5+(W18/(W10*12.5))*100	=0.5+(X18/(X10*12.5))*100
minus	Royalties ZAR		=(V20%)*V18	=(W20%)*W18	=(X20%)*X18
equals	**Earnings before Tax (EBT)**		**=V18-V21**	**=W18-W21**	**=X18-X21**
minus	Tax	0.28	=V23*$C$25	=W23*$C$25	=X23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=V23-V25**	**=W23-W25**	**=X23-X25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	22	23	24
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((Y2*1000)*Y3)	=((Z2*1000)*Z3)	=((AA2*1000)*AA3)
Oz Content	content 4E oz		=Y5/31.10348	=Z5/31.10348	=AA5/31.10348
Recovery	4E oz recovered	0.92	=Y6*0.92	=Z6*0.92	=AA6*0.92
Basket Price	Basket price (ZAR/4E oz)		=X8	=Y8	=Z8
Revenue	**Revenue (ZAR)**		**=Y7*Y8**	**=Z7*Z8**	**=AA7*AA8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(Y2*1000)*Y12	=(Z2*1000)*Z12	=(AA2*1000)*AA12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(Y2*1000)*Y14	=(Z2*1000)*Z14	=(AA2*1000)*AA14
Equals	**Operating free cash flow**		**=Y10-Y13-Y15**	**=Z10-Z13-Z15**	**=AA10-AA13-AA15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=Y16-(Y17*1000)**	**=Z16-(Z17*1000)**	**=AA16-(AA17*1000)**
	Royalties %		=0.5+(Y18/(Y10*12.5))*100	=0.5+(Z18/(Z10*12.5))*100	=0.5+(AA18/(AA10*12.5))*100
minus	Royalties ZAR		=(Y20%)*Y18	=(Z20%)*Z18	=(AA20%)*AA18
equals	**Earnings before Tax (EBT)**		**=Y18-Y21**	**=Z18-Z21**	**=AA18-AA21**
minus	Tax	0.28	=Y23*$C$25	=Z23*$C$25	=AA23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=Y23-Y25**	**=Z23-Z25**	**=AA23-AA25**

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Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	25	26	27
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((AB2*1000)*AB3)	=((AC2*1000)*AC3)	=((AD2*1000)*AD3)
Oz Content	content 4E oz		=AB5/31.10348	=AC5/31.10348	=AD5/31.10348
Recovery	4E oz recovered	0.92	=AB6*0.92	=AC6*0.92	=AD6*0.92
Basket Price	Basket price (ZAR/4E oz)		=AA8	=AB8	=AC8
Revenue	**Revenue (ZAR)**		**=AB7*AB8**	**=AC7*AC8**	**=AD7*AD8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(AB2*1000)*AB12	=(AC2*1000)*AC12	=(AD2*1000)*AD12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(AB2*1000)*AB14	=(AC2*1000)*AC14	=(AD2*1000)*AD14
Equals	**Operating free cash flow**		**=AB10-AB13-AB15**	**=AC10-AC13-AC15**	**=AD10-AD13-AD15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=AB16-(AB17*1000)**	**=AC16-(AC17*1000)**	**=AD16-(AD17*1000)**
	Royalties %		=0.5+(AB18/(AB10*12.5))*100	=0.5+(AC18/(AC10*12.5))*100	=0.5+(AD18/(AD10*12.5))*100
minus	Royalties ZAR		=(AB20%)*AB18	=(AC20%)*AC18	=(AD20%)*AD18
equals	**Earnings before Tax (EBT)**		**=AB18-AB21**	**=AC18-AC21**	**=AD18-AD21**
minus	Tax	0.28	=AB23*$C$25	=AC23*$C$25	=AD23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=AB23-AB25**	**=AC23-AC25**	**=AD23-AD25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	28	29	30
Tons	Tons milled ('000)		2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((AE2*1000)*AE3)	=((AF2*1000)*AF3)	=((AG2*1000)*AG3)
Oz Content	content 4E oz		=AE5/31.10348	=AF5/31.10348	=AG5/31.10348
Recovery	4E oz recovered	0.92	=AE6*0.92	=AF6*0.92	=AG6*0.92
Basket Price	Basket price (ZAR/4E oz)		=AD8	=AE8	=AF8
Revenue	**Revenue (ZAR)**		**=AE7*AE8**	**=AF7*AF8**	**=AG7*AG8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(AE2*1000)*AE12	=(AF2*1000)*AF12	=(AG2*1000)*AG12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(AE2*1000)*AE14	=(AF2*1000)*AF14	=(AG2*1000)*AG14
Equals	**Operating free cash flow**		**=AE10-AE13-AE15**	**=AF10-AF13-AF15**	**=AG10-AG13-AG15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=AE16-(AE17*1000)**	**=AF16-(AF17*1000)**	**=AG16-(AG17*1000)**
	Royalties %		=0.5+(AE18/(AE10*12.5))*100	=0.5+(AF18/(AF10*12.5))*100	=0.5+(AG18/(AG10*12.5))*100
minus	Royalties ZAR		=(AE20%)*AE18	=(AF20%)*AF18	=(AG20%)*AG18
equals	**Earnings before Tax (EBT)**		**=AE18-AE21**	**=AF18-AF21**	**=AG18-AG21**
minus	Tax	0.28	=AE23*$C$25	=AF23*$C$25	=AG23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=AE23-AE25**	**=AF23-AF25**	**=AG23-AG25**

Table A1. Formulas used for MR discounted cash flow.

Note	Year	MR	31	32	33
Tons	Tons milled ('000)		2100	1750	1099
	Grade (4E g/t)		5.02	5.02	5.02
	content 4E g/t		=((AH2*1000)*AH3)	=((AI2*1000)*AI3)	=((AJ2*1000)*AJ3)
Oz Content	content 4E oz		=AH5/31.10348	=AI5/31.10348	=AJ5/31.10348
Recovery	4E oz recovered	0.92	=AH6*0.92	=AI6*0.92	=AJ6*0.92
Basket Price	Basket price (ZAR/4E oz)		=AG8	=AH8	=AI8
Revenue	**Revenue (ZAR)**		**=AH7*AH8**	**=AI7*AI8**	**=AJ7*AJ8**
	Cost per ton (ZAR/t)		685	1299	1299
Minus cost3	Operating Cost (ZAR)		=(AH2*1000)*AH12	=(AI2*1000)*AI12	=(AJ2*1000)*AJ12
	SIB %	0.12	=$C$14*345	=$C$14*682	=$C$14*682
Minus cost4	SIB Capital		=(AH2*1000)*AH14	=(AI2*1000)*AI14	=(AJ2*1000)*AJ14
Equals	**Operating free cash flow**		**=AH10-AH13-AH15**	**=AI10-AI13-AI15**	**=AJ10-AJ13-AJ15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=AH16-(AH17*1000)**	**=AI16-(AI17*1000)**	**=AJ16-(AJ17*1000)**
	Royalties %		=0.5+(AH18/(AH10*12.5))*10(	=0.5+(AI18/(AI10*12.5))*10(	=0.5+(AJ18/(AJ10*12.5))*10(
minus	Royalties ZAR		=(AH20%)*AH18	=(AI20%)*AI18	=(AJ20%)*AJ18
equals	**Earnings before Tax (EBT)**		**=AH18-AH21**	**=AI18-AI21**	**=AJ18-AJ21**
minus	Tax	0.28	=AH23*$C$25	=AI23*$C$25	=AJ23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=AH23-AH25**	**=AI23-AI25**	**=AJ23-AJ25**

Table A2. MR discounted cash flow.

Note	Year	MR	1	2	3	4
Tons	Tons milled ('000)		0	5	10	60
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		0.00	25,100.00	50,200.00	301,200.00
Oz Content	content 4E oz		0.00	806.98	1,613.97	9,683.80
Recovery	4E oz recovered	0.92	0	742.4249634	1484.849927	8909.099561
Basket Price	Basket price (ZAR/4E oz)		9,254.41	9,854.81	9,594.39	9,753.51
Revenue	**Revenue (ZAR)**		**0.00**	**7,316,454.80**	**14,246,236.42**	**86,895,018.38**
	Cost per ton (ZAR/t)		1,299.00	1,299.00	1,299.00	1,299.00
Minus cost3	Operating Cost (ZAR)		0.00	6,495,000.00	12,990,000.00	77,940,000.00
	SIB %	0.12				
Minus cost4	SIB Capital					
Equals	**Operating free cash flow**		**0.00**	**821,454.80**	**1,256,236.42**	**8,955,018.38**
Minus cost5	Project Capital (R'000)		2,500,000.00	2,400,000.00	1,500,000.00	530,000.00
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**-2,500,000,000.00**	**-2,399,178,545.20**	**-1,498,743,763.58**	**-521,044,981.62**
	Royalties %					
minus	Royalties ZAR					
equals	**Earnings before Tax (EBT)**		**-2,500,000,000.00**	**-2,399,178,545.20**	**-1,498,743,763.58**	**-521,044,981.62**
minus	Tax	0.28				
equals	**Nett Cash Flow (ZAR)**		**-2,500,000,000.00**	**-2,399,178,545.20**	**-1,498,743,763.58**	**-521,044,981.62**

NPV (12.0%)	12.00%	R -1,664,541,443.47
IRR		9%
Payback Period		19 yrs 3 months

Table A2. MR discounted cash flow.

Note	Year	MR	5	6	7	8
Tons	Tons milled ('000)		106	600	1200	1720
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		532,120.00	3,012,000.00	6,024,000.00	8,634,400.00
Oz Content	content 4E oz		17,108.05	96,838.04	193,676.08	277,602.38
Recovery	4E oz recovered	0.92	15739.40922	89090.99561	178181.9912	255394.1874
Basket Price	Basket price (ZAR/4E oz)		9,941.62	10,126.67	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**156,475,275.89**	**902,195,335.20**	**1,740,172,989.85**	**2,494,247,952.12**
	Cost per ton (ZAR/t)		1,299.00	1,299.00	1,299.00	1,299.00
Minus cost3	Operating Cost (ZAR)		137,694,000.00	779,400,000.00	1,558,800,000.00	2,234,280,000.00
	SIB %	0.12			81.84	81.84
Minus cost4	SIB Capital				98,208,000.00	140,764,800.00
Equals	**Operating free cash flow**		**18,781,275.89**	**122,795,335.20**	**83,164,989.85**	**119,203,152.12**
Minus cost5	Project Capital (R'000)		110,000.00	60,000.00		
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**-91,218,724.11**	**62,795,335.20**	**83,164,989.85**	**119,203,152.12**
	Royalties %			1.06	0.88	0.88
minus	Royalties ZAR			663,635.24	733,789.45	1,051,764.88
equals	**Earnings before Tax (EBT)**		**-91,218,724.11**	**62,131,699.95**	**82,431,200.40**	**118,151,387.24**
minus	Tax	0.28		17,396,875.99	23,080,736.11	33,082,388.43
equals	**Nett Cash Flow (ZAR)**		**-91,218,724.11**	**44,734,823.97**	**59,350,464.29**	**85,068,998.81**

Table A2. MR discounted cash flow.

Note	Year	MR	9	10	11	12
Tons	Tons milled ('000)		2180	2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		10,943,600.00	12,650,400.00	12,650,400.00	12,650,400.00
Oz Content	content 4E oz		351,844.87	406,719.76	406,719.76	406,719.76
Recovery	4E oz recovered	0.92	323697.284	374182.1815	374182.1815	374182.1815
Basket Price	Basket price (ZAR/4E oz)		9,766.27	9,766.27	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**3,161,314,264.90**	**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		1,493,300,000.00	1,726,200,000.00	1,726,200,000.00	1,726,200,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		90,252,000.00	104,328,000.00	104,328,000.00	104,328,000.00
Equals	**Operating free cash flow**		**1,577,762,264.90**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
Minus cost5	Project Capital (R'000)					
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**1,577,762,264.90**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
	Royalties %		4.49	4.49	4.49	4.49
minus	Royalties ZAR		70,883,719.29	81,938,978.26	81,938,978.26	81,938,978.26
equals	**Earnings before Tax (EBT)**		**1,506,878,545.61**	**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**
minus	Tax	0.28	421,925,992.77	487,730,964.12	487,730,964.12	487,730,964.12
equals	**Nett Cash Flow (ZAR)**		**1,084,952,552.84**	**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**

Table A2. MR discounted cash flow.

Note	Year	MR	13	14	15	16
Tons	Tons milled ('000)		2520	2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		12,650,400.00	12,650,400.00	12,650,400.00	12,650,400.00
Oz Content	content 4E oz		406,719.76	406,719.76	406,719.76	406,719.76
Recovery	4E oz recovered	0.92	374182.1815	374182.1815	374182.1815	374182.1815
Basket Price	Basket price (ZAR/4E oz)		9,766.27	9,766.27	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		1,726,200,000.00	1,726,200,000.00	1,726,200,000.00	1,726,200,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		104,328,000.00	104,328,000.00	104,328,000.00	104,328,000.00
Equals	**Operating free cash flow**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
Minus cost5	Project Capital (R'000)					
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
	Royalties %		4.49	4.49	4.49	4.49
minus	Royalties ZAR		81,938,978.26	81,938,978.26	81,938,978.26	81,938,978.26
equals	**Earnings before Tax (EBT)**		**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**
minus	Tax	0.28	487,730,964.12	487,730,964.12	487,730,964.12	487,730,964.12
equals	**Nett Cash Flow (ZAR)**		**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**

Table A2. MR discounted cash flow.

Note	Year	MR	17	18	19	20
Tons	Tons milled ('000)		2520	2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		12,650,400.00	12,650,400.00	12,650,400.00	12,650,400.00
Oz Content	content 4E oz		406,719.76	406,719.76	406,719.76	406,719.76
Recovery	4E oz recovered	0.92	374182.1815	374182.1815	374182.1815	374182.1815
Basket Price	Basket price (ZAR/4E oz)		9,766.27	9,766.27	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		1,726,200,000.00	1,726,200,000.00	1,726,200,000.00	1,726,200,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		104,328,000.00	104,328,000.00	104,328,000.00	104,328,000.00
Equals	**Operating free cash flow**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
Minus cost5	Project Capital (R'000)					
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
	Royalties %		4.49	4.49	4.49	4.49
minus	Royalties ZAR		81,938,978.26	81,938,978.26	81,938,978.26	81,938,978.26
equals	**Earnings before Tax (EBT)**		**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**
minus	Tax	0.28	487,730,964.12	487,730,964.12	487,730,964.12	487,730,964.12
equals	**Nett Cash Flow (ZAR)**		**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**

Table A2. MR discounted cash flow.

Note	Year	MR	21	22	23	24
Tons	Tons milled ('000)		2520	2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		12,650,400.00	12,650,400.00	12,650,400.00	12,650,400.00
Oz Content	content 4E oz		406,719.76	406,719.76	406,719.76	406,719.76
Recovery	4E oz recovered	0.92	374182.1815	374182.1815	374182.1815	374182.1815
Basket Price	Basket price (ZAR/4E oz)		9,766.27	9,766.27	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		1,726,200,000.00	1,726,200,000.00	1,726,200,000.00	1,726,200,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		104,328,000.00	104,328,000.00	104,328,000.00	104,328,000.00
Equals	**Operating free cash flow**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
Minus cost5	Project Capital (R'000)					
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
	Royalties %		4.49	4.49	4.49	4.49
minus	Royalties ZAR		81,938,978.26	81,938,978.26	81,938,978.26	81,938,978.26
equals	**Earnings before Tax (EBT)**		**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**
minus	Tax	0.28	487,730,964.12	487,730,964.12	487,730,964.12	487,730,964.12
equals	**Nett Cash Flow (ZAR)**		**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**

Table A2. MR discounted cash flow.

Note	Year	MR	25	26	27	28
Tons	Tons milled ('000)		2520	2520	2520	2520
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		12,650,400.00	12,650,400.00	12,650,400.00	12,650,400.00
Oz Content	content 4E oz		406,719.76	406,719.76	406,719.76	406,719.76
Recovery	4E oz recovered	0.92	374182.1815	374182.1815	374182.1815	374182.1815
Basket Price	Basket price (ZAR/4E oz)		9,766.27	9,766.27	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**	**3,654,363,278.69**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		1,726,200,000.00	1,726,200,000.00	1,726,200,000.00	1,726,200,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		104,328,000.00	104,328,000.00	104,328,000.00	104,328,000.00
Equals	**Operating free cash flow**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
Minus cost5	Project Capital (R'000)					
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**	**1,823,835,278.69**
	Royalties %		4.49	4.49	4.49	4.49
minus	Royalties ZAR		81,938,978.26	81,938,978.26	81,938,978.26	81,938,978.26
equals	**Earnings before Tax (EBT)**		**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**	**1,741,896,300.43**
minus	Tax	0.28	487,730,964.12	487,730,964.12	487,730,964.12	487,730,964.12
equals	**Nett Cash Flow (ZAR)**		**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**	**1,254,165,336.31**

Table A2. MR discounted cash flow.

Note	Year	MR	29	30	31	32
Tons	Tons milled ('000)		2520	2520	2100	1750
	Grade (4E g/t)		5.02	5.02	5.02	5.02
	content 4E g/t		12,650,400.00	12,650,400.00	10,542,000.00	8,785,000.00
Oz Content	content 4E oz		406,719.76	406,719.76	338,933.14	282,444.28
Recovery	4E oz recovered	0.92	374182.1815	374182.1815	311818.4846	259848.7372
Basket Price	Basket price (ZAR/4E oz)		9,766.27	9,766.27	9,766.27	9,766.27
Revenue	**Revenue (ZAR)**		**3,654,363,278.69**	**3,654,363,278.69**	**3,045,302,732.24**	**2,537,752,276.87**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	1,299.00
Minus cost3	Operating Cost (ZAR)		1,726,200,000.00	1,726,200,000.00	1,438,500,000.00	2,273,250,000.00
	SIB %	0.12	41.40	41.40	41.40	81.84
Minus cost4	SIB Capital		104,328,000.00	104,328,000.00	86,940,000.00	143,220,000.00
Equals	**Operating free cash flow**		**1,823,835,278.69**	**1,823,835,278.69**	**1,519,862,732.24**	**121,282,276.87**
Minus cost5	Project Capital (R'000)					
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**1,823,835,278.69**	**1,823,835,278.69**	**1,519,862,732.24**	**121,282,276.87**
	Royalties %		4.49	4.49	4.49	0.88
minus	Royalties ZAR		81,938,978.26	81,938,978.26	68,282,481.88	1,070,109.62
equals	**Earnings before Tax (EBT)**		**1,741,896,300.43**	**1,741,896,300.43**	**1,451,580,250.36**	**120,212,167.25**
minus	Tax	0.28	487,730,964.12	487,730,964.12	406,442,470.10	33,659,406.83
equals	**Nett Cash Flow (ZAR)**		**1,254,165,336.31**	**1,254,165,336.31**	**1,045,137,780.26**	**86,552,760.42**

Table A2. MR discounted cash flow.

Note	Year	MR	33
Tons	Tons milled ('000)		1099
	Grade (4E g/t)		5.02
	content 4E g/t		5,516,980.00
Oz Content	content 4E oz		177,375.01
Recovery	4E oz recovered	0.92	163185.007
Basket Price	Basket price (ZAR/4E oz)		9,766.27
Revenue	**Revenue (ZAR)**		**1,593,708,429.87**
	Cost per ton (ZAR/t)		1,299.00
Minus cost3	Operating Cost (ZAR)		1,427,601,000.00
	SIB %	0.12	81.84
Minus cost4	SIB Capital		89,942,160.00
Equals	**Operating free cash flow**		**76,165,269.87**
Minus cost5	Project Capital (R'000)		
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**76,165,269.87**
	Royalties %		0.88
minus	Royalties ZAR		672,028.84
equals	**Earnings before Tax (EBT)**		**75,493,241.03**
minus	Tax	0.28	21,138,107.49
equals	**Nett Cash Flow (ZAR)**		**54,355,133.54**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	1	2	3
Tons	Tons milled ('000)		0	5	10
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((D2*1000)*D3)	=((E2*1000)*E3)	=((F2*1000)*F3)
Oz Content	content 4E oz	0.9	=D5/31.10348	=E5/31.10348	=F5/31.10348
Recovery	4E oz recovered		=D6*0.9	=E6*0.9	=F6*0.9
Basket Price	Basket price (ZAR/Pt oz)		8442.33	8984.3	8708.86
Revenue	**Revenue (ZAR)**		**=D7*D8**	**=E7*E8**	**=F7*F8**
	Cost per ton (ZAR/t)		1299	1299	1299
Minus cost3	Operating Cost (ZAR)		=(D2*1000)*D12	=(E2*1000)*E12	=(F2*1000)*F12
	SIB %	0.12			
Minus cost4	SIB Capital				
Equals	**Operating free cash flow**		**=D10-D13-D15**	**=E10-E13-E15**	**=F10-F13-F15**
Minus cost5	Project Capital (R'000)		2500000	2400000	1500000
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=D16-(D17*1000)**	**=E16-(E17*1000)**	**=F16-(F17*1000)**
	Royalties %				
minus	Royalties ZAR				
equals	**Earnings before Tax (EBT)**		**=D18-D21**	**=E18-E21**	**=F18-F21**
minus	Tax	0.28			
equals	**Nett Cash Flow (ZAR)**		**=D23-D25**	**=E23-E25**	**=F23-F25**

NPV (12.0%)	0.12	**=NPV(E29,D27:AI27)**
IRR		=IRR(D27:AI27)
Payback Period		16 yr 9 days

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	4	5	6
Tons	Tons milled ('000)		60	106	600
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((G2*1000)*G3)	=((H2*1000)*H3)	=((I2*1000)*I3)
Oz Content	content 4E oz	0.9	=G5/31.10348	=H5/31.10348	=I5/31.10348
Recovery	4E oz recovered		=G6*0.9	=H6*0.9	=I6*0.9
Basket Price	Basket price (ZAR/Pt oz)		8755.66	8890.39	8892.67
Revenue	**Revenue (ZAR)**		**=G7*G8**	**=H7*H8**	**=I7*I8**
	Cost per ton (ZAR/t)		1299	1299	1299
Minus cost3	Operating Cost (ZAR)		=(G2*1000)*G12	=(H2*1000)*H12	=(I2*1000)*I12
	SIB %	0.12			
Minus cost4	SIB Capital				
Equals	**Operating free cash flow**		**=G10-G13-G15**	**=H10-H13-H15**	**=I10-I13-I15**
Minus cost5	Project Capital (R'000)		530000	110000	60000
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=G16-(G17*1000)**	**=H16-(H17*1000)**	**=I16-(I17*1000)**
	Royalties %				=0.5+(I18/(I10*12.5))*100
minus	Royalties ZAR				=(I20%)*I18
equals	**Earnings before Tax (EBT)**		**=G18-G21**	**=H18-H21**	**=I18-I21**
minus	Tax	0.28			=I23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=G23-G25**	**=H23-H25**	**=I23-I25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	7	8	9
Tons	Tons milled ('000)		1200	1720	2307
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((J2*1000)*J3)	=((K2*1000)*K3)	=((L2*1000)*L3)
Oz Content	content 4E oz	0.9	=J5/31.10348	=K5/31.10348	=L5/31.10348
Recovery	4E oz recovered		=J6*0.9	=K6*0.9	=L6*0.9
Basket Price	Basket price (ZAR/Pt oz)		8581.62	=J8	=K8
Revenue	**Revenue (ZAR)**		**=J7*J8**	**=K7*K8**	**=L7*L8**
	Cost per ton (ZAR/t)		1299	1299	685
Minus cost3	Operating Cost (ZAR)		=(J2*1000)*J12	=(K2*1000)*K12	=(L2*1000)*L12
	SIB %	0.12	=$C$14*682	=$C$14*682	=$C$14*345
Minus cost4	SIB Capital		=(J2*1000)*J14	=(K2*1000)*K14	=(L2*1000)*L14
Equals	**Operating free cash flow**		**=J10-J13-J15**	**=K10-K13-K15**	**=L10-L13-L15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=J16-(J17*1000)**	**=K16-(K17*1000)**	**=L16-(L17*1000)**
	Royalties %		=0.5+(J18/(J10*12.5))*100	=0.5+(K18/(K10*12.5))*100	=0.5+(L18/(L10*12.5))*100
minus	Royalties ZAR		=(J20%)*J18	=(K20%)*K18	=(L20%)*L18
equals	**Earnings before Tax (EBT)**		**=J18-J21**	**=K18-K21**	**=L18-L21**
minus	Tax	0.28	=J23*$C$25	=K23*$C$25	=L23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=J23-J25**	**=K23-K25**	**=L23-L25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	10	11	12
Tons	Tons milled ('000)		3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((M2*1000)*M3)	=((N2*1000)*N3)	=((O2*1000)*O3)
Oz Content	content 4E oz	0.9	=M5/31.10348	=N5/31.10348	=O5/31.10348
Recovery	4E oz recovered		=M6*0.9	=N6*0.9	=O6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=L8	=M8	=N8
Revenue	**Revenue (ZAR)**		**=M7*M8**	**=N7*N8**	**=O7*O8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(M2*1000)*M12	=(N2*1000)*N12	=(O2*1000)*O12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(M2*1000)*M14	=(N2*1000)*N14	=(O2*1000)*O14
Equals	**Operating free cash flow**		**=M10-M13-M15**	**=N10-N13-N15**	**=O10-O13-O15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=M16-(M17*1000)**	**=N16-(N17*1000)**	**=O16-(O17*1000)**
	Royalties %		=0.5+(M18/(M10*12.5))*100	=0.5+(N18/(N10*12.5))*100	=0.5+(O18/(O10*12.5))*100
minus	Royalties ZAR		=(M20%)*M18	=(N20%)*N18	=(O20%)*O18
equals	**Earnings before Tax (EBT)**		**=M18-M21**	**=N18-N21**	**=O18-O21**
minus	Tax	0.28	=M23*$C$25	=N23*$C$25	=O23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=M23-M25**	**=N23-N25**	**=O23-O25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	13	14	15
Tons	Tons milled ('000)		3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((P2*1000)*P3)	=((Q2*1000)*Q3)	=((R2*1000)*R3)
Oz Content	content 4E oz	0.9	=P5/31.10348	=Q5/31.10348	=R5/31.10348
Recovery	4E oz recovered		=P6*0.9	=Q6*0.9	=R6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=O8	=P8	=Q8
Revenue	**Revenue (ZAR)**		**=P7*P8**	**=Q7*Q8**	**=R7*R8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(P2*1000)*P12	=(Q2*1000)*Q12	=(R2*1000)*R12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(P2*1000)*P14	=(Q2*1000)*Q14	=(R2*1000)*R14
Equals	**Operating free cash flow**		**=P10-P13-P15**	**=Q10-Q13-Q15**	**=R10-R13-R15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=P16-(P17*1000)**	**=Q16-(Q17*1000)**	**=R16-(R17*1000)**
	Royalties %		=0.5+(P18/(P10*12.5))*100	=0.5+(Q18/(Q10*12.5))*100	=0.5+(R18/(R10*12.5))*100
minus	Royalties ZAR		=(P20%)*P18	=(Q20%)*Q18	=(R20%)*R18
equals	**Earnings before Tax (EBT)**		**=P18-P21**	**=Q18-Q21**	**=R18-R21**
minus	Tax	0.28	=P23*$C$25	=Q23*$C$25	=R23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=P23-P25**	**=Q23-Q25**	**=R23-R25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	16	17	18
Tons	Tons milled ('000)		3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((S2*1000)*S3)	=((T2*1000)*T3)	=((U2*1000)*U3)
Oz Content	content 4E oz	0.9	=S5/31.10348	=T5/31.10348	=U5/31.10348
Recovery	4E oz recovered		=S6*0.9	=T6*0.9	=U6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=R8	=S8	=T8
Revenue	**Revenue (ZAR)**		**=S7*S8**	**=T7*T8**	**=U7*U8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(S2*1000)*S12	=(T2*1000)*T12	=(U2*1000)*U12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(S2*1000)*S14	=(T2*1000)*T14	=(U2*1000)*U14
Equals	**Operating free cash flow**		**=S10-S13-S15**	**=T10-T13-T15**	**=U10-U13-U15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=S16-(S17*1000)**	**=T16-(T17*1000)**	**=U16-(U17*1000)**
	Royalties %		=0.5+(S18/(S10*12.5))*100	=0.5+(T18/(T10*12.5))*100	=0.5+(U18/(U10*12.5))*100
minus	Royalties ZAR		=(S20%)*S18	=(T20%)*T18	=(U20%)*U18
equals	**Earnings before Tax (EBT)**		**=S18-S21**	**=T18-T21**	**=U18-U21**
minus	Tax	0.28	=S23*$C$25	=T23*$C$25	=U23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=S23-S25**	**=T23-T25**	**=U23-U25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	19	20	21
Tons	Tons milled ('000)		3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((V2*1000)*V3)	=((W2*1000)*W3)	=((X2*1000)*X3)
Oz Content	content 4E oz	0.9	=V5/31.10348	=W5/31.10348	=X5/31.10348
Recovery	4E oz recovered		=V6*0.9	=W6*0.9	=X6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=U8	=V8	=W8
Revenue	**Revenue (ZAR)**		**=V7*V8**	**=W7*W8**	**=X7*X8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(V2*1000)*V12	=(W2*1000)*W12	=(X2*1000)*X12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(V2*1000)*V14	=(W2*1000)*W14	=(X2*1000)*X14
Equals	**Operating free cash flow**		**=V10-V13-V15**	**=W10-W13-W15**	**=X10-X13-X15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=V16-(V17*1000)**	**=W16-(W17*1000)**	**=X16-(X17*1000)**
	Royalties %		=0.5+(V18/(V10*12.5))*100	=0.5+(W18/(W10*12.5))*100	=0.5+(X18/(X10*12.5))*100
minus	Royalties ZAR		=(V20%)*V18	=(W20%)*W18	=(X20%)*X18
equals	**Earnings before Tax (EBT)**		**=V18-V21**	**=W18-W21**	**=X18-X21**
minus	Tax	0.28	=V23*$C$25	=W23*$C$25	=X23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=V23-V25**	**=W23-W25**	**=X23-X25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	22	23	24
Tons	Tons milled ('000)		3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((Y2*1000)*Y3)	=((Z2*1000)*Z3)	=((AA2*1000)*AA3)
Oz Content	content 4E oz	0.9	=Y5/31.10348	=Z5/31.10348	=AA5/31.10348
Recovery	4E oz recovered		=Y6*0.9	=Z6*0.9	=AA6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=X8	=Y8	=Z8
Revenue	**Revenue (ZAR)**		**=Y7*Y8**	**=Z7*Z8**	**=AA7*AA8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(Y2*1000)*Y12	=(Z2*1000)*Z12	=(AA2*1000)*AA12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(Y2*1000)*Y14	=(Z2*1000)*Z14	=(AA2*1000)*AA14
Equals	**Operating free cash flow**		**=Y10-Y13-Y15**	**=Z10-Z13-Z15**	**=AA10-AA13-AA15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=Y16-(Y17*1000)**	**=Z16-(Z17*1000)**	**=AA16-(AA17*1000)**
	Royalties %		=0.5+(Y18/(Y10*12.5))*100	=0.5+(Z18/(Z10*12.5))*100	=0.5+(AA18/(AA10*12.5))*100
minus	Royalties ZAR		=(Y20%)*Y18	=(Z20%)*Z18	=(AA20%)*AA18
equals	**Earnings before Tax (EBT)**		**=Y18-Y21**	**=Z18-Z21**	**=AA18-AA21**
minus	Tax	0.28	=Y23*$C$25	=Z23*$C$25	=AA23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=Y23-Y25**	**=Z23-Z25**	**=AA23-AA25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	25	26	27
Tons	Tons milled ('000)		3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((AB2*1000)*AB3)	=((AC2*1000)*AC3)	=((AD2*1000)*AD3)
Oz Content	content 4E oz	0.9	=AB5/31.10348	=AC5/31.10348	=AD5/31.10348
Recovery	4E oz recovered		=AB6*0.9	=AC6*0.9	=AD6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=AA8	=AB8	=AC8
Revenue	**Revenue (ZAR)**		**=AB7*AB8**	**=AC7*AC8**	**=AD7*AD8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(AB2*1000)*AB12	=(AC2*1000)*AC12	=(AD2*1000)*AD12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(AB2*1000)*AB14	=(AC2*1000)*AC14	=(AD2*1000)*AD14
Equals	**Operating free cash flow**		**=AB10-AB13-AB15**	**=AC10-AC13-AC15**	**=AD10-AD13-AD15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=AB16-(AB17*1000)**	**=AC16-(AC17*1000)**	**=AD16-(AD17*1000)**
	Royalties %		=0.5+(AB18/(AB10*12.5))*100	=0.5+(AC18/(AC10*12.5))*100	=0.5+(AD18/(AD10*12.5))*100
minus	Royalties ZAR		=(AB20%)*AB18	=(AC20%)*AC18	=(AD20%)*AD18
equals	**Earnings before Tax (EBT)**		**=AB18-AB21**	**=AC18-AC21**	**=AD18-AD21**
minus	Tax	0.28	=AB23*$C$25	=AC23*$C$25	=AD23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=AB23-AB25**	**=AC23-AC25**	**=AD23-AD25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	28	29	30
Tons	Tons milled ('000)		3006	3006	2500
	Grade (4E g/t)		6.34	6.34	6.34
	content 4E g/t		=((AE2*1000)*AE3)	=((AF2*1000)*AF3)	=((AG2*1000)*AG3)
Oz Content	content 4E oz	0.9	=AE5/31.10348	=AF5/31.10348	=AG5/31.10348
Recovery	4E oz recovered		=AE6*0.9	=AF6*0.9	=AG6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=AD8	=AE8	=AF8
Revenue	**Revenue (ZAR)**		**=AE7*AE8**	**=AF7*AF8**	**=AG7*AG8**
	Cost per ton (ZAR/t)		685	685	685
Minus cost3	Operating Cost (ZAR)		=(AE2*1000)*AE12	=(AF2*1000)*AF12	=(AG2*1000)*AG12
	SIB %	0.12	=$C$14*345	=$C$14*345	=$C$14*345
Minus cost4	SIB Capital		=(AE2*1000)*AE14	=(AF2*1000)*AF14	=(AG2*1000)*AG14
Equals	**Operating free cash flow**		**=AE10-AE13-AE15**	**=AF10-AF13-AF15**	**=AG10-AG13-AG15**
Minus cost5	Project Capital (R'000)				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=AE16-(AE17*1000)**	**=AF16-(AF17*1000)**	**=AG16-(AG17*1000)**
	Royalties %		=0.5+(AE18/(AE10*12.5))*100	=0.5+(AF18/(AF10*12.5))*100	=0.5+(AG18/(AG10*12.5))*100
minus	Royalties ZAR		=(AE20%)*AE18	=(AF20%)*AF18	=(AG20%)*AG18
equals	**Earnings before Tax (EBT)**		**=AE18-AE21**	**=AF18-AF21**	**=AG18-AG21**
minus	Tax	0.28	=AE23*$C$25	=AF23*$C$25	=AG23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=AE23-AE25**	**=AF23-AF25**	**=AG23-AG25**

Table A3. Formulas used for UG2 discounted cash flow.

Note	Year	UG2	31	32
Tons	Tons milled ('000)		1600	1022
	Grade (4E g/t)		6.34	6.34
	content 4E g/t		=((AH2*1000)*AH3)	=((AI2*1000)*AI3)
Oz Content	content 4E oz	0.9	=AH5/31.10348	=AI5/31.10348
Recovery	4E oz recovered		=AH6*0.9	=AI6*0.9
Basket Price	Basket price (ZAR/Pt oz)		=AG8	=AH8
Revenue	**Revenue (ZAR)**		**=AH7*AH8**	**=AI7*AI8**
	Cost per ton (ZAR/t)		1299	1299
Minus cost3	Operating Cost (ZAR)		=(AH2*1000)*AH12	=(AI2*1000)*AI12
	SIB %	0.12	=$C$14*682	=$C$14*682
Minus cost4	SIB Capital		=(AH2*1000)*AH14	=(AI2*1000)*AI14
Equals	**Operating free cash flow**		**=AH10-AH13-AH15**	**=AI10-AI13-AI15**
Minus cost5	Project Capital (R'000)			
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**=AH16-(AH17*1000)**	**=AI16-(AI17*1000)**
	Royalties %		=0.5+(AH18/(AH10*12.5))*100	=0.5+(AI18/(AI10*12.5))*100
minus	Royalties ZAR		=(AH20%)*AH18	=(AI20%)*AI18
equals	**Earnings before Tax (EBT)**		**=AH18-AH21**	**=AI18-AI21**
minus	Tax	0.28	=AH23*$C$25	=AI23*$C$25
equals	**Nett Cash Flow (ZAR)**		**=AH23-AH25**	**=AI23-AI25**

Table A4. UG2 discounted cash flow.

Note	Year	UG2	1	2	3	4	5
Tons	Tons milled ('000)		0	5	10	60	106
	Grade (4E g/t)		6.34	6.34	6.34	6.34	6.34
	content 4E g/t		0.00	31,700.00	63,400.00	380,400.00	672,040.00
Oz Content	content 4E oz	0.9	0.00	1,019.18	2,038.36	12,230.14	21,606.59
Recovery	4E oz recovered		0	917.26	1,834.52	11,007.13	19,445.93
Basket Price	Basket price (ZAR/Pt oz)		8,442.33	8,984.30	8,708.86	8,755.66	8,890.39
Revenue	**Revenue (ZAR)**		**0.00**	**8,240,942.97**	**15,976,594.90**	**96,374,630.37**	**172,881,960.07**
	Cost per ton (ZAR/t)		1,299.00	1,299.00	1,299.00	1,299.00	1,299.00
Minus cost3	Operating Cost (ZAR)		0	6,495,000.00	12,990,000.00	77,940,000.00	137,694,000.00
	SIB %	0.12					
Minus cost4	SIB Capital						
Equals	**Operating free cash flow**		**0.00**	**1,745,942.97**	**2,986,594.90**	**18,434,630.37**	**35,187,960.07**
Minus cost5	Project Capital (R'000)		2,500,000.00	2,400,000.00	1,500,000.00	530,000.00	110,000.00
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**-2,500,000,000.00**	**-2,398,254,057.03**	**-1,497,013,405.10**	**-511,565,369.63**	**-74,812,039.93**
	Royalties %						
minus	Royalties ZAR						
equals	**Earnings before Tax (EBT)**		**-2,500,000,000.00**	**-2,398,254,057.03**	**-1,497,013,405.10**	**-511,565,369.63**	**-74,812,039.93**
minus	Tax	0.28					
equals	**Nett Cash Flow (ZAR)**		**-2,500,000,000.00**	**-2,398,254,057.03**	**-1,497,013,405.10**	**-511,565,369.63**	**-74,812,039.93**

NPV (12.0%)	12.00%	**R -109,614,208.27**
IRR		12%
Payback Period		16 yr 9 days

Table A4. UG2 discounted cash flow.

Note	Year	UG2	6	7	8	9	10
Tons	Tons milled ('000)		600	1200	1720	2307	3006
	Grade (4E g/t)		6.34	6.34	6.34	6.34	6.34
	content 4E g/t		3,804,000.00	7,608,000.00	10,904,800.00	14,626,380.00	19,058,040.00
Oz Content	content 4E oz	0.9	122,301.43	244,602.85	350,597.43	470,248.99	612,730.15
Recovery	4E oz recovered		110,071.28	220,142.57	315,537.68	423,224.09	551,457.14
Basket Price	Basket price (ZAR/Pt oz)		8,892.67	8,581.62	8,581.62	8,581.62	8,581.62
Revenue	**Revenue (ZAR)**		**978,827,335.49**	**1,889,179,875.18**	**2,707,824,487.76**	**3,631,948,310.03**	**4,732,395,587.32**
	Cost per ton (ZAR/t)		1,299.00	1,299.00	1,299.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		779,400,000.00	1,558,800,000.00	2,234,280,000.00	1,580,295,000.00	2,059,110,000.00
	SIB %	0.12		81.84	81.84	41.4	41.40
Minus cost4	SIB Capital			98,208,000.00	140,764,800.00	95,509,800.00	124,448,400.00
Equals	**Operating free cash flow**		**199,427,335.49**	**232,171,875.18**	**332,779,687.76**	**1,956,143,510.03**	**2,548,837,187.32**
Minus cost5	Project Capital (R'000)		60,000.00				
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**139,427,335.49**	**232,171,875.18**	**332,779,687.76**	**1,956,143,510.03**	**2,548,837,187.32**
	Royalties %		1.64	1.48	1.48	4.81	4.81
minus	Royalties ZAR		2,285,975.17	3,443,491.34	4,935,670.92	94,066,001.48	122,567,143.68
equals	**Earnings before Tax (EBT)**		**137,141,360.32**	**228,728,383.84**	**327,844,016.83**	**1,862,077,508.54**	**2,426,270,043.64**
minus	Tax	0.28	38,399,580.89	64,043,947.47	91,796,324.71	521,381,702.39	679,355,612.22
equals	**Nett Cash Flow (ZAR)**		**98,741,779.43**	**164,684,436.36**	**236,047,692.12**	**1,340,695,806.15**	**1,746,914,431.42**

Table A4. UG2 discounted cash flow.

Note	Year	UG2	11	12	13	14	15
Tons	Tons milled ('000)		3006	3006	3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34	6.34	6.34
	content 4E g/t		19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00
Oz Content	content 4E oz	0.9	612,730.15	612,730.15	612,730.15	612,730.15	612,730.15
Recovery	4E oz recovered		551,457.14	551,457.14	551,457.14	551,457.14	551,457.14
Basket Price	Basket price (ZAR/Pt oz)		8,581.62	8,581.62	8,581.62	8,581.62	8,581.62
Revenue	**Revenue (ZAR)**		**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00
Equals	**Operating free cash flow**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**
Minus cost5	Project Capital (R'000)						
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**
	Royalties %		4.81	4.81	4.81	4.81	4.81
minus	Royalties ZAR		122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68
equals	**Earnings before Tax (EBT)**		**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**
minus	Tax	0.28	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22
equals	**Nett Cash Flow (ZAR)**		**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**

Table A4. UG2 discounted cash flow.

Note	Year	UG2	16	17	18	19	20
Tons	Tons milled ('000)		3006	3006	3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34	6.34	6.34
	content 4E g/t		19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00
Oz Content	content 4E oz	0.9	612,730.15	612,730.15	612,730.15	612,730.15	612,730.15
Recovery	4E oz recovered		551,457.14	551,457.14	551,457.14	551,457.14	551,457.14
Basket Price	Basket price (ZAR/Pt oz)		8,581.62	8,581.62	8,581.62	8,581.62	8,581.62
Revenue	**Revenue (ZAR)**		**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00
Equals	**Operating free cash flow**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**
Minus cost5	Project Capital (R'000)						
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**
	Royalties %		4.81	4.81	4.81	4.81	4.81
minus	Royalties ZAR		122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68
equals	**Earnings before Tax (EBT)**		**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**
minus	Tax	0.28	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22
equals	**Nett Cash Flow (ZAR)**		**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**

Table A4. UG2 discounted cash flow.

Note	Year	UG2	21	22	23	24	25
Tons	Tons milled ('000)		3006	3006	3006	3006	3006
	Grade (4E g/t)		6.34	6.34	6.34	6.34	6.34
	content 4E g/t		19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00
Oz Content	content 4E oz	0.9	612,730.15	612,730.15	612,730.15	612,730.15	612,730.15
Recovery	4E oz recovered		551,457.14	551,457.14	551,457.14	551,457.14	551,457.14
Basket Price	Basket price (ZAR/Pt oz)		8,581.62	8,581.62	8,581.62	8,581.62	8,581.62
Revenue	**Revenue (ZAR)**		**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00
Equals	**Operating free cash flow**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**
Minus cost5	Project Capital (R'000)						
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**
	Royalties %		4.81	4.81	4.81	4.81	4.81
minus	Royalties ZAR		122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68
equals	**Earnings before Tax (EBT)**		**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**
minus	Tax	0.28	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22
equals	**Nett Cash Flow (ZAR)**		**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**

Table A4. UG2 discounted cash flow.

Note	Year	UG2	26	27	28	29	30
Tons	Tons milled ('000)		3006	3006	3006	3006	2500
	Grade (4E g/t)		6.34	6.34	6.34	6.34	6.34
	content 4E g/t		19,058,040.00	19,058,040.00	19,058,040.00	19,058,040.00	15,850,000.00
Oz Content	content 4E oz	0.9	612,730.15	612,730.15	612,730.15	612,730.15	509,589.28
Recovery	4E oz recovered		551,457.14	551,457.14	551,457.14	551,457.14	458,630.35
Basket Price	Basket price (ZAR/Pt oz)		8,581.62	8,581.62	8,581.62	8,581.62	8,581.62
Revenue	**Revenue (ZAR)**		**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**4,732,395,587.32**	**3,935,791,406.62**
	Cost per ton (ZAR/t)		685.00	685.00	685.00	685.00	685.00
Minus cost3	Operating Cost (ZAR)		2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	2,059,110,000.00	1,712,500,000.00
	SIB %	0.12	41.40	41.40	41.40	41.40	41.40
Minus cost4	SIB Capital		124,448,400.00	124,448,400.00	124,448,400.00	124,448,400.00	103,500,000.00
Equals	**Operating free cash flow**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,119,791,406.62**
Minus cost5	Project Capital (R'000)						
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,548,837,187.32**	**2,119,791,406.62**
	Royalties %		4.81	4.81	4.81	4.81	4.81
minus	Royalties ZAR		122,567,143.68	122,567,143.68	122,567,143.68	122,567,143.68	101,935,415.57
equals	**Earnings before Tax (EBT)**		**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,426,270,043.64**	**2,017,855,991.05**
minus	Tax	0.28	679,355,612.22	679,355,612.22	679,355,612.22	679,355,612.22	564,999,677.50
equals	**Nett Cash Flow (ZAR)**		**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,746,914,431.42**	**1,452,856,313.56**

Table A4. UG2 discounted cash flow.

Note	Year	UG2	31	32
Tons	Tons milled ('000)		1600	1022
	Grade (4E g/t)		6.34	6.34
	content 4E g/t		10,144,000.00	6,479,480.00
Oz Content	content 4E oz	0.9	326,137.14	208,320.10
Recovery	4E oz recovered		293,523.43	187,488.09
Basket Price	Basket price (ZAR/Pt oz)		8,581.62	8,581.62
Revenue	**Revenue (ZAR)**		**2,518,906,500.24**	**1,608,951,527.03**
	Cost per ton (ZAR/t)		1,299.00	1,299.00
Minus cost3	Operating Cost (ZAR)		2,078,400,000.00	1,327,578,000.00
	SIB %	0.12	81.84	81.84
Minus cost4	SIB Capital		130,944,000.00	83,640,480.00
Equals	**Operating free cash flow**		**309,562,500.24**	**197,733,047.03**
Minus cost5	Project Capital (R'000)			
Equals	**CASH FLOW BEFORE TAX (EBIT)**		**309,562,500.24**	**197,733,047.03**
	Royalties %		1.48	1.48
minus	Royalties ZAR		4,591,321.79	2,932,706.79
equals	**Earnings before Tax (EBT)**		**304,971,178.45**	**194,800,340.23**
minus	Tax	0.28	85,391,929.97	54,544,095.27
equals	**Nett Cash Flow (ZAR)**		**219,579,248.48**	**140,256,244.97**

Appendix B

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)						
Year			1	2	3	4	5
	NPV	IRR					
Base case	5.02						
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62	-91,218,724.11
	R -1,664,541,443.47	9%					
Sensitivity 1	6.02						
Cash flow			-2,500,000,000.00	-2,397,721,084.08	-1,495,905,867.88	-503,735,217.00	-60,048,350.42
	R 193,102,568.75	12%					
Sensitivity 2	5.5						
Cash flow			-2,500,000,000.00	-2,398,478,963.86	-1,497,381,573.65	-512,736,294.60	-76,256,944.74
	R -770,732,665.59	11%					
Sensitivity 3	5.95						
Cash flow			-2,500,000,000.00	-2,397,823,106.36	-1,496,104,520.58	-504,946,900.52	-62,230,276.58
	R 63,583,340.09	12%					
Sensitivity 4	4.02						
Cash flow			-2,500,000,000.00	-2,400,636,006.32	-1,501,581,659.28	-538,354,746.24	-122,389,097.79
	R -3,640,340,928.96	5%					
Sensitivity 5	4.14						
Cash flow			-2,500,000,000.00	-2,400,461,110.98	-1,501,241,111.80	-536,277,574.48	-118,648,652.95
	R -3,398,000,156.19	5%					
Sensitivity 6	3.5						
Cash flow			-2,500,000,000.00	-2,401,393,886.10	-1,503,057,365.05	-547,355,823.84	-138,597,692.11
	R -4,695,125,217.32	1%					
Sensitivity 7	3.02						
Cash flow							
	R -5,677,948,333.87	nothing					

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)						
Year			6	7	8	9	10
	NPV	IRR					
Base case	5.02						
Cash flow			44,734,823.97	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	6.02						
Cash flow			170,606,938.32	302,818,896.82	434,040,418.78	1,507,417,490.82	1,742,519,301.32
	R 193,102,568.75	12%					
Sensitivity 2	5.5						
Cash flow			105,492,552.48	176,900,428.79	253,557,281.26	1,288,080,156.16	1,488,973,391.53
	R -770,732,665.59	11%					
Sensitivity 3	5.95						
Cash flow			161,877,647.77	285,941,282.49	409,849,171.57	1,477,927,985.10	1,708,430,514.89
	R 63,583,340.09	12%					
Sensitivity 4	4.02						
Cash flow			-116,924,851.10	-263,483,016.09	-377,658,989.74	658,711,658.88	761,446,504.76
	R -3,640,340,928.96	5%					
Sensitivity 5	4.14						
Cash flow			-95,358,428.74	-221,885,255.38	-318,035,532.71	710,150,472.02	820,907,885.08
	R -3,398,000,156.19	5%					
Sensitivity 6	3.5						
Cash flow			-210,379,347.97	-443,739,979.19	-636,027,303.50	434,499,710.60	502,265,720.51
	R -4,695,125,217.32	1%					
Sensitivity 7	3.02						
Cash flow							
	R -5,677,948,333.87	nothing					

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)						
Year			11	12	13	14	15
	NPV	IRR					
Base case	5.02						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	6.02						
Cash flow			1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32
	R 193,102,568.75	12%					
Sensitivity 2	5.5						
Cash flow			1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53
	R -770,732,665.59	11%					
Sensitivity 3	5.95						
Cash flow			1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89
	R 63,583,340.09	12%					
Sensitivity 4	4.02						
Cash flow			761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76
	R -3,640,340,928.96	5%					
Sensitivity 5	4.14						
Cash flow			820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08
	R -3,398,000,156.19	5%					
Sensitivity 6	3.5						
Cash flow			502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51
	R -4,695,125,217.32	1%					
Sensitivity 7	3.02						
Cash flow							
	R -5,677,948,333.87	nothing					

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)						
Year			16	17	18	19	20
	NPV	IRR					
Base case	5.02						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	6.02						
Cash flow			1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32
	R 193,102,568.75	12%					
Sensitivity 2	5.5						
Cash flow			1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53
	R -770,732,665.59	11%					
Sensitivity 3	5.95						
Cash flow			1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89
	R 63,583,340.09	12%					
Sensitivity 4	4.02						
Cash flow			761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76
	R -3,640,340,928.96	5%					
Sensitivity 5	4.14						
Cash flow			820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08
	R -3,398,000,156.19	5%					
Sensitivity 6	3.5						
Cash flow			502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51
	R -4,695,125,217.32	1%					
Sensitivity 7	3.02						
Cash flow							
	R -5,677,948,333.87	nothing					

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)						
Year			21	22	23	24	25
	NPV	IRR					
Base case	5.02						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	6.02						
Cash flow			1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32
	R 193,102,568.75	12%					
Sensitivity 2	5.5						
Cash flow			1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53
	R -770,732,665.59	11%					
Sensitivity 3	5.95						
Cash flow			1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89
	R 63,583,340.09	12%					
Sensitivity 4	4.02						
Cash flow			761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76
	R -3,640,340,928.96	5%					
Sensitivity 5	4.14						
Cash flow			820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08
	R -3,398,000,156.19	5%					
Sensitivity 6	3.5						
Cash flow			502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51
	R -4,695,125,217.32	1%					
Sensitivity 7	3.02						
Cash flow							
	R -5,677,948,333.87	nothing					

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)						
Year			26	27	28	29	30
	NPV	IRR					
Base case	5.02						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	6.02						
Cash flow			1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32	1,742,519,301.32
	R 193,102,568.75	12%					
Sensitivity 2	5.5						
Cash flow			1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53	1,488,973,391.53
	R -770,732,665.59	11%					
Sensitivity 3	5.95						
Cash flow			1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89	1,708,430,514.89
	R 63,583,340.09	12%					
Sensitivity 4	4.02						
Cash flow			761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76	761,446,504.76
	R -3,640,340,928.96	5%					
Sensitivity 5	4.14						
Cash flow			820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08	820,907,885.08
	R -3,398,000,156.19	5%					
Sensitivity 6	3.5						
Cash flow			502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51	502,265,720.51
	R -4,695,125,217.32	1%					
Sensitivity 7	3.02						
Cash flow							
	R -5,677,948,333.87	nothing					

Table B1.1. MR grade sensitivity analysis data.

MR	GRADE (g/t)				
Year			31	32	33
	NPV	IRR			
Base case	5.02				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -1,664,541,443.47	9%			
Sensitivity 1	6.02				
Cash flow			1,452,099,417.77	441,610,891.20	277,331,639.67
	R 193,102,568.75	12%			
Sensitivity 2	5.5				
Cash flow			1,240,811,159.61	257,979,791.98	162,011,309.36
	R -770,732,665.59	11%			
Sensitivity 3	5.95				
Cash flow			1,423,692,095.74	416,997,703.63	261,874,557.88
	R 63,583,340.09	12%			
Sensitivity 4	4.02				
Cash flow			634,538,753.97	-384,246,065.14	-241,306,528.91
	R -3,640,340,928.96	5%			
Sensitivity 5	4.14				
Cash flow			684,089,904.23	-323,582,664.10	-203,209,913.05
	R -3,398,000,156.19	5%			
Sensitivity 6	3.5				
Cash flow			418,554,767.10	-647,120,802.98	-406,391,864.27
	R -4,695,125,217.32	1%			
Sensitivity 7	3.02				
Cash flow					
	R -5,677,948,333.87	nothing			

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %						
Year			1	2	3	4	5
	NPV	IRR					
Base case	0.92						
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62	-91,218,724.11
	R -1,664,541,443.47	9%					
Sensitivity 1	0.95						
Cash flow			-2,500,000,000.00	-2,398,939,965.15	-1,498,279,212.40	-518,211,448.41	-86,116,269.46
	R -1,359,208,040.84	10%					
Sensitivity 2	0.99						
Cash flow			-2,500,000,000.00	-2,398,621,858.42	-1,497,659,810.81	-514,433,404.13	-79,312,996.59
	R -952,941,837.84	10%					
Sensitivity 3	0.9						
Cash flow			-2,500,000,000.00	-2,399,337,598.56	-1,499,053,464.38	-522,934,003.76	-94,620,360.54
	R -1,868,429,009.72	9%					
Sensitivity 4	0.85						
Cash flow			-2,500,000,000.00	-2,399,735,231.98	-1,499,827,716.35	-527,656,559.10	-103,124,451.62
	R -2,379,446,492.33	8%					
Sensitivity 5	0.8						
Cash flow			-2,500,000,000.00	-2,400,132,865.39	-1,500,601,968.33	-532,379,114.45	-111,628,542.70
	R -2,892,597,334.99	7%					

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %						
Year			6	7	8	9	10
	NPV	IRR					
Base case	0.92						
Cash flow			44,734,823.97	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	0.95						
Cash flow			65,536,886.46	99,603,886.80	142,765,571.08	1,154,308,666.65	1,334,338,458.70
	R -1,359,208,040.84	10%					
Sensitivity 2	0.99						
Cash flow			93,139,045.59	153,004,547.53	219,306,518.13	1,246,647,460.09	1,441,078,715.33
	R -952,941,837.84	10%					
Sensitivity 3	0.9						
Cash flow			30,814,169.22	32,408,554.20	46,452,261.02	1,038,661,705.24	1,200,654,815.23
	R -1,868,429,009.72	9%					
Sensitivity 4	0.85						
Cash flow			-4,193,277.62	-35,362,022.36	-50,685,565.38	922,725,547.84	1,066,636,871.81
	R -2,379,446,492.33	8%					
Sensitivity 5	0.8						
Cash flow			-39,538,840.88	-103,815,701.10	-148,802,504.91	806,445,970.20	932,221,947.20
	R -2,892,597,334.99	7%					

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %						
Year			11	12	13	14	15
	NPV	IRR					
Base case	0.92						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	0.95						
Cash flow			1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70
	R -1,359,208,040.84	10%					
Sensitivity 2	0.99						
Cash flow			1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33
	R -952,941,837.84	10%					
Sensitivity 3	0.9						
Cash flow			1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23
	R -1,868,429,009.72	9%					
Sensitivity 4	0.85						
Cash flow			1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81
	R -2,379,446,492.33	8%					
Sensitivity 5	0.8						
Cash flow			932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20
	R -2,892,597,334.99	7%					

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %						
Year			16	17	18	19	20
	NPV	IRR					
Base case	0.92						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	0.95						
Cash flow			1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70
	R -1,359,208,040.84	10%					
Sensitivity 2	0.99						
Cash flow			1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33
	R -952,941,837.84	10%					
Sensitivity 3	0.9						
Cash flow			1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23
	R -1,868,429,009.72	9%					
Sensitivity 4	0.85						
Cash flow			1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81
	R -2,379,446,492.33	8%					
Sensitivity 5	0.8						
Cash flow			932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20
	R -2,892,597,334.99	7%					

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %						
Year			21	22	23	24	25
	NPV	IRR					
Base case	0.92						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	0.95						
Cash flow			1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70
	R -1,359,208,040.84	10%					
Sensitivity 2	0.99						
Cash flow			1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33
	R -952,941,837.84	10%					
Sensitivity 3	0.9						
Cash flow			1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23
	R -1,868,429,009.72	9%					
Sensitivity 4	0.85						
Cash flow			1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81
	R -2,379,446,492.33	8%					
Sensitivity 5	0.8						
Cash flow			932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20
	R -2,892,597,334.99	7%					

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %						
Year			26	27	28	29	30
	NPV	IRR					
Base case	0.92						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	0.95						
Cash flow			1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70	1,334,338,458.70
	R -1,359,208,040.84	10%					
Sensitivity 2	0.99						
Cash flow			1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33	1,441,078,715.33
	R -952,941,837.84	10%					
Sensitivity 3	0.9						
Cash flow			1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23	1,200,654,815.23
	R -1,868,429,009.72	9%					
Sensitivity 4	0.85						
Cash flow			1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81	1,066,636,871.81
	R -2,379,446,492.33	8%					
Sensitivity 5	0.8						
Cash flow			932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20	932,221,947.20
	R -2,892,597,334.99	7%					

Table B1. 2. MR recovery sensitivity analysis data.

MR	RECOVERY %				
Year			31	32	33
	NPV	IRR			
Base case	0.92				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -1,664,541,443.47	9%			
Sensitivity 1	0.95				
Cash flow			1,111,948,715.58	145,255,668.25	91,220,559.66
	R -1,359,208,040.84	10%			
Sensitivity 2	0.99				
Cash flow			1,200,898,929.44	223,131,631.81	140,126,664.78
	R -952,941,837.84	10%			
Sensitivity 3	0.9				
Cash flow			1,000,545,679.36	47,262,474.87	29,680,834.22
	R -1,868,429,009.72	9%			
Sensitivity 4	0.85				
Cash flow			888,864,059.84	-51,569,615.93	-32,385,718.81
	R -2,379,446,492.33	8%			
Sensitivity 5	0.8				
Cash flow			776,851,622.67	-151,397,897.44	-95,077,879.59
	R -2,892,597,334.99	7%			

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE						
Year			1	2	3	4	5
	NPV	IRR					
Base case							
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62	-91,218,724.11
	R -1,664,541,443.47	9%					
Sensitivity 1	9766.27	same price					
Cash flow			-2,500,000,000.00	-2,399,244,277.35	-1,498,488,554.71	-520,931,328.23	-93,978,679.88
	R -1,677,461,637.39	9%					
Sensitivity 2	10017.9	17-May-11					
Cash flow			-2,500,000,000.00	-2,399,057,460.96	-1,498,114,921.92	-518,689,531.51	-90,018,172.34
	R -1,436,450,457.43	10%					
Sensitivity 3	10889.31	18-Feb-11					
Cash flow			-2,500,000,000.00	-2,398,410,504.42	-1,496,821,008.84	-510,926,053.06	-76,302,693.75
	R -604,367,856.45	11%					
Sensitivity 4	10973.33	20-Aug-11					
Cash flow			-2,500,000,000.00	-2,398,348,125.88	-1,496,696,251.75	-510,177,510.52	-74,980,268.58
	R -524,326,182.51	11%					
Sensitivity 5	7588.51	2009					
Cash flow			-2,500,000,000.00	-2,400,861,100.74	-1,501,722,201.48	-540,333,208.89	-128,255,335.71
	R -3,895,524,368.13	4%					
Sensitivity 6	8678.18	2010					
Cash flow			-2,500,000,000.00	-2,400,052,102.53	-1,500,104,205.06	-530,625,230.38	-111,104,573.66
	R -2,767,543,137.17	7%					

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE						
Year			6	7	8	9	10
	NPV	IRR					
Base case							
Cash flow			44,734,823.97	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	9766.27	same price					
Cash flow			21,921,625.39	59,350,781.02	85,069,452.79	1,084,953,097.66	1,254,165,966.11
	R -1,677,461,637.39	9%					
Sensitivity 2	10017.9	17-May-11					
Cash flow			37,862,193.62	91,171,478.97	130,679,119.86	1,139,761,279.17	1,317,522,212.61
	R -1,436,450,457.43	10%					
Sensitivity 3	10889.31	18-Feb-11					
Cash flow			92,646,906.11	200,553,187.51	287,459,568.76	1,329,155,463.27	1,536,454,939.19
	R -604,367,856.45	11%					
Sensitivity 4	10973.33	20-Aug-11					
Cash flow			97,898,614.38	211,040,080.60	302,490,782.20	1,347,386,632.99	1,557,529,502.35
	R -524,326,182.51	11%					
Sensitivity 5	7588.51	2009					
Cash flow			-163,332,088.94	-304,872,177.87	-436,983,454.95	607,429,802.96	702,166,561.22
	R -3,895,524,368.13	4%					
Sensitivity 6	8678.18	2010					
Cash flow			-66,252,303.76	-110,712,607.51	-158,688,070.76	847,187,285.78	979,317,412.92
	R -2,767,543,137.17	7%					

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE						
Year			11	12	13	14	15
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	9766.27	same price					
Cash flow			1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11
	R -1,677,461,637.39	9%					
Sensitivity 2	10017.9	17-May-11					
Cash flow			1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61
	R -1,436,450,457.43	10%					
Sensitivity 3	10889.31	18-Feb-11					
Cash flow			1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19
	R -604,367,856.45	11%					
Sensitivity 4	10973.33	20-Aug-11					
Cash flow			1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35
	R -524,326,182.51	11%					
Sensitivity 5	7588.51	2009					
Cash flow			702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22
	R -3,895,524,368.13	4%					
Sensitivity 6	8678.18	2010					
Cash flow			979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92
	R -2,767,543,137.17	7%					

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE						
Year			16	17	18	19	20
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	9766.27	same price					
Cash flow			1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11
	R -1,677,461,637.39	9%					
Sensitivity 2	10017.9	17-May-11					
Cash flow			1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61
	R -1,436,450,457.43	10%					
Sensitivity 3	10889.31	18-Feb-11					
Cash flow			1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19
	R -604,367,856.45	11%					
Sensitivity 4	10973.33	20-Aug-11					
Cash flow			1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35
	R -524,326,182.51	11%					
Sensitivity 5	7588.51	2009					
Cash flow			702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22
	R -3,895,524,368.13	4%					
Sensitivity 6	8678.18	2010					
Cash flow			979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92
	R -2,767,543,137.17	7%					

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE						
Year			21	22	23	24	25
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	9766.27	same price					
Cash flow			1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11
	R -1,677,461,637.39	9%					
Sensitivity 2	10017.9	17-May-11					
Cash flow			1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61
	R -1,436,450,457.43	10%					
Sensitivity 3	10889.31	18-Feb-11					
Cash flow			1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19
	R -604,367,856.45	11%					
Sensitivity 4	10973.33	20-Aug-11					
Cash flow			1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35
	R -524,326,182.51	11%					
Sensitivity 5	7588.51	2009					
Cash flow			702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22
	R -3,895,524,368.13	4%					
Sensitivity 6	8678.18	2010					
Cash flow			979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92
	R -2,767,543,137.17	7%					

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE						
Year			26	27	28	29	30
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	9766.27	same price					
Cash flow			1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11	1,254,165,966.11
	R -1,677,461,637.39	9%					
Sensitivity 2	10017.9	17-May-11					
Cash flow			1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61	1,317,522,212.61
	R -1,436,450,457.43	10%					
Sensitivity 3	10889.31	18-Feb-11					
Cash flow			1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19	1,536,454,939.19
	R -604,367,856.45	11%					
Sensitivity 4	10973.33	20-Aug-11					
Cash flow			1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35	1,557,529,502.35
	R -524,326,182.51	11%					
Sensitivity 5	7588.51	2009					
Cash flow			702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22	702,166,561.22
	R -3,895,524,368.13	4%					
Sensitivity 6	8678.18	2010					
Cash flow			979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92	979,317,412.92
	R -2,767,543,137.17	7%					

Table B1.3. MR basket price sensitivity analysis data.

MR	BASKET PRICE				
Year			31	32	33
	NPV	IRR			
Base case					
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -1,664,541,443.47	9%			
Sensitivity 1	9766.27	same price			
Cash flow			1,045,138,305.09	86,553,222.32	54,355,423.61
	R -1,677,461,637.39	9%			
Sensitivity 2	10017.9	17-May-11			
Cash flow			1,097,935,177.18	132,958,406.83	83,497,879.49
	R -1,436,450,457.43	10%			
Sensitivity 3	10889.31	18-Feb-11			
Cash flow			1,280,379,115.99	292,473,398.45	183,673,294.23
	R -604,367,856.45	11%			
Sensitivity 4	10973.33	20-Aug-11			
Cash flow			1,297,941,251.96	307,766,784.21	193,277,540.48
	R -524,326,182.51	11%			
Sensitivity 5	7588.51	2009			
Cash flow			585,138,801.02	-444,605,259.40	-279,212,102.90
	R -3,895,524,368.13	4%			
Sensitivity 6	8678.18	2010			
Cash flow			816,097,844.10	-161,455,885.95	-101,394,296.38
	R -2,767,543,137.17	7%			

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL						
Year			1	2	3	4	5
	NPV	IRR					
Base case							
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62	-91,218,724.11
	R -1,664,541,443.47	9%					
Sensitivity 1	-10%						
Cash flow			-2,000,000,000.00	-1,899,178,545.20	-1,298,743,763.58	-691,044,981.62	-371,218,724.11
	R -958,484,759.15	10%					
Sensitivity 2	-20%						
Cash flow			-1,500,000,000.00	-1,199,178,545.20	-948,743,763.58	-821,044,981.62	-781,218,724.11
	R -168,857,029.37	12%					
Sensitivity 3	10%						
Cash flow			-3,000,000,000.00	-2,299,178,545.20	-1,398,743,763.58	-841,044,981.62	-131,218,724.11
	R -2,202,317,000.69	8%					
Sensitivity 4	20%						
Cash flow			-3,500,000,000.00	-2,499,178,545.20	-1,598,743,763.58	-491,044,981.62	-251,218,724.11
	R -2,816,465,537.28	8%					
Sensitivity 5	-30%						
Cash flow			-1,000,000,000.00	-999,178,545.20	-898,743,763.58	-841,044,981.62	-661,218,724.11
	R 457,051,834.11	13%					

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL						
Year			6	7	8	9	10
	NPV	IRR					
Base case							
Cash flow			44,734,823.97	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	-10%						
Cash flow			16,297,402.91	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -958,484,759.15	10%					
Sensitivity 2	-20%						
Cash flow			-277,204,664.80	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -168,857,029.37	12%					
Sensitivity 3	10%						
Cash flow			12,795,335.20	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -2,202,317,000.69	8%					
Sensitivity 4	20%						
Cash flow			-27,204,664.80	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R -2,816,465,537.28	8%					
Sensitivity 5	-30%						
Cash flow			-417,204,664.80	59,350,464.29	85,068,998.81	1,084,952,552.84	1,254,165,336.31
	R 457,051,834.11	13%					

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL						
Year			11	12	13	14	15
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	-10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -958,484,759.15	10%					
Sensitivity 2	-20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -168,857,029.37	12%					
Sensitivity 3	10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,202,317,000.69	8%					
Sensitivity 4	20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,816,465,537.28	8%					
Sensitivity 5	-30%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R 457,051,834.11	13%					

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL						
Year			16	17	18	19	20
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	-10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -958,484,759.15	10%					
Sensitivity 2	-20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -168,857,029.37	12%					
Sensitivity 3	10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,202,317,000.69	8%					
Sensitivity 4	20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,816,465,537.28	8%					
Sensitivity 5	-30%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R 457,051,834.11	13%					

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL						
Year			21	22	23	24	25
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	-10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -958,484,759.15	10%					
Sensitivity 2	-20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -168,857,029.37	12%					
Sensitivity 3	10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,202,317,000.69	8%					
Sensitivity 4	20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,816,465,537.28	8%					
Sensitivity 5	-30%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R 457,051,834.11	13%					

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL						
Year			26	27	28	29	30
	NPV	IRR					
Base case							
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%					
Sensitivity 1	-10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -958,484,759.15	10%					
Sensitivity 2	-20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -168,857,029.37	12%					
Sensitivity 3	10%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,202,317,000.69	8%					
Sensitivity 4	20%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -2,816,465,537.28	8%					
Sensitivity 5	-30%						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R 457,051,834.11	13%					

Table B1.4. MR capital sensitivity analysis data.

MR	CAPITAL				
Year			31	32	33
	NPV	IRR			
Base case					
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -1,664,541,443.47	9%			
Sensitivity 1	-10%				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -958,484,759.15	10%			
Sensitivity 2	-20%				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -168,857,029.37	12%			
Sensitivity 3	10%				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -2,202,317,000.69	8%			
Sensitivity 4	20%				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R -2,816,465,537.28	8%			
Sensitivity 5	-30%				
Cash flow			1,045,137,780.26	86,552,760.42	54,355,133.54
	R 457,051,834.11	13%			

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			1	2	3	4
	NPV	IRR				
Base case						
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			-2,500,000,000.00	-2,397,183,545.20	-1,494,753,763.58	-497,104,981.62
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			-2,500,000,000.00	-2,398,183,545.20	-1,496,753,763.58	-509,104,981.62
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			-2,500,000,000.00	-2,399,178,545.20	-1,498,743,763.58	-521,044,981.62
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			5	6	7	8
	NPV	IRR				
Base case						
Cash flow			-91,218,724.11	44,734,823.97	59,350,464.29	85,068,998.81
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			-48,924,724.11	210,662,350.65	392,138,535.09	562,065,233.63
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			-70,124,724.11	128,405,418.74	227,224,508.26	325,688,461.83
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			-91,218,724.11	44,734,823.97	59,350,464.29	85,068,998.81
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			-91,218,724.11	44,734,823.97	59,350,464.29	85,068,998.81
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			9	10	11	12
	NPV	IRR				
Base case						
Cash flow			1,084,952,552.84	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,084,952,552.84	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,084,952,552.84	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			991,219,802.03	1,145,813,716.11	1,145,813,716.11	1,145,813,716.11
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			1,206,422,127.40	1,394,579,706.90	1,394,579,706.90	1,394,579,706.90
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			13	14	15	16
	NPV	IRR				
Base case						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			1,145,813,716.11	1,145,813,716.11	1,145,813,716.11	1,145,813,716.11
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			1,394,579,706.90	1,394,579,706.90	1,394,579,706.90	1,394,579,706.90
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			17	18	19	20
	NPV	IRR				
Base case						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			1,145,813,716.11	1,145,813,716.11	1,145,813,716.11	1,145,813,716.11
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			1,394,579,706.90	1,394,579,706.90	1,394,579,706.90	1,394,579,706.90
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			21	22	23	24
	NPV	IRR				
Base case						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			1,145,813,716.11	1,145,813,716.11	1,145,813,716.11	1,145,813,716.11
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			1,394,579,706.90	1,394,579,706.90	1,394,579,706.90	1,394,579,706.90
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			25	26	27	28
	NPV	IRR				
Base case						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,254,165,336.31	1,254,165,336.31
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			1,145,813,716.11	1,145,813,716.11	1,145,813,716.11	1,145,813,716.11
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			1,394,579,706.90	1,394,579,706.90	1,394,579,706.90	1,394,579,706.90
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)					
Year			29	30	31	32
	NPV	IRR				
Base case						
Cash flow			1,254,165,336.31	1,254,165,336.31	1,045,137,780.26	86,552,760.42
	R -1,664,541,443.47	9%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,045,137,780.26	571,868,697.01
	R -1,173,492,270.55	10%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,254,165,336.31	1,254,165,336.31	1,045,137,780.26	331,369,074.54
	R -1,417,097,555.68	10%				
Sensitivity 3	750	substitute steady state cost only				
Cash flow			1,145,813,716.11	1,145,813,716.11	954,844,763.42	86,552,760.42
	R -1,996,501,036.22	9%				
Sensitivity 4	600	substitute steady state cost only				
Cash flow			1,394,579,706.90	1,394,579,706.90	1,162,149,755.75	86,552,760.42
	R -1,234,350,393.34	10%				

Table B1.5. MR operating cost sensitivity analysis data

MR	OPERATING COST (R/t)		
Year			33
	NPV	IRR	
Base case			
Cash flow			54,355,133.54
	R -1,664,541,443.47	9%	
Sensitivity 1	900	substitute build up cost only	
Cash flow			359,133,541.72
	R -1,173,492,270.55	10%	
Sensitivity 2	1100	substitute build up cost only	
Cash flow			208,099,778.81
	R -1,417,097,555.68	10%	
Sensitivity 3	750	substitute steady state cost only	
Cash flow			54,355,133.54
	R -1,996,501,036.22	9%	
Sensitivity 4	600	substitute steady state cost only	
Cash flow			54,355,133.54
	R -1,234,350,393.34	10%	

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)						
Year			1	2	3	4	5
	NPV	IRR					
Base case	6.34						
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63	-74,812,039.93
	R -109,614,208.27	12%					
Sensitivity 1	7.34						
Cash flow			-2,500,000,000.00	-2,396,954,223.75	-1,494,493,437.45	-496,364,323.83	-47,543,591.97
	R 1,697,914,378.22	14%					
Sensitivity 2	6.5						
Cash flow			-2,500,000,000.00	-2,398,046,083.71	-1,496,610,210.28	-509,133,202.30	-70,449,088.26
	R 180,397,568.85	12%					
Sensitivity 3	6.45						
Cash flow			-2,500,000,000.00	-2,398,111,075.37	-1,496,736,208.66	-509,893,254.59	-71,812,510.66
	R 89,806,479.39	12%					
Sensitivity 4	5.34						
Cash flow			-2,500,000,000.00	-2,399,553,890.31	-1,499,533,372.75	-526,766,415.43	-102,080,487.89
	R -1,954,144,712.91	9%					
Sensitivity 5	4.34						
Cash flow			-2,500,000,000.00	-2,400,853,723.58	-1,502,053,340.40	-541,967,461.23	-129,348,935.85
	R -3,908,109,684.06	4%					
Sensitivity 6	4.62						
Cash flow			-2,500,000,000.00	-2,400,489,770.27	-1,501,347,749.46	-537,711,168.41	-121,713,770.42
	R -3,358,922,571.79	6%					
Sensitivity 7	3.9						
Cash flow			-2,500,000,000.00	-2,401,425,650.22	-1,503,162,126.17	-548,655,921.38	-141,347,052.96
	R -4,775,586,805.60	1%					
Sensitivity 8	3.34						
Cash flow							
	R -5,890,575,308.75	nothing					

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)						
Year			6	7	8	9	10
	NPV	IRR					
Base case	6.34						
Cash flow			98,741,779.43	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	7.34						
Cash flow			206,102,191.25	372,397,460.47	533,769,693.34	1,724,165,460.46	2,246,571,900.37
	R 1,697,914,378.22	14%					
Sensitivity 2	6.5						
Cash flow			116,036,245.68	198,154,344.35	284,021,226.90	1,402,176,414.96	1,827,023,105.06
	R 180,397,568.85	12%					
Sensitivity 3	6.45						
Cash flow			110,637,164.29	187,705,980.27	269,045,238.38	1,382,969,567.48	1,801,996,757.63
	R 89,806,479.39	12%					
Sensitivity 4	5.34						
Cash flow			-14,961,834.14	-65,806,023.12	-94,321,966.46	954,953,555.19	1,244,295,789.73
	R -1,954,144,712.91	9%					
Sensitivity 5	4.34						
Cash flow			-169,351,003.78	-363,783,921.41	-521,423,620.69	565,367,788.22	736,669,081.66
	R -3,908,109,684.06	4%					
Sensitivity 6	4.62						
Cash flow			-126,122,036.28	-280,350,109.89	-401,835,157.50	674,983,466.06	879,497,312.08
	R -3,358,922,571.79	6%					
Sensitivity 7	3.9						
Cash flow			-237,282,238.42	-494,894,196.66	-709,348,348.54	391,970,628.19	510,734,160.53
	R -4,775,586,805.60	1%					
Sensitivity 8	3.34						
Cash flow							
	R -5,890,575,308.75	nothing					

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)						
Year			11	12	13	14	15
	NPV	IRR					
Base case	6.34						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	7.34						
Cash flow			2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37
	R 1,697,914,378.22	14%					
Sensitivity 2	6.5						
Cash flow			1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06
	R 180,397,568.85	12%					
Sensitivity 3	6.45						
Cash flow			1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63
	R 89,806,479.39	12%					
Sensitivity 4	5.34						
Cash flow			1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73
	R -1,954,144,712.91	9%					
Sensitivity 5	4.34						
Cash flow			736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66
	R -3,908,109,684.06	4%					
Sensitivity 6	4.62						
Cash flow			879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08
	R -3,358,922,571.79	6%					
Sensitivity 7	3.9						
Cash flow			510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53
	R -4,775,586,805.60	1%					
Sensitivity 8	3.34						
Cash flow							
	R -5,890,575,308.75	nothing					

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)						
Year			16	17	18	19	20
	NPV	IRR					
Base case	6.34						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	7.34						
Cash flow			2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37
	R 1,697,914,378.22	14%					
Sensitivity 2	6.5						
Cash flow			1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06
	R 180,397,568.85	12%					
Sensitivity 3	6.45						
Cash flow			1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63
	R 89,806,479.39	12%					
Sensitivity 4	5.34						
Cash flow			1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73
	R -1,954,144,712.91	9%					
Sensitivity 5	4.34						
Cash flow			736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66
	R -3,908,109,684.06	4%					
Sensitivity 6	4.62						
Cash flow			879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08
	R -3,358,922,571.79	6%					
Sensitivity 7	3.9						
Cash flow			510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53
	R -4,775,586,805.60	1%					
Sensitivity 8	3.34						
Cash flow							
	R -5,890,575,308.75	nothing					

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)						
Year			21	22	23	24	25
	NPV	IRR					
Base case	6.34						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	7.34						
Cash flow			2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37
	R 1,697,914,378.22	14%					
Sensitivity 2	6.5						
Cash flow			1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06
	R 180,397,568.85	12%					
Sensitivity 3	6.45						
Cash flow			1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63
	R 89,806,479.39	12%					
Sensitivity 4	5.34						
Cash flow			1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73
	R -1,954,144,712.91	9%					
Sensitivity 5	4.34						
Cash flow			736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66
	R -3,908,109,684.06	4%					
Sensitivity 6	4.62						
Cash flow			879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08
	R -3,358,922,571.79	6%					
Sensitivity 7	3.9						
Cash flow			510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53
	R -4,775,586,805.60	1%					
Sensitivity 8	3.34						
Cash flow							
	R -5,890,575,308.75	nothing					

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)						
Year			26	27	28	29	30
	NPV	IRR					
Base case	6.34						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -109,614,208.27	12%					
Sensitivity 1	7.34						
Cash flow			2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	2,246,571,900.37	1,868,406,437.43
	R 1,697,914,378.22	14%					
Sensitivity 2	6.5						
Cash flow			1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,827,023,105.06	1,519,480,293.63
	R 180,397,568.85	12%					
Sensitivity 3	6.45						
Cash flow			1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,801,996,757.63	1,498,666,631.43
	R 89,806,479.39	12%					
Sensitivity 4	5.34						
Cash flow			1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,244,295,789.73	1,034,843,471.16
	R -1,954,144,712.91	9%					
Sensitivity 5	4.34						
Cash flow			736,669,081.66	736,669,081.66	736,669,081.66	736,669,081.66	612,665,570.25
	R -3,908,109,684.06	4%					
Sensitivity 6	4.62						
Cash flow			879,497,312.08	879,497,312.08	879,497,312.08	879,497,312.08	731,451,523.69
	R -3,358,922,571.79	6%					
Sensitivity 7	3.9						
Cash flow			510,734,160.53	510,734,160.53	510,734,160.53	510,734,160.53	424,762,275.89
	R -4,775,586,805.60	1%					
Sensitivity 8	3.34						
Cash flow							
	R -5,890,575,308.75	nothing					

Table B2.1. UG2 grade sensitivity analysis data.

UG2	GRADE (g/t)			
Year			31	32
	NPV	IRR		
Base case	6.34			
Cash flow			219,579,248.48	140,256,244.97
	R -109,614,208.27	12%		
Sensitivity 1	7.34			
Cash flow			496,529,947.30	317,158,503.84
	R 1,697,914,378.22	14%		
Sensitivity 2	6.5			
Cash flow			264,205,792.46	168,761,449.94
	R 180,397,568.85	12%		
Sensitivity 3	6.45			
Cash flow			250,274,640.36	159,862,926.53
	R 89,806,479.39	12%		
Sensitivity 4	5.34			
Cash flow			-87,741,364.15	-56,044,796.35
	R -1,954,144,712.91	9%		
Sensitivity 5	4.34			
Cash flow			-485,045,228.54	-309,822,639.73
	R -3,908,109,684.06	4%		
Sensitivity 6	4.62			
Cash flow			-373,800,146.51	-238,764,843.59
	R -3,358,922,571.79	6%		
Sensitivity 7	3.9			
Cash flow			-659,858,928.88	-421,484,890.82
	R -4,775,586,805.60	1%		
Sensitivity 8	3.34			
Cash flow				
	R -5,890,575,308.75	nothing		

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %						
Year			1	2	3	4	5
	NPV	IRR					
Base case	0.9						
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63	-74,812,039.93
	R -109,614,208.27	12%					
Sensitivity 1	0.95						
Cash flow			-2,500,000,000.00	-2,397,796,226.87	-1,496,125,816.49	-506,211,223.50	-65,207,486.60
	R 528,368,500.82	13%					
Sensitivity 2	0.92						
Cash flow			-2,500,000,000.00	-2,398,070,924.96	-1,496,658,369.66	-509,423,711.18	-70,970,218.60
	R 145,775,588.28	12%					
Sensitivity 3	0.99						
Cash flow			-2,500,000,000.00	-2,397,429,962.73	-1,495,415,745.61	-501,927,906.60	-57,523,843.93
	R 1,037,657,844.54	13%					
Sensitivity 4	0.85						
Cash flow			-2,500,000,000.00	-2,398,711,887.20	-1,497,900,993.71	-516,919,515.76	-84,416,593.27
	R -749,371,190.75	11%					
Sensitivity 5	0.8						
Cash flow			-2,500,000,000.00	-2,399,169,717.36	-1,498,788,582.31	-522,273,661.90	-94,021,146.61
	R -1,391,235,122.87	10%					

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %						
Year			6	7	8	9	10
	NPV	IRR					
Base case	0.9						
Cash flow			98,741,779.43	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	0.95						
Cash flow			136,749,090.92	238,234,422.73	341,469,339.25	1,475,969,214.02	1,923,174,450.52
	R 528,368,500.82	13%					
Sensitivity 2	0.92						
Cash flow			113,973,167.91	194,161,900.51	278,298,724.07	1,394,835,744.51	1,817,458,278.29
	R 145,775,588.28	12%					
Sensitivity 3	0.99						
Cash flow			166,996,232.40	296,753,975.25	425,347,364.52	1,584,017,460.34	2,063,960,331.94
	R 1,037,657,844.54	13%					
Sensitivity 4	0.85						
Cash flow			60,477,734.88	90,616,096.71	129,883,071.95	1,205,146,621.84	1,570,295,078.14
	R -749,371,190.75	11%					
Sensitivity 5	0.8						
Cash flow			21,908,819.82	15,932,212.52	22,836,171.29	1,069,269,953.02	1,393,249,015.50
	R -1,391,235,122.87	10%					

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %						
Year			11	12	13	14	15
	NPV	IRR					
Base case	0.9						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	0.95						
Cash flow			1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52
	R 528,368,500.82	13%					
Sensitivity 2	0.92						
Cash flow			1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29
	R 145,775,588.28	12%					
Sensitivity 3	0.99						
Cash flow			2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94
	R 1,037,657,844.54	13%					
Sensitivity 4	0.85						
Cash flow			1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14
	R -749,371,190.75	11%					
Sensitivity 5	0.8						
Cash flow			1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50
	R -1,391,235,122.87	10%					

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %						
Year			16	17	18	19	20
	NPV	IRR					
Base case	0.9						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	0.95						
Cash flow			1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52
	R 528,368,500.82	13%					
Sensitivity 2	0.92						
Cash flow			1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29
	R 145,775,588.28	12%					
Sensitivity 3	0.99						
Cash flow			2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94
	R 1,037,657,844.54	13%					
Sensitivity 4	0.85						
Cash flow			1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14
	R -749,371,190.75	11%					
Sensitivity 5	0.8						
Cash flow			1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50
	R -1,391,235,122.87	10%					

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %						
Year			21	22	23	24	25
	NPV	IRR					
Base case	0.9						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	0.95						
Cash flow			1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52
	R 528,368,500.82	13%					
Sensitivity 2	0.92						
Cash flow			1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29
	R 145,775,588.28	12%					
Sensitivity 3	0.99						
Cash flow			2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94
	R 1,037,657,844.54	13%					
Sensitivity 4	0.85						
Cash flow			1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14
	R -749,371,190.75	11%					
Sensitivity 5	0.8						
Cash flow			1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50
	R -1,391,235,122.87	10%					

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %						
Year			26	27	28	29	30
	NPV	IRR					
Base case	0.9						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -109,614,208.27	12%					
Sensitivity 1	0.95						
Cash flow			1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,923,174,450.52	1,599,446,482.47
	R 528,368,500.82	13%					
Sensitivity 2	0.92						
Cash flow			1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,817,458,278.29	1,511,525,514.21
	R 145,775,588.28	12%					
Sensitivity 3	0.99						
Cash flow			2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	2,063,960,331.94	1,716,533,875.53
	R 1,037,657,844.54	13%					
Sensitivity 4	0.85						
Cash flow			1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,570,295,078.14	1,305,967,297.19
	R -749,371,190.75	11%					
Sensitivity 5	0.8						
Cash flow			1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,393,249,015.50	1,158,723,399.45
	R -1,391,235,122.87	10%					

Table B2.2. UG2 recovery sensitivity analysis data.

UG2	RECOVERY %			
Year			31	32
	NPV	IRR		
Base case	0.9			
Cash flow			219,579,248.48	140,256,244.97
	R -109,614,208.27	12%		
Sensitivity 1	0.95			
Cash flow			317,645,896.98	202,896,316.69
	R 528,368,500.82	13%		
Sensitivity 2	0.92			
Cash flow			258,882,534.02	165,361,218.60
	R 145,775,588.28	12%		
Sensitivity 3	0.99			
Cash flow			395,671,967.00	252,735,468.92
	R 1,037,657,844.54	13%		
Sensitivity 4	0.85			
Cash flow			120,821,462.28	77,174,709.03
	R -749,371,190.75	11%		
Sensitivity 5	0.8			
Cash flow			21,242,950.03	13,568,934.33
	R -1,391,235,122.87	10%		

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE						
Year			1	2	3	4	5
	NPV	IRR					
Base case							
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63	-74,812,039.93
	R -109,614,208.27	12%					
Sensitivity 1	8581.62	same price					
Cash flow			-2,500,000,000.00	-2,398,623,417.19	-1,497,246,834.37	-513,481,006.24	-80,816,444.36
	R -126,888,043.85	12%					
Sensitivity 2	9138.84	17-May-11					
Cash flow			-2,500,000,000.00	-2,398,112,301.18	-1,496,224,602.35	-507,347,614.12	-69,980,784.94
	R 617,587,539.35	13%					
Sensitivity 3	10280.69	18-Feb-11					
Cash flow			-2,500,000,000.00	-2,397,064,927.04	-1,494,129,854.08	-494,779,124.48	-47,776,453.25
	R 2,137,227,223.43	15%					
Sensitivity 4	9731.02	20-Aug-11					
Cash flow			-2,500,000,000.00	-2,397,569,117.73	-1,495,138,235.46	-500,829,412.79	-58,465,295.92
	R 1,406,572,565.06	14%					
Sensitivity 5	7211.26	2010					
Cash flow			-2,500,000,000.00	-2,399,880,394.57	-1,499,760,789.13	-528,564,734.80	-107,464,364.82
	R -1,997,124,816.75	9%					
Sensitivity 6	6672.89	2009					
Cash flow			-2,500,000,000.00	-2,400,374,220.21	-1,500,748,440.43	-534,490,642.55	-117,933,468.51
	R -2,770,664,171.11	7%					

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE						
Year			6	7	8	9	10
	NPV	IRR					
Base case							
Cash flow			98,741,779.43	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	8581.62	same price					
Cash flow			74,683,345.11	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -126,888,043.85	12%					
Sensitivity 2	9138.84	17-May-11					
Cash flow			117,709,839.40	250,602,285.27	359,196,608.88	1,498,775,500.24	1,952,890,833.86
	R 617,587,539.35	13%					
Sensitivity 3	10280.69	18-Feb-11					
Cash flow			204,992,122.92	424,935,692.50	609,074,492.59	1,821,791,247.48	2,373,777,412.19
	R 2,137,227,223.43	15%					
Sensitivity 4	9731.02	20-Aug-11					
Cash flow			163,106,971.27	341,269,894.41	489,153,515.32	1,666,432,236.34	2,171,346,034.86
	R 1,406,572,565.06	14%					
Sensitivity 5	7211.26	2010					
Cash flow			-45,647,348.01	-69,502,696.03	-99,620,530.97	950,148,427.25	1,238,034,751.77
	R -1,997,124,816.75	9%					
Sensitivity 6	6672.89	2009					
Cash flow			-104,906,425.52	-188,020,851.04	-269,496,553.15	795,763,920.28	1,036,873,144.50
	R -2,770,664,171.11	7%					

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE						
Year			11	12	13	14	15
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	8581.62	same price					
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -126,888,043.85	12%					
Sensitivity 2	9138.84	17-May-11					
Cash flow			1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86
	R 617,587,539.35	13%					
Sensitivity 3	10280.69	18-Feb-11					
Cash flow			2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19
	R 2,137,227,223.43	15%					
Sensitivity 4	9731.02	20-Aug-11					
Cash flow			2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86
	R 1,406,572,565.06	14%					
Sensitivity 5	7211.26	2010					
Cash flow			1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77
	R -1,997,124,816.75	9%					
Sensitivity 6	6672.89	2009					
Cash flow			1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50
	R -2,770,664,171.11	7%					

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE						
Year			16	17	18	19	20
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	8581.62	same price					
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -126,888,043.85	12%					
Sensitivity 2	9138.84	17-May-11					
Cash flow			1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86
	R 617,587,539.35	13%					
Sensitivity 3	10280.69	18-Feb-11					
Cash flow			2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19
	R 2,137,227,223.43	15%					
Sensitivity 4	9731.02	20-Aug-11					
Cash flow			2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86
	R 1,406,572,565.06	14%					
Sensitivity 5	7211.26	2010					
Cash flow			1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77
	R -1,997,124,816.75	9%					
Sensitivity 6	6672.89	2009					
Cash flow			1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50
	R -2,770,664,171.11	7%					

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE						
Year			21	22	23	24	25
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	8581.62	same price					
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -126,888,043.85	12%					
Sensitivity 2	9138.84	17-May-11					
Cash flow			1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86
	R 617,587,539.35	13%					
Sensitivity 3	10280.69	18-Feb-11					
Cash flow			2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19
	R 2,137,227,223.43	15%					
Sensitivity 4	9731.02	20-Aug-11					
Cash flow			2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86
	R 1,406,572,565.06	14%					
Sensitivity 5	7211.26	2010					
Cash flow			1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77
	R -1,997,124,816.75	9%					
Sensitivity 6	6672.89	2009					
Cash flow			1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50
	R -2,770,664,171.11	7%					

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE						
Year			26	27	28	29	30
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -109,614,208.27	12%					
Sensitivity 1	8581.62	same price					
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -126,888,043.85	12%					
Sensitivity 2	9138.84	17-May-11					
Cash flow			1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,952,890,833.86	1,624,160,706.80
	R 617,587,539.35	13%					
Sensitivity 3	10280.69	18-Feb-11					
Cash flow			2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	2,373,777,412.19	1,974,199,444.60
	R 2,137,227,223.43	15%					
Sensitivity 4	9731.02	20-Aug-11					
Cash flow			2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	2,171,346,034.86	1,805,843,342.37
	R 1,406,572,565.06	14%					
Sensitivity 5	7211.26	2010					
Cash flow			1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,238,034,751.77	1,029,636,353.76
	R -1,997,124,816.75	9%					
Sensitivity 6	6672.89	2009					
Cash flow			1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	1,036,873,144.50	862,336,281.19
	R -2,770,664,171.11	7%					

Table B2.3. UG2 basket price sensitivity analysis data.

UG2	BASKET PRICE			
Year			31	32
	NPV	IRR		
Base case				
Cash flow			219,579,248.48	140,256,244.97
	R -109,614,208.27	12%		
Sensitivity 1	8581.62	same price		
Cash flow			219,579,248.48	140,256,244.97
	R -126,888,043.85	12%		
Sensitivity 2	9138.84	17-May-11		
Cash flow			334,136,380.36	213,429,612.95
	R 617,587,539.35	13%		
Sensitivity 3	10280.69	18-Feb-11		
Cash flow			566,580,923.34	361,903,564.78
	R 2,137,227,223.43	15%		
Sensitivity 4	9731.02	20-Aug-11		
Cash flow			455,026,525.88	290,648,193.41
	R 1,406,572,565.06	14%		
Sensitivity 5	7211.26	2010		
Cash flow			-92,670,261.37	-59,193,129.45
	R -1,997,124,816.75	9%		
Sensitivity 6	6672.89	2009		
Cash flow			-250,694,468.05	-160,131,091.47
	R -2,770,664,171.11	7%		

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL						
Year			1	2	3	4	5
	NPV	IRR					
Base case							
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63	-74,812,039.93
	R -109,614,208.27	12%					
Sensitivity 1	-10%						
Cash flow			-2,000,000,000.00	-1,898,254,057.03	-1,297,013,405.10	-681,565,369.63	-354,812,039.93
	R 596,616,577.82	13%					
Sensitivity 2	-20%						
Cash flow			-1,500,000,000.00	-1,198,254,057.03	-947,013,405.10	-811,565,369.63	-764,812,039.93
	R 1,397,532,757.66	15%					
Sensitivity 3	10%						
Cash flow			-3,000,000,000.00	-2,298,254,057.03	-1,397,013,405.10	-831,565,369.63	-114,812,039.93
	R -649,014,609.00	11%					
Sensitivity 4	20%						
Cash flow			-3,500,000,000.00	-2,498,254,057.03	-1,597,013,405.10	-481,565,369.63	-234,812,039.93
	R -1,257,250,334.20	10%					

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL						
Year			6	7	8	9	10
	NPV	IRR					
Base case							
Cash flow			98,741,779.43	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	-10%						
Cash flow			70,648,004.39	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R 596,616,577.82	13%					
Sensitivity 2	-20%						
Cash flow			-200,572,664.51	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R 1,397,532,757.66	15%					
Sensitivity 3	10%						
Cash flow			63,595,137.67	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -649,014,609.00	11%					
Sensitivity 4	20%						
Cash flow			35,265,978.93	164,684,436.36	236,047,692.12	1,340,695,806.15	1,746,914,431.42
	R -1,257,250,334.20	10%					

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL						
Year			11	12	13	14	15
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	-10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 596,616,577.82	13%					
Sensitivity 2	-20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 1,397,532,757.66	15%					
Sensitivity 3	10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -649,014,609.00	11%					
Sensitivity 4	20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -1,257,250,334.20	10%					

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL						
Year			16	17	18	19	20
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	-10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 596,616,577.82	13%					
Sensitivity 2	-20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 1,397,532,757.66	15%					
Sensitivity 3	10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -649,014,609.00	11%					
Sensitivity 4	20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -1,257,250,334.20	10%					

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL						
Year			21	22	23	24	25
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%					
Sensitivity 1	-10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 596,616,577.82	13%					
Sensitivity 2	-20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 1,397,532,757.66	15%					
Sensitivity 3	10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -649,014,609.00	11%					
Sensitivity 4	20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -1,257,250,334.20	10%					

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL						
Year			26	27	28	29	30
	NPV	IRR					
Base case							
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -109,614,208.27	12%					
Sensitivity 1	-10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R 596,616,577.82	13%					
Sensitivity 2	-20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R 1,397,532,757.66	15%					
Sensitivity 3	10%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -649,014,609.00	11%					
Sensitivity 4	20%						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,452,856,313.56
	R -1,257,250,334.20	10%					

Table B2.4. UG2 capital sensitivity analysis data.

UG2	CAPITAL			
Year			31	32
	NPV	IRR		
Base case				
Cash flow			219,579,248.48	140,256,244.97
	R -109,614,208.27	12%		
Sensitivity 1	-10%			
Cash flow			219,579,248.48	140,256,244.97
	R 596,616,577.82	13%		
Sensitivity 2	-20%			
Cash flow			219,579,248.48	140,256,244.97
	R 1,397,532,757.66	15%		
Sensitivity 3	10%			
Cash flow			219,579,248.48	140,256,244.97
	R -649,014,609.00	11%		
Sensitivity 4	20%			
Cash flow			219,579,248.48	140,256,244.97
	R -1,257,250,334.20	10%		

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			1	2	3	4
	NPV	IRR				
Base case						
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			-2,500,000,000.00	-2,396,259,057.03	-1,493,023,405.10	-487,625,369.63
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			-2,500,000,000.00	-2,397,259,057.03	-1,495,023,405.10	-499,625,369.63
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			-2,500,000,000.00	-2,398,254,057.03	-1,497,013,405.10	-511,565,369.63
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			5	6	7	8
	NPV	IRR				
Base case						
Cash flow			-74,812,039.93	98,741,779.43	164,684,436.36	236,047,692.12
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			-32,518,039.93	262,946,915.61	493,928,428.29	707,964,080.55
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			-53,718,039.93	181,481,721.09	330,641,255.55	473,919,132.95
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			-74,812,039.93	98,741,779.43	164,684,436.36	236,047,692.12
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			-74,812,039.93	98,741,779.43	164,684,436.36	236,047,692.12
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			9	10	11	12
	NPV	IRR				
Base case						
Cash flow			1,340,695,806.15	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,340,695,806.15	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,340,695,806.15	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			1,468,401,520.99	1,913,313,815.39	1,913,313,815.39	1,913,313,815.39
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			1,242,215,527.41	1,618,595,524.66	1,618,595,524.66	1,618,595,524.66
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			13	14	15	16
	NPV	IRR				
Base case						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			1,913,313,815.39	1,913,313,815.39	1,913,313,815.39	1,913,313,815.39
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			1,618,595,524.66	1,618,595,524.66	1,618,595,524.66	1,618,595,524.66
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			17	18	19	20
	NPV	IRR				
Base case						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			1,913,313,815.39	1,913,313,815.39	1,913,313,815.39	1,913,313,815.39
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			1,618,595,524.66	1,618,595,524.66	1,618,595,524.66	1,618,595,524.66
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			21	22	23	24
	NPV	IRR				
Base case						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			1,913,313,815.39	1,913,313,815.39	1,913,313,815.39	1,913,313,815.39
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			1,618,595,524.66	1,618,595,524.66	1,618,595,524.66	1,618,595,524.66
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			25	26	27	28
	NPV	IRR				
Base case						
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,746,914,431.42	1,746,914,431.42	1,746,914,431.42	1,746,914,431.42
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			1,913,313,815.39	1,913,313,815.39	1,913,313,815.39	1,913,313,815.39
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			1,618,595,524.66	1,618,595,524.66	1,618,595,524.66	1,618,595,524.66
	R -494,323,435.75	11%				

Table B2.5. UG2 operating cost sensitivity analysis data.

UG2	OPERATING COST (R/t)					
Year			29	30	31	32
	NPV	IRR				
Base case						
Cash flow			1,746,914,431.42	1,452,856,313.56	219,579,248.48	140,256,244.97
	R -109,614,208.27	12%				
Sensitivity 1	900	substitute build up cost only				
Cash flow			1,746,914,431.42	1,452,856,313.56	658,571,237.72	420,662,378.09
	R 377,297,010.34	13%				
Sensitivity 2	1100	substitute build up cost only				
Cash flow			1,746,914,431.42	1,452,856,313.56	440,855,007.40	281,596,135.98
	R 135,569,447.15	12%				
Sensitivity 3	600	substitute steady state cost only				
Cash flow			1,913,313,815.39	1,591,245,688.12	219,579,248.48	140,256,244.97
	R 389,263,003.77	13%				
Sensitivity 4	750	substitute steady state cost only				
Cash flow			1,618,595,524.66	1,346,137,329.22	219,579,248.48	140,256,244.97
	R -494,323,435.75	11%				

Appendix C

Table C1. Risk register for the TPM project.

Risk Analysis TPM Project Risk Register							
Main Risk Number	Main Risk Identified	Risk Breakdown	Unwanted event	Likelyhood	Consequence	Risk Rating	Final Risk Rating
1	Legislation and Govermant						16
		nationalisation of mines	who knows	2	5	19	
		DMR	section 54, fines	5	3	16	
		changes to MPRDA	new regulations, unplanned costs	3	3	13	
		taxation	go up, change	3	3	13	
		royalties	go up, change	3	3	13	
		political	unstability	4	3	17	
		local government	new demands	4	4	21	
		mining rights	not renewed	1	5	15	
2	Economic factors						17
		supply and demand	over supply, less demand	3	4	18	
		metal prices	drop	4	4	21	
		exchange rates	loss of value	4	4	21	
		inflation	raise	3	3	13	
		cost of capital	go up	3	3	13	
3	Capital						18
		deferrement	loose value	3	4	18	
		cost overruns	over spend	4	3	17	
		company profile	changes, cant give us money	3	4	18	
		planning	fatal flaws	3	4	18	

Table C1. Risk register for the TPM project.

Risk Analysis TPM Project
Risk Register

Main Risk Number	Main Risk Identified	Risk Breakdown	Unwanted event	Likelyhood	Consequence	Risk Rating	Final Risk Rating
4	Infrastructure						20
		power	not in place, not sufficient	3	4	18	
		roads	unsafe, non existant	4	4	21	
		surface infrastructure on mine	late, not planned well	3	4	18	
		underground infrastructure	late	4	5	24	
		water	not sufficient	4	5	24	
		training/education	very low, must invest	4	4	21	
		stores/spares	not in place, not planned	4	4	21	
		accessability	no airport, railways	3	3	13	
		management practises and development policies	not implemented, monitored	3	4	18	
		health care	low, inadequate	4	4	21	
		housing	poverty, not in place	5	5	25	
5	Labor						20
		skills retention	low, high turnover	5	4	23	
		renumeration	very high to keep skills	4	3	17	
		health and safety	poverty, high TB, AIDS rates	4	3	17	
		strikes/violence	disruption of operation, damages	3	4	18	
		theft	disruption, damages, safety	5	4	23	
		recruitment	difficult	5	3	20	
		training	expensive, long	5	3	20	
6	Community						21
		relationship	tentative-bad	5	5	25	
		access to land/lease agreements	not in place	4	5	24	
		upgrading local community	poverty, not delivering on promises	4	4	21	
		poverty	major problem	5	2	16	
		uprising/vandalism	unhappiness due to poverty	3	4	18	

Table C1. Risk register for the TPM project.

Risk Analysis TPM Project Risk Register							
Main Risk Number	Main Risk Identified	Risk Breakdown	Unwanted event	Likelyhood	Consequence	Risk Rating	Final Risk Rating
7	SHE						16
	Safety, Health and	policies and procedures	not properly implemented, fatalities	3	4	18	
	Environment	management	supervision lacking	4	3	17	
		occupational health	managed, need good quality doctors	4	3	17	
		environmental management	major spill/accident	3	4	18	
		environmental conservation	historical/archilogical sites	2	3	9	
8	Build-up						23
		design	flawed	3	4	18	
		construction	late	4	5	24	
		commisioning	late, inadequate	4	5	24	
		time, cost, quality	late, overspent, poor	4	5	24	
		technology	not in place, not appropriate	4	4	24	
9	Ore body						15
		grade	lower than expected	3	3	13	
		structure	more than expected	3	3	13	
		dilution/losses	high due to structure/mining practises	4	3	17	
		geotechnical	problems, safety, cost	4	3	17	
10	Processing						14
		recovery	low	3	3	13	
		transport	expensive, safety	3	3	13	
		management	skills	3	2	8	
		production	down on production-cant fill plant	4	4	21	

Table C2. Risk matrix template.

Event Risk Rating/ Priority (1)					
Consequence / Likelihood	1 Minor	2 Low	3 Medium	4 High	5 Major
5 Almost Certain	Medium (11)	Significant (16)	Significant (20)	High (23)	High (25)
4 Likely	Medium (7)	Medium (12)	Significant (17)	High (21)	High (24)
3 Possible	Low (4)	Medium (8)	Significant (13)	Significant (18)	High (22)
2 Unlikely	Low (2)	Low (5)	Medium (9)	Significant (14)	Significant (19)
1 Rare	Low (1)	Low (3)	Medium (6)	Medium (10)	Significant (15)