An assessment of the economic viability of mining the UG2 Reef within the No.12 Shaft lease area, Impala Platinum Limited.

By
Manivasan Balakrishna

Submitted in partial fulfillment of the requirements for the degree M. Sc Earth Science Practice and Management
Faculty of Natural & Agricultural Science
University of Pretoria
August 2006

Abstract

The Impala Platinum Mines are located in the western limb of the Bushveld Complex. The economic platinum mineralization which is currently being mined at Impala Platinum occurs in the Merensky and the Upper Group 2 (UG2) Reefs. These ore bearing horizons are stratigraphically located in the Upper Critical Zone of the Bushveld Complex. Due to the variable stratigraphic distance between the UG2 Chromitite Layer and the Merensky Reef, each of these reef horizons is influenced by different geological structural features. It has been observed on several shafts that there is no direct correlation between geological structures encountered on the Merensky Reef to that seen on the UG2 Reef horizon. This observation was prevalent during the geological structural evaluation of the two reef horizons in the No. 12 Shaft area. The Merensky Reef presented itself geologically more favorable to the UG2 Chromitite Layer in terms of structural complexities. Other positive factors included global supply and demand for the resultant metals and the encouraging metal price. There was nonetheless limited trial mining of the UG2 Reef within the No. 12 Shaft area. However, as a result of unfavorable market conditions at the time and complicated geological features which would result in very low extraction rates, it was decided to temporarily cease all mining operations related to the UG2 Reef horizon in this area. It has been strongly recommended in previous geological evaluation reports that the UG2 Chromitite Layer in the No. 12 Shaft area be appraised in significant detail in order to determine the possibility of economically exploiting this mineralized horizon. Due to the current favorable economic climate for platinum group metals, it was suggested by senior management that the UG2 Chromitite Layer be re – evaluated. This dissertation discusses some of the investigative research which was conducted during this re - assessment. The project work comprised research in the following aspects, exploratory drilling, interpretation of geological structural data, grade estimation, mineral resource estimation, financial and sensitivity analyses and aspects concerning risk management. Throughout the project all the input parameters and resultant calculations related to grade, mineral resource and reserve, financial and sensitivity analyses were based on tentative estimates which reflect the author's personal opinions and assumptions. It is not absolute data of Impala Platinum and thus does not in any way reflect the views of the company. The UG2 Chromitite Layer in the No. 12 Shaft area was benchmarked against the neighboring shafts in terms of its geological and metallurgical characteristics. It was found that regionally, the UG2 Reef displays very limited variability. The financial assessment based on average market input data and assumptions have revealed positive results with regards to general financial and marketing decisions and strategies. In the risk assessment, the high impact risks generally facing all mining companies were found to be within manageable levels. From the investigative geological research based on general business decision criteria, market averages, estimations and assumptions which are used to broadly evaluate projects in the mining industry, it has been demonstrated that it would be economically viable to mine the UG2 Chromitite Layer in the No. 12 Shaft area of Impala Platinum.

Contents

Chapter 1. The purpose and objectives of the investigation

1.1. Introduction

Platinum in South Africa was first recorded in 1892, associated with the gold mineralization in the Witwatersrand conglomerates. Later, in 1906, platinum was discovered in the chromitite layers of the Bushveld Complex (Cawthorn, 1999). Several other small uneconomic platinum occurrences in the Bushveld Complex were discovered until 1923 when mineralization was found in quartz veins northwest of the town, Naboomspruit. This discovery sparked an interest in the metal which led to further discoveries near Steelpoort in 1924.

Dr Hans Merensky, who was involved in the early exploration, recognized the potential for laterally continuous platiniferous ore bodies and he soon discovered the famous Merensky Reef. He proved that this layer of rock could be traced for over 100km in the Steelpoort area and equal distances around Rustenburg. Extensive assaying on samples from the Bushveld Complex during this period confirmed the continuous nature of the platinum mineralization. Presently, the Bushveld Complex is regarded as the largest and economically the most important depository of the platinum group elements on Earth.

1.2. Geological Setting

The 2.05 Ga old Bushveld Complex, located in the Kaapvaal Craton, is a layered complex comprising mafic to ultramafic rocks (Walraven 1987; Anhaeusser, 1987; Buchanan, 1987). The magmatic event which gave rise to the complex commenced with the extrusion of large volumes of basaltic and felsic magma at the close of the Transvaal basin sedimentation. The main Complex has an elliptical form and covers an area of approximately 66 000 km^2. The mafic rocks have a thickness varying between 7km and 9km.

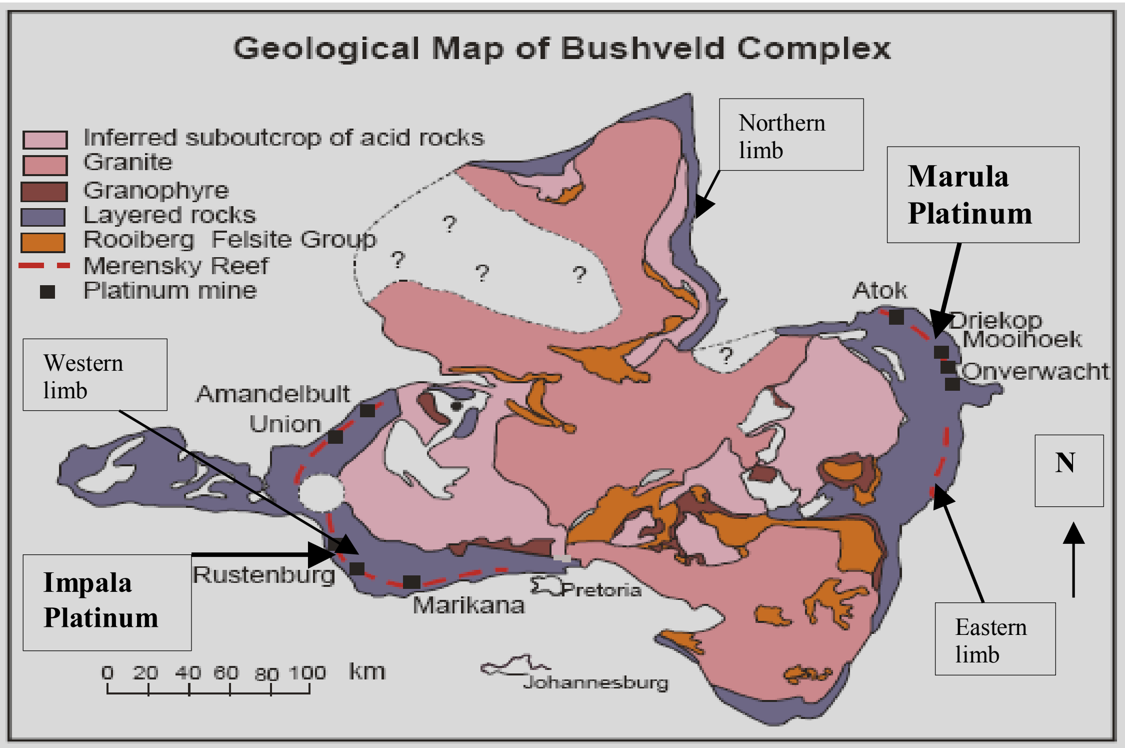


Figure 1. Geological Map of the Bushveld Complex showing the locality of Impala and Marula Platinum Mines (adapted from Cawthorn et al., 2002).

The Bushveld Complex (Figure 1) contains at least 85 percent of the world's platinum group element reserves (Ondendaal, 1992; Morrissey, 1988; Vermaak 1985).
It exhibits a transgressive relationship to the encircling sedimentary rocks, hence a variety of floor and roof rocks are present in different parts of the complex (Du Toit, 1986).

In the Rustenburg area, the Magaliesberg Quartzites belonging to the Pretoria Group form the floor rocks to the Complex. It is proposed by Willemse (1969) that the abundance of diabase sills within the Pretoria Group probably represents an early phase of Bushveld activity. The layered, mafic rocks of the Complex are collectively known as the 'Rustenburg Layered Suite.' These mafic rocks extend into the eastern and western limbs of the Complex. Several mines exploit the resources contained within the eastern and western limbs of the Bushveld Complex (Figure 1).

Implats has mines within the western (Impala Platinum – Rustenburg operations) as well as the eastern limb (Marula Platinum) of the Bushveld Complex.

1.3. Impala Platinum Mines – Rustenburg Operations

The Impala Platinum Mines are located north of the town of Rustenburg. The mining lease area of Impala Platinum extends to a strike length of 24km, covering an area of approximately 10700 hectares (Figure 2). The regional strike of the orebody is north – northwest to south - southeast with an average dip of 9^0. Both, the mineralized Merensky and UG2 Reefs are currently being mined at Impala Platinum mines. The Merensky Reef outcrops in the southern region of the lease area with a regional dip of 9^0 in a north westerly direction.

The exclusive mining of the Merensky Reef is confined to only two (No. 12 and No. 14 Shaft areas) of the 13 operating shafts of Impala Platinum.
The present study is conducted within the No. 12 Shaft area which is situated in the northern part of the mining lease area as shown in Figure 2.

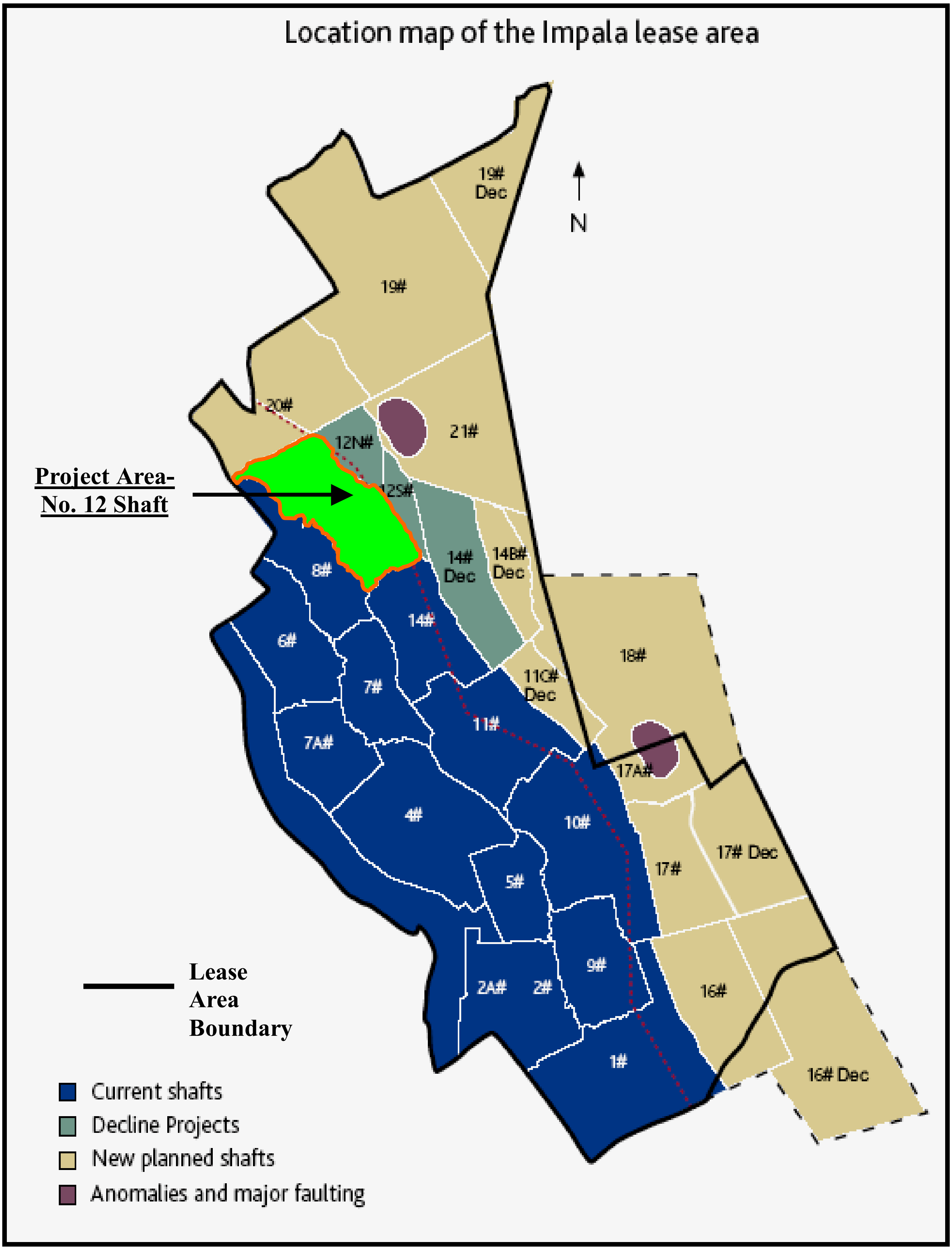


Figure 2. Location map of Impala Platinum Operations (adapted from Implats Annual Report 2004).

In terms of shaft boundaries, the No. 12 Shaft area is bounded in the west by No. 8 Shaft, in the south by No. 14 Shaft whilst the northern boundary is created with No. 20 Shaft.

1.4. Objectives of the Investigation

The principle objective of this study is to investigate the viability of exploiting the UG2 Chromitite Layer for its Platinum Group Element content in the No. 12 Shaft area. Favorable market conditions for platinum group metals were among the key factors which led to the re – evaluation of the UG2 Chromitite Layer in this area. This investigation has been further motivated by the recommendations documented in previous geological evaluation reports (Slabbert, 1994; Knoetze, 1996).

Question1.
Is it viable to mine the UG2 Reef within the No. 12 Shaft area?

This question will be addressed by an analysis of the following sub questions.
Sub question 1.a.
What structural and lithological complications could be anticipated?
Sub question 1.b
What are the estimated grade of the UG2 Reef, mineral resource and mineral reserve in the No.12 Shaft area?
Sub question 1.c.
What are the outcomes of a discounted cash flow model of the projected future mining?
Sub question 1.d.
What optimization strategies can be employed to enhance the viability of the proposed area to be mined?
Sub question 1.e.
Which aspects must be examined in the risk and sensitivity analysis?

1.5. The Delimitations

The analysis and the resultant conclusions related to grade, mineral resource and reserves, financial viability, optimization strategies, risk and sensitivity aspects will be based on market related averages and assumptions. The evaluation process will utilize generic business practices and strategies common to the mining industry and is thus not specifically related to Impala Platinum.

1.6. Assumptions

For the financial analysis of the project, all the input parameters such as metal prices, exchange rates, recovery and discount rates, working costs, capital costs and the tons to be mined were based on motivated assumptions, calculated market averages and estimated values.

1.7. Importance of the Study

The mineral reserves within the No. 12 Shaft area will be significantly increased if the findings of the investigation prove to be favorable. This in turn will create employment as well as contribute to the wealth of the company.

1.8. Disclaimer

The content of this report has been derived from basic investigative research. It is for information purposes only and is not intended to serve as financial, investment or any other type of advice. The statements, concerning the economic outlook for the platinum group metals industry and market, expectations of metal prices and production may contain forward looking views. These views involve both known and unknown risks, assumptions, uncertainties and other important factors that are not in any way related to that of Impala Platinum. All the views, opinions and conclusions expressed in this document are solely of a personal nature. All the outcomes originating from the financial assessment, grade, mineral resource / reserve estimation, risk and sensitivity analyses have been derived from the author's interpretation of the data. Furthermore the data itself and especially the input parameters, which were used in the discounted cash flow model, have been derived from assumptions and calculated market related averages and estimates. It is not absolute company data. Consequently, no assurance can be given that these will prove to be correct. Furthermore, due to the confidential nature of some of the data contained herein, access to this document will be restricted for the two year period agreed upon with the University of Pretoria.

1.9. Acknowledgements

I would like to thank Mr. J.C. Terblanche and Impala Platinum for giving me the opportunity to do this postgraduate course. Mr. J.C. Terblanche has also acted as my internal functional mentor within the company and he is thanked in this regard for all his assistance, guidance and support during the course. Special thanks also go to Schalk Kok (Senior Geologist) for his assistance with the compilation of the plans. I am grateful to Nico Strydom (Senior Accountant) for this assistance regarding the financial aspects of this dissertation. Mr. JJ Vermaak is thanked for reviewing the confidential aspects of this dissertation and the suggestions made by him. My academic supervisor, Professor H. F. J. Theart has played a major role during the course as well as assisted me in compiling the dissertation. His efforts are highly appreciated.

Chapter 2. Geological overview

2.1. Stratigraphy of the Bushveld Complex

The ultra – mafic layered rocks of the Bushveld Complex is exposed in three main limbs known as the western, northern and eastern limbs (Figure 1). The different limbs of the Bushveld Complex are elliptical in plan and combined measure approximately 450km east to west and 200km north to south with the northern limb extending some 100km further to the north. Granites, felsites (of the Transvaal Supergroup) and rocks belonging to the Karoo Sequence occupy the area between the western and eastern limbs (Lee, 1996). The complex contains layers rich in chrome, the platinum group elements and vanadium, at different stratigraphic levels.

The Bushveld Complex is stratigraphically subdivided into the Marginal Zone at the base, followed by the Lower Zone, Critical Zone, Main Zone and Upper Zone (Figure 3). The Critical Zone contains various layers of chromitite which are of economic interest for chrome and Platinum Group Elements (PGE). These are the Lower Group Chromitite Layers (LG1 TO LG7), the Middle Group Chromitite Layers (MG1 to MG4) and the Upper Group Chromitite Layers (UG1 to UG3). The PGE – enriched Merensky Pyroxenite that only contains minor chromitite stringers is developed above the Upper Group Chromitite Layers (Du Toit, 1986; Lee 1996; Vermaak, 1995; Viljoen and Schurmann, 1998.). In the northern limb where the succession is poorly developed a thick pyroxenitc unit directly overlies the floor rocks. This unit carries PGE mineralization and is known as the Platreef. At present only the UG2 Chromitite Unit, Merensky Pyroxenite and Platreef are mined for their contained metals of economic abundance.

2. 2. Stratigraphic Nomenclature within the Upper Critical Zone

Impala Platinum Mines exploit the mineralized zones contained in the Upper Critical Zone. The footwall development phases, which precede stoping operations, thus extend from above the Merensky Reef to the footwall of the UG2 Chromitite Layer. For the reasons of simplicity a local nomenclature system which describes the geological succession from above the Merensky Reef to below the UG2 Chromitite layer has been designed and is used on all the shafts (Du Toit, 1986). The naming system includes rock names such as pyroxenite, norite, anorthositic norite, spotted and mottled anorthosite as opposed to the scientific cumulate terminology (Figure 3). Du Toit (1986) has described the characteristics of each unit shown in the informal stratigraphic column. The descriptions of these units with particular reference to its characteristics within the No.12 Shaft area are attached as Appendix I.

All the stratigraphic units shown in Figure 3 can be traced from the south of the mining lease up to the extreme north. There is however a distinct thinning of the footwall layers in a northwesterly direction, which is believed to reflect a slow subsidence of the basin during the accumulation of the magmatic sediment (Du Toit, 1986).

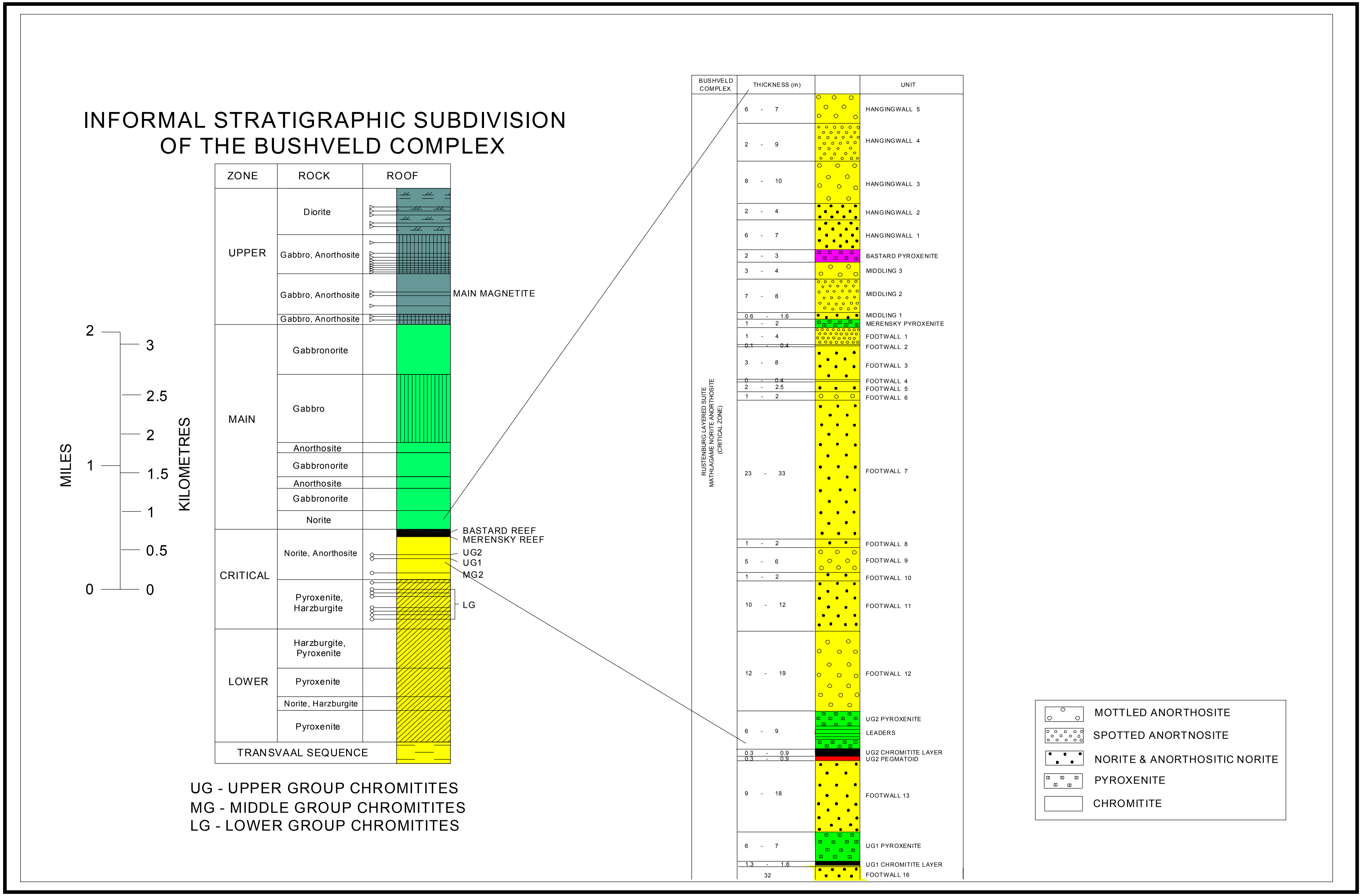


Figure 3. Informal Geological Succession (Du Toit 1986).

Chapter 3. Geological exploration

3.1. Previous work

A total of 51 exploration boreholes were drilled to the depth of the UG2 Chromitite Layer within the confines of the No.12 Shaft boundary (Slabbert, 1994). Potholed reef by definition is where the UG2 Chromitite Layer cuts through the immediate underlying UG2 pegmatoid to rest directly on an underlying footwall lithology. From personal experience, knowledge and observations on a mine wide basis, the following typical telltale signs are indicative of potholed reef:

- reef changes its dip
- thinning of the chromitite layer and footwall lithologies
- disappearance of the underlying pegmatoid.

Similarly, based on these indicators, Slabbert (1994) concluded that 45% of the boreholes comprised typical potholed reef. This geological loss when combined with other mining and rock engineering losses in turn suggested that a low global extraction rate could be anticipated for the UG2 Chromitite Layer within the No. 12 Shaft area. Knoetze (1996) quantified the potholed reef intersections according to their severity in order to investigate the possibility of mining through them. Following these investigative studies (Slabbert 1994; Knoetze 1996), the decision was made to commence trial mining of the UG2 Chromitite Layer. Only limited development was undertaken, but this was however curtailed due to the combination of depressed market conditions and the low anticipated extraction rates.

However, the previous workers (Slabbert 1994; Knoetze, 1996) have strongly recommended that the UG2 Chromitite Layer be further assessed by a more detailed evaluation drilling program.

3.2. Recent work

Due to favorable market conditions, senior management suggested that the UG2 Chromitite Layer be re – evaluated throughout the No. 12 Shaft area. This investigation has involved a re – assessment of the previously drilled boreholes as well as drilling additional evaluation boreholes from existing underground Merensky workings. This evaluation program culminated with 55 boreholes being drilled within the demarcated boundaries of the No. 12 Shaft area (Figure 5).

The average middling distance, between the UG2 Chromitite Layer and the overlying Merensky Reef within the No. 12 Shaft area, is approximately 60m. For this reason, geological structures mapped on the Merensky Reef horizon are not expected to necessarily extend down to the depth of the UG2 Chromitite Layer. An additional investigation by Hahn and Ovendale, (1994) shows that there is no evidence to support the super positioning of Merensky geological structures onto the UG2 Reef horizon. However, approximately 20% of the UG2 boreholes, which have intersected potholed reef, tend to co – incide with potholes also encountered within the overlying Merensky Reef. In such cases, it is assumed that these potholes extend to the depth of the UG2 Chromitite Layer and are thus represented in Figure 5. Other geological features (such as dykes, faults, replacement pegmatoid bodies and dunite intrusions) which are envisaged to influence the UG2 Reef horizon, have been shown as well. These features will be discussed in a subsequent section.

With the availability of widespread, representative borehole data throughout the No. 12 Shaft area, it is now possible to conduct a detailed evaluation of the UG2 Chromitite Layer. From the investigation of all the available data, it is apparent that the loss in potential mineral reserves due to geological structures and rock engineering requirements will be variable, thus resulting in different extraction rates for the individual mineral resource blocks within the No.12 Shaft area. Assigning a single global extraction rate upon which important decisions are to be made is neither entirely representative, nor accurate, but is merely used to derive estimates of the viability of mining in this area.

The UG2 Chromitite Layer within the No. 12 Shaft area will be evaluated in the manner in which mining projects are generally assessed. This evaluation comprises the following sequential phases:

(a) Information phase – during this stage, basic information regarding geological, rock engineering, marketing and governmental factors are required. Mine development alternatives are considered in this phase.
(b) Estimation phase – includes the estimation of mineral reserves, operating costs, revenue, capital expenditure and intangible items to be considered which will impact on ultimate ‘go’ or ‘no go’ decisions.
(c) Analysis phase – includes financial evaluations such as discounted cash flow methods, sensitivity and risk analysis
(d) Decision phase – is based on the outcomes of the earlier phases especially the risks and returns of the projects being considered.

Comprehensive information including scenario analyses and possible options in each of the above phases is presently beyond the scope of this particular project. However the general level of investigative research will be sufficient to make valid and justifiable conclusions.

Chapter 4. Structural, lithological and chemical complications of the UG2 Chromitite Layer

4.1. The characteristics of the UG2 Chromitite Layer within the No. 12 Shaft area.

The UG2 Reef is a chromitite layer varying in thickness from 56cm to 68cm. The stratigraphic distance between the Merensky Reef and underlying UG2 Reef decreases northwards within the lease area. Thus the middling distance which is typically about 120m at the No.1 Shaft area (located in the south – Figure 2) decreases to approximately 60m at the No. 12 Shaft area (located in the northern part of the lease area). The UG2 Reef is commonly underlain by a coarse pegmatoidal feldspathic pyroxenite. A characteristic feature in the hangingwall is the presence of 3 to 4 thinner chromitite stringers. These stringers form potential partings during mining on the igneous layer contacts and are often referred to as the 'Leader Chromitite Layers' or colloquially as the 'Triplets.' Geological features such as potholes, depressions and rolls are commonly encountered in the underground mine workings. Relative ore metal abundances for the UG2 Chromitite Layer on Impala Platinum mines are given in Table 1.

Table 1. The relative ore metal abundances in the UG2 Chromitite Layer (Implats Annual Report 2004).

Metal	Percentage %
Platinum Pt	47.4
Palladium Pd	25.7
Rhodium Rh	9.0
Ruthenium Ru	13.6
Iridium Ir	3.7
Gold Au	0.7

Within the No. 12 Shaft area, the in – situ grades of the UG2 Reef range from 3.85 g/t to 10.88 g/t as obtained from drill hole assay results (Balakrishna, 2004).

In terms of vertical grade distribution within the UG2 Reef, there are often two peaks developed in the PGE distribution (Figure 4).
The precious metals are concentrated towards the bottommost 20cm, lower values in the middle and high values close to the top (Lee 1996). This PGE distribution tends to correlate with a textural change in the chromite – silicate gangue from a granular silicate chromite to a poikilitic silicate- chromite texture, as best observed on slightly weathered exposures (Lee 1996).

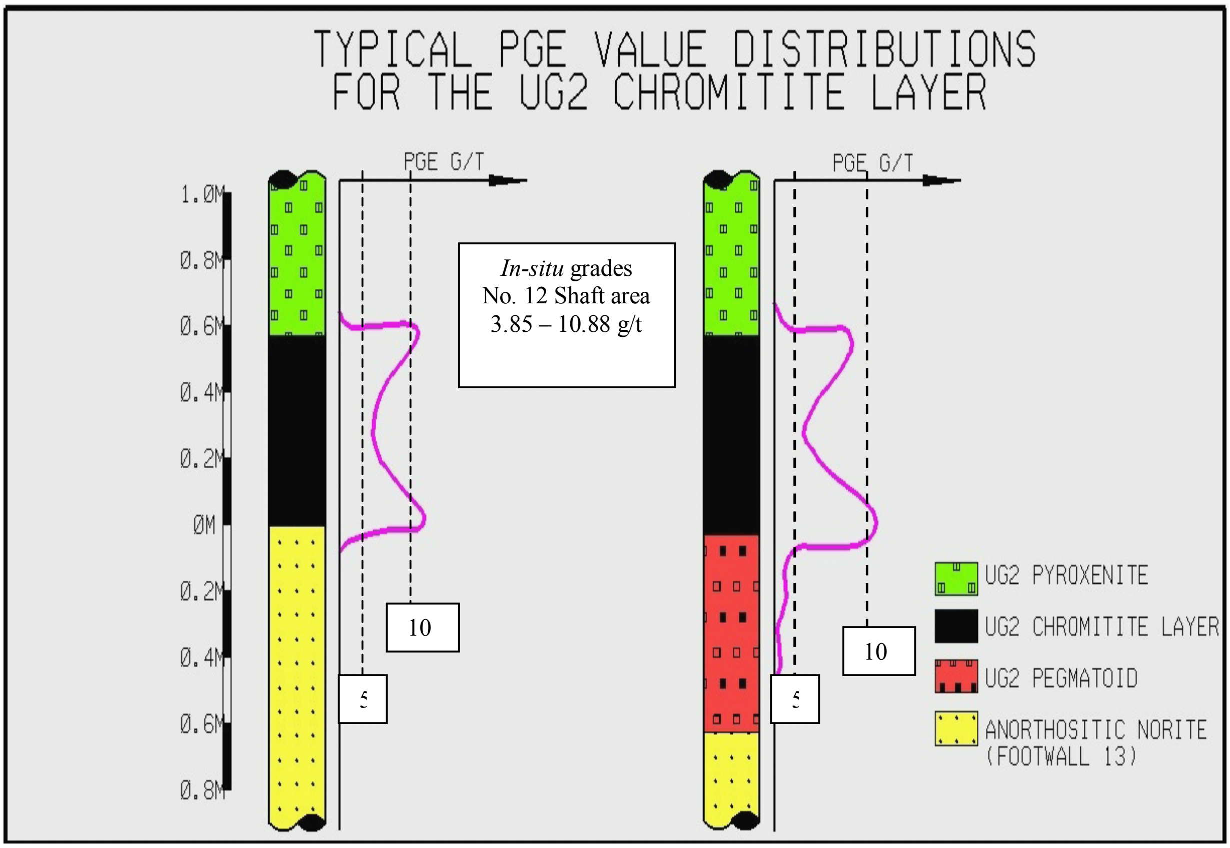


Figure 4. Grade distribution in the UG2 Reef in the No. 12 Shaft area compared with the general trend reported by Lee, (1996).

The Platinum Group Minerals (PGM's) are interstitial to the chromite grains. Laurite is commonly the only PGM, which is enclosed by chromite (Viljoen and Schurmann 1998). The grain sizes of the PGM's are small with an average diameter of 9.3µm (Lee 1996). Despite the low sulphide content, most PGM's are attached to small base metal sulphide grains and thus Platinum Group Elements may still be considered to be sulphide associated. The PGM's are known to also occur within the orthopyroxene gangue.

The major silicate gangue minerals are orthpyroxene and plagioclase (30% to 50% by volume). There are no visible coarse sulphide minerals in the UG2 Chromitite Layer. The sulphide grains are generally finer than those of the Merensky Reef. The fine sulphides have been found to give lower flotation recoveries than do the coarse fractions (McLaren, 1978).

4.2. The geological features of the UG2 Chromitite Layer within the No. 12 Shaft area

A. Structural Complications

The No. 12 Shaft block comprises a number of geological features such as faults, dykes, ultramafic replacement pegmatoid, pegmatite veins, dunite intrusions, potholes and local dip and strike changes all of which have been encountered in the underground Merensky workings. Borehole data and information obtained from underground workings of the neighboring shafts suggest that some of these geological features are likely to affect the UG2 Chromitite Layer within the No. 12 Shaft area. These anticipated structural features are shown in Figure 5.

[a] Strike

The strike direction is approximately NS at the No. 12 Shaft area, changing to NNW to the north and NW in the extreme north of the area. A change in the strike direction of the layering from NNW to NW in the extreme northern part of the area was also observed from the old Merensky workings. The strike and dip changes in the vicinity of boreholes BH12/ 02428 and BH12/ 02435A (Figure 5) are related to the presence of a dunite body. This dunite body has extended down into the UG2 stopes of the No. 8 Shaft area. Hence it is envisaged that this feature will affect the UG2 Chromitite Layer within the No. 12 Shaft area as well.

[b] Dip

The average dip of the UG2 Reef is 9^0 whilst relatively flat dips of approximately 6^0 can be expected in the vicinity of borehole BH12/ 09349. The flat dips will result in long (350m – 400m) back lengths and will also affect advance strike gullies (ASG) directions and will limit the success of mining potholed/rolling reef. The shallow dips (1^0 to $2^{0)}$ and reverse dips have been encountered on the Merensky Reef in the upper northern portion of the No. 12 Shaft area in the vicinity of the dunite intrusion. These shallow dips are expected to be encountered on the UG2 Reef horizon as well.

[c] Faults

The major faults expected to be intersected on the UG2 horizon are:

- 3 distinct approximately east - west trending fault zones lying to the northern portion of No. 12 Shaft block. Their displacements are variable and range from 3m to 10m and are known to splay outwards thus resulting in a varied thickness of the fault zone itself. Hence these faults when encountered are deemed a mining limit in terms of stoping operations.
- strike faults are generally steeply dipping and are associated with lamprophyre dykes. Changes in displacement and direction of throw along strike are known to occur. However these faults are effectively negotiated with minor mining problems.
- a prominent well defined northeast – southwest striking fault has been intersected in the Merensky workings with its effect being most intensely felt in the southwestern block. In this region of the shaft area, the fault tends to splay outwards resulting in sympathetic faulting, shearing and jointing. Throws are highly variable and range from 1.5m to as much as 25m.

[d] Dykes and Sills

These occur as later intrusions into the layered succession. The dykes are, by definition, near vertical in dip whilst sills are relatively flat lying. Two types of dykes namely Dolerite and Lamprophyre Dykes occur within the No. 12 Shaft area.

Dolerite Dykes

These are grey to almost black, hard and generally competent in nature. Two prominent vertical dipping and approximately north – south striking dolerite dykes between 15m and 30m thick have been mapped.

Historically these dykes have been relatively competent and are not expected to impact on the mining of the UG2 Reef. However problems may arise where these dykes are accompanied by shearing or faulting. Changes in the dip of the minor dolerite dykes (0.15m – 1.2m wide) are known to occur over short distances, resulting in dolerite sills being encountered in the hangingwall to the reef. Such cases are followed by changes in the mine layouts, motivated by recommendations from the rock engineers.

Lamprophyre dykes
These intrusive features are commonly encountered in the underground workings of the Merensky Reef and the same is expected to occur in the case of the UG2 Chromitite Layer as was observed in the drill core. Lamprophyre dykes are mica rich, shiny brown, fine to medium grained dykes ranging in thickness from 0.1m to 4m. These dykes are soft and incompetent especially when associated with faults and shear zones. The incompetent nature of these dykes (due to high mica content) causes potentially dangerous ground conditions.

[e] Pegmatoidal Veins
Pegmatite veins occur as white, coarse grained, sub – vertical and often laterally discontinuous bodies of variable thickness. Generally these features are not problematic; however the concentration of coarse grained mica on the edges of these veins can lead to localized ground problems. Pegmatite veins are omnipresent features.

[f] Dunite Intrusion
The full extent and exact shape of the intrusion occurring nearby the boundary between the No. 8 and No. 12 Shaft areas is not yet known. Prospect drilling from the No. 8 Shaft area indicates an irregular shape and the possibility of more than one plug – like intrusion. A high degree of serpentisation has occurred within the No. 8 Shaft area which is located updip of the dunite body. Blocky ground and poor roof competency can be expected in areas surrounding the dunite pipe.

[g] Ultramafic Replacement Pegmatoid Bodies

This is the name given to rock types which have 'replaced' the original rock. There are two types of replacement pegmatoid namely feldspathic replacement pegmatoid and ultramafic replacement pegmatoid which occur in the No. 12 Shaft area. The ultramafic replacement pegmatoid is most common and typically occurs as a dense, very coarse grained, almost black rock, made up predominately of large, very dark iron rich pyroxenes with minor interstitial white feldspars. This type is commonly magnetite rich.
Feldspathic replacement pegmatoid are anorthositic, consisting predominately of very coarse anorthite crystals together with occasional large pyroxene crystals.
Replacement Pegmatoid occurs as highly irregular bodies of unpredictable shape and size and presents a problem in that it obliterates the original strata and marker horizons. These bodies tend to affect the anorthositic horizons (Footwalls 12, 13 and 16) and disrupt the chromitite layers to a lesser extent.
Despite these replacement pegmatoid bodies being relatively small in size and scarce in distribution, they do however influence the mining operations.
Some of the UG2 exploratory boreholes have intersected replacement pegmatoid bodies. It is therefore expected that these phenomena will affect the UG2 Reef horizon by way of replacement resulting in obliteration of the grade and creating poor ground conditions.

[h] Potholes

Geological features such as potholes, depressions and rolls are commonly encountered in underground mine workings. Potholes result in the slumping of the reef and hangingwall rocks to a level below the current elevation of the stoping operations. Thus when intersected, these phenomena often mark the end of conventional stoping methods and additional development is required to expose the reef horizon.

In general, potholes are assumed to be circular in shape and are known to vary from a few meters to several hundreds of meters in diameter (Hahn et al, 1994). However, due to the coalescing of potholes, these slump structures become composite with an irregular shape. The hangingwall rocks tend to thicken due to a progressive infilling of the pothole whilst the reef and underlying footwalls thin out.

With regards to the origin of potholes, Schmidt (1952) and Ferguson et al (1963) have suggested that potholes have formed from strong eddying currents and the scouring action of early formed pyroxene crystals on the floor rocks.
This theory proposes that the footwall rocks had to be in a semi- solid state in order to be affected by the eddy currents. However evidence such as ‘knife – edge’ structures seen between adjacent potholes in the underground workings of Impala Platinum casts some doubt on the fact that the eddying currents were exclusively responsible for the final shaping of the potholes. It is believed that these currents are the result of a primary force and is to a lesser extent responsible for the formation of the potholes.

Cousins (1969), proposes that a density unstable condition arises when a relatively heavy pyroxenite layer overlies a lighter, partly consolidated and volatile – rich footwall layer. Consequently, potholes are expected at all horizons where these conditions prevail and in the case of the underground workings these would typically be at the base of the UG1 and UG2 Units, Merensky Unit and the Bastard Unit. The lighter footwall rocks become buoyant, re – melt and are assimilated into the anorthositic magma of the upper portion of the preceding unit. It is believed that the slumping action gives rise to a vortex which partly aids in shaping the pothole.

Potholes display varied geometries and physical dimensions and will thus constitute geological losses of varying proportions across the lease area (Monei 2004).

The potholes occurring in the UG2 Chromitite Layer are not mined due to the following reasons:

- slumping creates unstable, dangerous hangingwall domes due to the presence of the overlying chromitite leaders. These triplets are potential fracture planes which result in an unsafe mining environment.
- thinning of the reef in the vicinity of potholes tends to make the ore very fine grained. This fine grained PGM mineral grains show a preference to associate with silicate gangue minerals which makes it difficult to extract by flotation. This results in lower extraction rates (McLaren, 1978).
- the UG2 Chromitite Layer decreases significantly in width or is totally absent. Additional barren footwall and/or hangingwall have to be mined in order to create a

practical mine stope width. This results in excessive dilution and thus becomes uneconomical to mine.
- potholing influences the grade distribution in that it shears away the enriched contacts of the UG2 Chromitite Layer as well as the underlying mineralized pegmatoid.

A critical analysis of the potholes affecting the UG2 Chromitite Layer in the No. 12 Shaft area, as determined from borehole information will be discussed in Chapter 8.

B. Lithological Complications

Apart from the generally sub vertical planes of weakness created by fractures, fissures, faults and dykes, certain features within the succession are expected to present themselves as problems.

- **Stratigraphic features influencing footwall development**

There are several undesirable lithological features found in the footwall succession below the UG2 Chromitite Layer such as:

[a] The chromitite stringers of F/W 16 are irregular and vary in thickness from a few millimeters to several centimeters.
The chromitite stringers are potential fracture planes resulting in poor ground conditions especially when present in the immediate hangingwall of the footwall drive excavations.

[b] If the barren UG1 Chromitite Layer is present very close to the hangingwall of the footwall excavations, then unstable roof conditions are known to prevail.

[c] The contact between the Footwall 13 norite and UG1 Pyroxenite is marked by a 0.5cm chromitite layer. This contact acts as a potential parting plane along which the 2 lithologies can easily separate resulting in rockfalls.

[d] Footwall 13 norite is known to occasionally contain anorthositic portions, which could result in localized scaling of the rocks.

[e] The UG2 Chromitite Layer itself maybe potentially dangerous especially when it is not fully exposed but left partially in the immediate hangingwall. This could result in the collapse of the hangingwall.

- **Stratigraphic features expected to influence the extraction of the UG2 Chromitite Layer.**

The hangingwall chromitite layers (Leader Chromitite Layers and Intermediate Chromitite Layers)

These units of very low cohesion create particularly poor roof stability conditions especially in instances where the distance between the UG2 Chromitite Layer and the overlying Leader Chromitite Layers/ Intermediate Chromitite Layer (ICL) is less than

0.5m. The Leader Chromitite Layers are generally very poorly mineralized and are only mined out together with the UG2 Chromitite Layer in cases when it becomes a safety hazard.

Within the No. 12 Shaft area, the average middling distance between the upper contact of the UG2 Chromitite Layer and overlying chromitite layers is found to be greater than 1.8m. A thin chromitite stringer typifies the contact between the Footwall 12 and the underlying UG2 Pyroxenite. This chromitite stringer is regarded as a plane of weakness, but would not necessarily affect the extraction of the UG2 Reef.

C. General mineralogical and mineral chemical features which could affect Platinum Group Element recoveries.

The effectiveness of recovery of the metals depends largely on the grain size. Fine-grained mineral grains, due to their affinity to and inclusion within gangue minerals, negatively affect flotation processes The UG2 Reef is associated with grain sizes of <10μ and thus requires intense milling in order to achieve good mineral liberation. However, intense milling also results in the chromite becoming entrained in bubbles of the flotation agents and ends up in the concentrate. Furthermore, if the UG2 Reef is milled intensively, then the derived chromite forms a problematic freeze layer in the mill (Pahla, 1997).

Portions of the UG2 orebody displaying an even smaller grain size of PGM's than the average of the entire orebody normally also have poorer recovery results.

Oxidized material is also problematic to handle in the flotation process and contributes significantly to low recovery rates. The oxidized material requires more milling to remove the outer coating. This in turn means more costs and finer resultant grains. These extremely fine PGM's tend to be lost in the slurry.

Figure 5. Geological structural features affecting the UG2 Reef.

Chapter 5. Common PGM assay techniques

5.1. Preconcentration by Fire Assay

This family of procedures is the most reliable method for accurately measuring precious metals in materials ranging from high purity bullion to parts per billion 'pathfinders' (Chan et al 2001).

The samples are melted (fired); reactions in the molten solution gather precious metals into a collector such as lead. Usually additional steps separate and purify the precious metals from this base collector into a purer form for quantitative measurement.

The fire assay technique is over 2000 years old and is considered to be the most accurate method for the separation of the precious metals (platinum group metals, silver and gold) from the gangue minerals and pre – concentration of the precious metals prior to analysis by instrumental techniques such as inductively coupled plasma mass spectrometry (ICP-MS), inductively coupled plasma optical emission spectrometry (ICP-OES) and graphite furnace atomic absorption spectrometry (GF-AAS).

A detailed description of the Fire Assay method given by Chan et al (2001) is attached in Appendix II.

There are 2 fire assay techniques (Sundquist, 2001) namely:

(A) Fire Assay Lead Collection Technique – 4E

- is used to determine the values for Pt, Pd, Rh and Au. This technique is often referred to as the 3PGE + Au (4E) analysis (Sundquist, 2001).
- this is not an absolute method of analysis. Values produced are dependent on:
- a unique method and procedure that is being practiced at a particular laboratory
- the sample type.

(B) Fire Assay Nickel Sulphide Collection Technique – 6E

- this technique involves the pre-concentration of the noble metals followed by the spectrometric measurement of the individual metals (Sundquist, 2001). Hence this method derives absolute values for the metals Pt, Pd, Rh, Ru, Ir and Au.
- this technique is often referred to as the 5PGE + Au (6E) analysis.

A 'fire assay correction factor (FACF)' shows the relationship between these two assay techniques (Sundquist, 2001). The FACF can be determined by assaying a large population of samples by both techniques and then creating a database from which the ratio between the two methods can be calculated i.e. 6E/4E = FACF.
On Impala Platinum Limited, the FACF for the UG2 Chromitite Layer in theory could be calculated as the percentage of Ru + Ir = 17.3% (from Table 1). Thus a crude FACF which is to be used in order to convert a 3PGE + Au to a 5PGE + Au analysis would be 1.173.

General concerns regarding fire assay methods

It has been found that the prill weights obtained are unique to the laboratory at which the fire assay technique is being carried out (Chan et al, 2001). Despite the fact that the flux composition may be the same with the same furnace temperatures and times, variations occur due to poor handling practices and variable furnace conditions. It is important that reducing conditions prevail during the fusion stage; otherwise the PGMs can oxidize and be lost to the slag.

At the cupellation stage, although the lead is oxidized, there is little danger of losses of Pt and Pd whilst in the metallic form. There can however be losses of Rh and Au and usually complete loss of Ru and Ir.

Chapter 6. Grade estimation of the UG2 Chromitite Layer within the No. 12 Shaft area

6.1. Analyses of borehole data

All 55 boreholes and their relevant data are listed in Table 2. The grades shown are in – situ reef channel grades (excluding diluting factors). These boreholes were assayed using the 4E (Pt, Pd, Rh + Au) analysis method. The collar positions of these exploration boreholes are shown in Figure 5.

Table 2. Geological Borehole Data - UG2 Chromitite Layer.

Borehole Number	Channel Width cm	Channel Grade 3PGE + Au (g/t)	Presence of Pegmatoid/Other Comments
BH 12/09331	72.0	6.63	Yes
BH 12/09330	62.0	7.65	Yes
BH 12/09041	65.0	5.60	Yes
BH 12/09043	69.0	5.90	Yes
BH 12/09045	69.0	5.80	Yes
BH 12/09046	64.0	7.91	Yes
BH 12/09334	59.0	6.03	Yes
BH 12/09050	58.0	7.79	Yes
BH 12/09052	72.0	8.90	Yes
BH 12/09053	51.0	6.90	No – potholed
BH 12/09349	68.0	0.54	No – potholed
BH 12/09336	47.0	5.04	Yes
BH 12/09509	48.0	7.26	Yes
BH 12/01724	62.0	7.94	No – potholed
BH 12/01725	54.0	5.94	Yes
BH 12/01727	59.0	8.11	Yes
BH 12/01728	60.8	10.54	Yes
BH 12/01730	54.0	5.00	No – potholed
BH 12/01731	78.6	6.60	No – potholed
BH 12/01733	69.3	8.04	Yes
BH 12/01734	72.0	7.65	Yes
BH 12/01735	70.5	6.88	Yes
BH 12/09345	62.0	5.85	Yes
BH 12/01737	78.9	8.85	Yes
BH 12/01738	69.6	9.05	No – potholed
BH 12/01739	58.2	7.32	Yes
BH 12/09507	113	9.98	No – potholed
BH 12/01742	63.5	6.85	No – potholed
BH 12/01743	67.0	8.34	No – potholed

Borehole Number	Channel Width cm	Channel Grade 3PGE + Au (g/t)	Presence of Pegmatoid/Comments
BH 12/01744	80.3	6.95	No – potholed
BH 12/01740	115.0	10.88	No – potholed
BH 12/01746	52.9	7.52	No – potholed
BH 12/01747	60.6	7.31	Yes
BH 12/02428	73.1	6.61	Yes
BH 12/02429	69.2	4.90	No – potholed
BH 12/09526	51.0	8.39	Yes
BH 12/02430	107.0	4.75	No – potholed
BH 12/02432	65.9	6.90	Yes
BH 12/02433	57.7	7.19	Yes
BH 12/02434	68.9	8.80	Yes
BH 12/09506	71.0	6.13	No – potholed
BH12/0 2435A	58.9	9.30	No – potholed
BH 12/02436	51.6	8.99	No – potholed
BH 12/02437	65.2	8.50	Yes
BH 12/02438	64.4	7.90	Yes
BH 12/09042	100.0	6.80	No – potholed
BH 12/09051	-	Not assayed	Potholed, no reef intersected
BH 12/02440	56.7	8.61	Yes
BH 12/02441	36.0	7.00	Yes
BH 12/02442	57.9	9.31	No – potholed
BH 12/02443	78.8	5.81	No – potholed
BH 12/09340	34.0	3.85	No – potholed
BH 12/02445	48.0	6.40	No – potholed
BH 12/09343	46	Not assayed	Drilled in 2001
BH 12/02446	62.0	7.10	No – potholed

6.2. Geostatistics

Average/Mean g = 1/n $\Sigma g_{I,}$
Where: g is the average value of the samples
n is the number of samples
gi is the measured value on sample i
Σ is the total sum of individual measured values

Thus the **arithmetic mean *in – situ* grade of the orebody:** (1/53) (387.88)
= 7.32 g/t over 66cm (channel width)
*** based on 3PGE + Au**

Standard Deviation S $= \sqrt{\Sigma (X_j - X)^2 / N}$
$= \sqrt{\Sigma (X - X)^2 / N}$
$= \sqrt{\Sigma x^2 / N}$
$= \sqrt{160.19/53}$
$= 1.74$

where x represents the deviations of each of the numbers X_j from the mean X.

Variance V = S^2
= 3.03

However, the arithmetic mean (without an additive constant) is only valid as calculated above, if the distribution of the samples is normal. In order to determine the nature of the distribution, it is necessary to establish a probability plot for the sample data.

One (1) sample has a value less than 1.5g/t. This represents 1.89% (1/53) and a proportion of 0.02 (1.89/100) of the total sample population. Two samples have values in the range 1.5g/t to 4g/t. This represents 3.77% and a proportion of 0.04 of the sample population. Similarly all the samples have been quantified according to representative percentiles and proportions of the total sample population (Table 3). The data in Table 3 has been used to establish the probability plot shown in Figure 6.

Table 3. Distribution of borehole data.

Value g/t	Number of samples	% of samples	Proportion
1.5	1	1.89	0.02
4	2	3.77	0.04
4.5	2	3.77	0.04
5	4	16.0	0.16
5.5	6	11.32	0.11
6	12	22.64	0.23
6.5	15	28.30	0.28
7	24	45.28	0.45
7.5	30	56.60	0.57
8	37	69.81	0.70
8.5	41	77.36	0.77
9	47	88.68	0.89
9.5	50	94.34	0.94
10	51	96.23	0.96
10.5	52	98.11	0.98
11	53	100	1

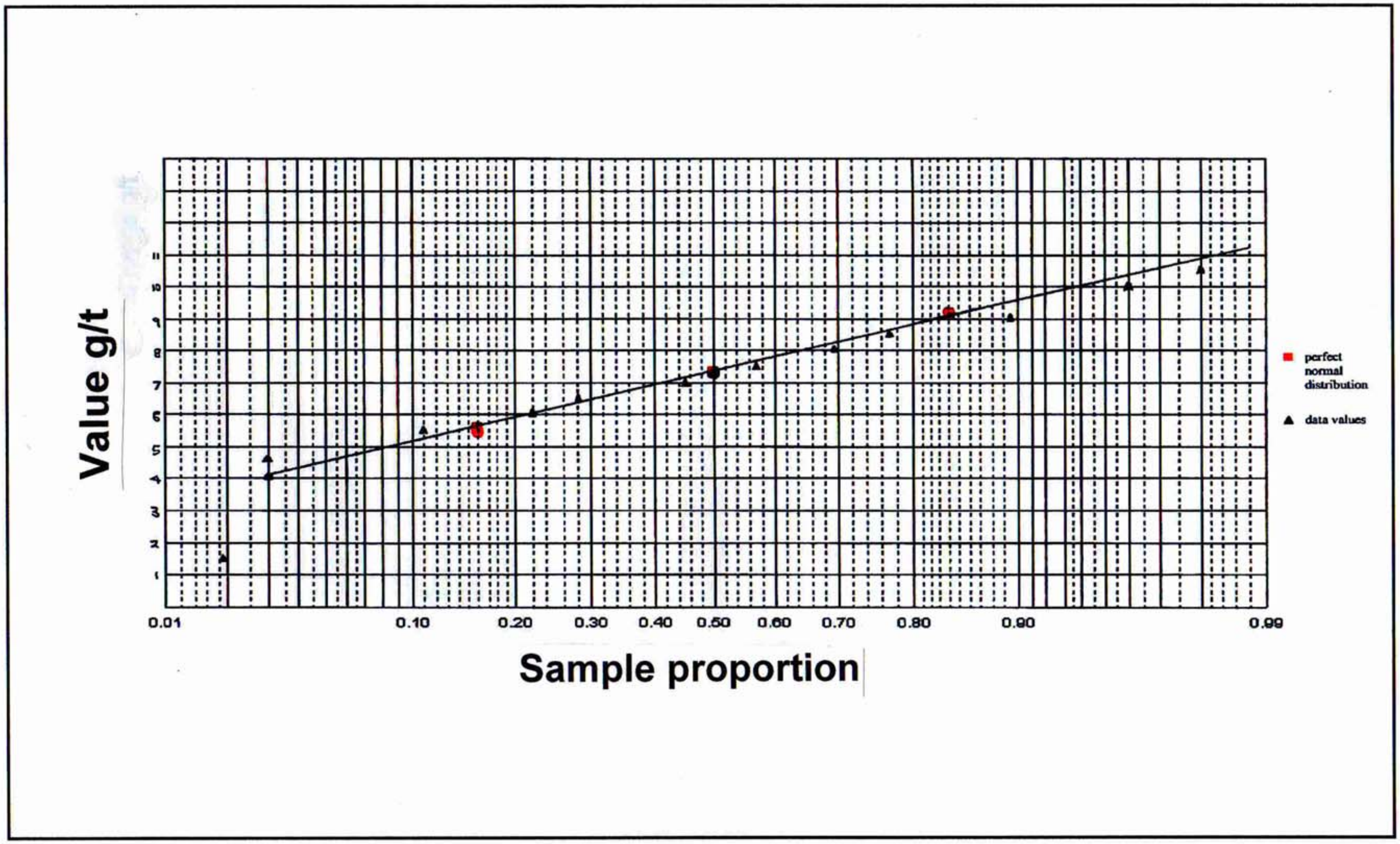


Figure 6. Probability plot for borehole distribution grades.

The straight line in this plot is the perfect normal distribution based on the following 3 points:

- 50% of the population lies below the mean which gives one point on the line at 7.32g/t versus 0.5 proportion

68% of the population lies between the 'mean minus one standard deviation' and the 'mean plus one standard deviation'.

- That is, 16% of the population lies below mean minus one standard deviation
  7.32 – 1.74 = 5.58
- 84% of the population lies below mean plus one standard deviation
  7.32 + 1.74 = 9.06

Most of the data values of the sample population tend to plot on or very close to the perfect normal distribution line with the centre of the sample distribution co- inciding with the normal distribution. Therefore it can be concluded that the distribution of the sample population is normal. However, from Figure 6, it is noted that the higher grades are not quite as high as would be expected whilst the lower grades are a bit higher than the ideal. Furthermore, only 53 samples are available from a very large population, thus it is not expected for these data values to lie exactly on the line. The more samples present, the closer it would conform to the ideal line (Clark, 2000).

Median = middle value = **7.26**
Range: is calculated by simply determining the difference between the highest and lowest values. However it has been confirmed that this is not a very reliable, stable statistic because it is subjective to the number of samples. Instead the range will be calculated using the "inter – quartile range" of the values i.e. the range of the central 50% of the sample data.
The lowest 25% and highest 25% sample values will be eliminated and the range will be calculated from the middle 50%.
25% of 53 samples = 13.25 samples, which approximates to 13 samples
Eliminating the lowest 13 samples and highest 13 samples gives a range from 6.13 g/t to 8.34 g/t. Thus the **middle 50% of the sample values lie within a range of 2.21 g/t.**

Confidence Interval Estimates for the Mean *In –situ* Grade (3PGE + Au)

@ 90% confidence level:

$$\mathbf{g - (1.645)\ S/\sqrt{n} < \mu < g + (1.645)\ S/\sqrt{n}}$$
$$7.32 - (1.645)\ 1.74/\sqrt{53} < \mu < 7.32 + (1.645)\ 1.74/\sqrt{53}$$
$$7.32 - 0.39 < \mu < 7.32 + 0.39$$
$$\boxed{\mathbf{6.93\ g/t < \mu < 7.71\ g/t}}$$

Thus it can be concluded that it is 90% probable that the mean grade of the population lies between 6.93 g/t and 7.71g/t. This is equivalent to saying that we are 90% confident that the true mean of the population lies between 6.93 g/t and 7.71g/t.

Impala Platinum quotes its grade based on 5PGE + Au (Implats Annual Report 2004). Therefore for the purposes of benchmarking and other mine wide comparisons, the 3PGE + Au analysis has to be converted to the 5PGE + Au analysis using the FACF (calculated in Chapter 5).
Thus 7.32 g/t (4E) * 1.173 = 8.59 g/t based on 5PGE + Au.

6.3. Dilution factors

However, due to dilution being introduced by various aspects, the average mill grade will be lower than the average in –situ grade. Dilution and lower grades are the result of several external factors such as: increased stope width (advanced strike gullies), off reef mining, uneconomic mining such as not extracting the mineralized contacts of the UG2 Chromitite Layer and off reef excavations in the footwalls and /or hangingwalls.

Thus the total stope width = channel + basal Pegmatoid (F/W 13) + external parameters
= 66cm + 14cm + 20cm
= 100cm, which compares favorably to mine wide observations

It is known that economical grades are confined to the reef channel whilst the footwall and hanging wall display sub- economic grades values (assumed to be 0.1g/t for calculation purposes).

Average cm.g/t: 567 cm.g/t (8.59 g/t over 66cm)
Additional cm.g/t due to increased stope width: = 3.4 cm.g/t (0.1g/t over 34cm)
Thus total cm.g/t = 567 + 3.4, which is approximately 570cm.g/t
New Grade over 100cm stope width = [570cm.g/t]/100cm = 5.70 g/t, which is 2.89g/t (8.59g/t – 5.70g/t) less than the average grade when the stope width is 66cm.
It is apparent that increasing the stope width does not promote profitable mining but rather contributes to dilution of the economic channel. Hence for mathematical reasons it could rather be denoted as –2.89g/t.

Therefore in terms of dilution, the average percentage loss is estimated at 51% (2.89/5.70 * 100%). Applying this value as the constant universal dilution, new grades (Table 4) can be calculated for each of the boreholes listed in Table 2.

Table 4. Borehole grades (5PGE + Au) of the UG2 Chromitite Layer – dilution included.

Borehole Number	**51% Dilution** – at 100cm stope width g/t	**Final grade** – inclusive of 51% dilution g/t
BH 12/09331	-3.97	3.81
BH 12/09330	- 4.58	4.40
BH 12/09041	- 3.35	3.22
BH 12/09043	- 3.53	3.39
BH 12/09045	- 3.47	3.33
BH 12/09046	- 4.73	4.55
BH 12/09334	- 3.61	3.47
BH 12/09050	- 4.66	4.48
BH 12/09052	- 5.32	5.12

Borehole Number	**51% Dilution** – at 100cm stope width g/t	**Final grade** – inclusive of 51% dilution g/t
BH 12/09053	- 4.13	3.97
BH 12/09349	- 0.32	0.31
BH 12/09336	- 3.02	2.90
BH 12/09509	- 4.34	4.17
BH 12/01724	- 4.75	4.56
BH 12/01725	- 3.55	3.41
BH 12/01727	- 4.85	4.66
BH 12/01728	- 6.31	6.06
BH 12/01730	- 2.99	2.87
BH 12/01731	- 3.95	3.79
BH 12/01733	- 4.81	4.62
BH 12/01734	- 4.58	4.40
BH 12/09345	- 3.50	3.36
BH 12/01735	- 4.12	3.95
BH 12/01737	- 5.29	5.09
BH 12/01738	- 5.41	5.20
BH 12/01739	- 4.38	4.21
BH 12/09507	- 5.97	5.74
BH 12/01742	- 4.10	3.94
BH 12/01743	- 4.99	4.79
BH 12/01744	- 4.16	3.99
BH 12/01740	- 6.51	6.25
BH 12/01746	- 4.50	4.32
BH 12/01747	- 4.37	4.20
BH 12/02428	- 3.95	3.80
BH 12/02429	- 2.93	2.82
BH 12/09526	- 5.02	4.82
BH 12/02430	- 2.84	2.73
BH 12/02432	- 4.13	3.97
BH 12/02433	- 4.30	4.13
BH 12/02434	- 5.26	5.06
BH 12/09506	- 3.67	3.52
BH12/0 2435A	- 5.56	5.35
BH 12/02436	- 5.38	5.17
BH 12/02437	- 5.08	4.89
BH 12/02438	- 4.73	4.54
BH 12/09042	- 4.07	3.91
BH 12/02440	- 5.15	4.95
BH 12/02441	- 4.19	4.02
BH 12/02442	- 5.57	5.35

Borehole Number	**51% Dilution** – at 100cm stope width g/t	**Final grade** – inclusive of 51% dilution g/t
BH 12/02443	- 3.48	3.34
BH 12/09340	- 2.30	2.21
BH 12/02445	- 3.83	3.68
BH 12/02446	- 4.25	4.08
Average mineral reserve grade (5PGE + Au)		**4.13g/t over 100cm**

From Table 4, it is concluded that the average borehole grade, inclusive of 51% dilution is approximately equal to 4.13 g/t over a mill stope width of 100cm. It will also be assumed that 4.13 g/t is the mill grade which is to be used in the financial calculations.

Chapter 7. The UG2 Reef mineral resource and reserve

The mineral reserve estimation process is shown in Figure 7. It is an iterative process that is initiated by exploration and data collection followed by geological interpretation as part of the mineral resource estimation. Other non – resource inputs or modifying factors are then considered as the mineral reserve is estimated. After the operation commences the estimates are managed and modified by the activities of grade control and reconciliation.

Iterations of Technical, Financial and Risk Assessments

	Step 1	**Step 2**	**Step 3**	**Step 4**	**Step 5**	**Step 6**
Activity	Exploration &Data Collection The Resource Database	Geological Interpretation & Modeling	Statistical Analysis Mineral Resource Estimation Classification & Reporting	Modifying Factors -mining -metallurgy -financial -marketing -legal, environmental social and governmental	Risk & Sensitivity Awareness, Mineral Reserve Estimation, Classification & Reporting	Maximize Mineral Resources & Reserves Grade Control Production & Reconciliation

Figure 7. The mineral reserve estimation process, as applied at Impala Platinum.

The South African Mineral Resource Committee has devised a code, which was accepted by the Johannesburg Stock Exchange, which must be adhered to in all public reports discussing mineral resources and/or mineral reserves. The relevant sections and definitions (as per the SAMREC Code) which are applicable to this project are attached as Appendix III.

7.1. The estimated UG2 mineral resource within the No. 12 Shaft area.

The entire mineral resource must be evaluated in terms of several parameters prior to it being classified as a mineral reserve. In this respect, all the geologically related factors, which influence the asset in the ground, must be taken into consideration. Generally, this results in a loss of ground due to structural features which act as barriers to the mining operations and thus ultimately define stoping limitations.

The Dip Factor

This factor is the inverse of the cosine of the dip of the reef horizon in degrees (1/cosine X, where X = dip of reef).

The borehole reef intersections were used to contour the UG2 Reef horizon (Figure 5). Although there are some localized areas which show variability in dip, in general an estimated regional dip of 9^0 is prevalent. This dip of 9^0 has been used in all the mineral resource and reserve calculations. This translates to a dip correction factor of 1.0125.

Geological Losses

The faults and shear zones, major dolerite dykes, replacement pegmatoid bodies and potholes which are expected to be encountered on the UG2 Chromitite Layer have been described earlier in this report. It is envisaged that all of these features will contribute towards geological losses. Estimates of these losses are listed in Table 6.

Rock Engineering Losses

The loss of ground attributed to rock engineering constraints stems from mine design, unfavorable ground conditions and legal requirements. The total direct rock engineering losses, takes several forms such as:

- bracket pillars
- shaft boundary pillars
- shaft protection pillars
- barrier pillars
- grid pillars

Except for the grid pillars that are fixed in terms of size and dimensions, all the other pillars are negotiable and can often be re – modeled when new information regarding stress regimes is obtained (Budavari, 1983). The definitions of these pillars are attached in Appendix IV and the estimates of these resultant losses are provided in Table 5.

The Extraction Factors

The extraction factors provide estimates of what is considered recoverable from the ground after the losses attributed to geological features, mine design and rock engineering requirements. The extraction factors are expressed as percentages and vary according to the geological characteristics of the individual mineral reserve blocks.

7.2. Compliance with the South African Code for Reporting of Mineral Resources and Mineral Reserves - SAMREC

Table 5 is a list of criteria, which was designed by SAMREC for preparing reports on Exploration Results, Mineral Resources and Reserves. This checklist was used to assess adherence to procedures as required by SAMREC.

Table 5. Sampling Techniques and Data (SAMREC, 2000).

Criteria	**Data related to sampling techniques and procedures**
Drilling Techniques	Drill type was core, with a diameter of 40mm or more. Holes were drilled through to the underlying Footwall 13 norite to ensure maximum representative sample recovery.
Logging	Samples were logged to a level of detail to support geological mineral reserve estimation, rock engineering, mining and metallurgical studies.
Drill sample recovery	No visible core losses were observed within the UG2 Chromitite Layer. Core recovery is assumed to be 100%.
Sub sampling techniques and sample preparation	The entire core was sent for assaying. Composite samples were crushed using a laboratory jaw crusher, blended in a cone blender and split on a rotary splitter. Samples were wet milled at a pulp density of 50% solids for 45minutes in a rubber lined rod mill.

Assay data and lab investigations	The samples were assayed using the 6-hour PGM fire assay method. A database comprising prill splits and associated accounting metals (determined from intensive testing – NiS collection) is used to calculate the percentage recovery for each metal present in the prill.
Data density and distribution	Approximately 3 to 5 holes were drilled in a 500m by 500m grid for a total area of 8.14km^2. Considering the uniformity of the deposit in this region, this density and distribution is considered sufficient to establish geological and grade continuity. Prior to analysis, physical sample compositing was applied within individual drill holes.
Audits and reviews	Audits are generally conducted internally whilst external audits are conducted by SRK Pty. Ltd and Snowden where deemed necessary.

Adhering to the procedures and principles as prescribed by SAMREC (2000), the estimated UG2 mineral resource within the No. 12 Shaft area has been calculated as shown below.

Table 6. Estimated UG2 Mineral Resource.

	UG2 Chromitite Layer			
	Designation	**Derivation**	**Inclined Area (m²)**	**Loss/Pillar %**
A	No 12 Shaft	Enclosed Area	**9578411**	-
B		Major Faults	342356	3.6
	Geological	Dolerite Dykes	146193	1.5
	Losses	Replacement Peg.	6645	0.1
		Dunite Bodies	939	0.9
	Sub-Total		**577133**	**6.0**
C	Rock	Major Faults/Dykes	503157	5.3
	Engineering			
	'Geology'			
	Sub-Total		**503157**	**5.3**
D	Rock	Boundary	186141	1.9
	Engineering	Shaft	24934	0.3
	Pillars	Cross-Cut	47398	0.5
		Barrier	462504	4.8
	Sub-Total		**720977**	**7.5**
E	**Sub-Total (B + C + D)**		**1801267**	**18.8**
F	**Available Ground (A - E)**		**7777144**	**81.2**
G	Rock Eng.	Grid Pillars	659172	8.5
H	Geological	Potholes	2014280	21.03
I	**Total Loss/Pillars (E + G + H)**		**4474719**	**46.72**
Estimated Mineral Resource (A - I)			**5 103 692 m²**	**Extraction rate: 53.28%**

7.3. The estimated UG2 mineral reserve

The applicable densities are as follows:

UG2 Chromitite Layer: (66cm) (4.04) is 2.66.64
Pegmatoid: (14cm) (3.28) is 45.92
F/W 13: (20cm) (2.94) is 58.8
This suggests that 100cm equates to 371.36

Thus the weighted average density equates to 3.71

Therefore an estimated mineral resource of 5103692m^2 will approximately equate to an estimated mineral resource of **18 934 697 tons** (5103692 m^2 * 1.0m * 3.71).

According to the definitions concerning mineral resources and reserves (Appendix III) as provided by SAMREC (2000), this estimated mineral resource can be classified as an indicated resource because tonnage, densities, shape, physical characteristics, grade and mineral content has been estimated with a reasonable level of confidence. Furthermore, these parameters have been based on sampling and testing information obtained from drill holes.

Chapter 8. Optimization Strategies

The objective in optimization exercises is to enhance the value of the project. The process involves reverting back to the primary data and conducting a critical re – interpretation of the data using new and improved analytical principles. The main aim is to optimize mineral resources and reserves thereby increasing revenue and shareholder wealth. In this light, the factors, which tend to impact on potential resources are re – evaluated in the following paragraphs.

8.1. Pothole analysis and classification: Previous versus New classification

Slabbert (1994) used the typical telltale signs (discussed in Section 3.1) to classify the borehole intersections as potholed or non potholed reef (Table 2). Although it appears to be over simplistic and non conclusive to make such general assumptions by merely inspecting a limited source of data (only borehole core), these telltale signs tend to provide some indication of such phenomena.

From the initial UG2 Reef evaluation program, of the 51 boreholes drilled to reef, 45% were defined as potholed intersections (Table 7) according to the typical telltale signs. The total loss in potential mineral reserves was estimated at 60% (45% geological losses and 15% due to rock engineering losses) which in turn implied a global extraction rate of 40% (100% - 60%).

Table 7. Pothole Classification (Slabbert 1994).

Number of Boreholes	Number of Boreholes Potholed	% Pothole Intersections
51	23	45

However this classification by Slabbert (1994) did not elaborate on the severity of the expected potholing. There was no differentiation between less severe, but mineable, and severe unmineable potholes.
It was further assumed by Slabbert (1994), that none of the potholed reef could be mined.

Knoetze (1996) designed a re – classification scheme based on pothole severity. He concluded that a relatively good indicator to determine pothole severity is a comparison of the actual thickness of the underlying Footwall 13 norite to the average thickness over the entire area. Knoetze (1996) re – classified the previously calculated 45% potholed reef intersections in terms of severity (based on Footwall 13 norite thickness – Table 8) as follows:

Minor potholes: UG2 Reef potholed to a maximum depth of 2m and is most likely mineable.
Moderate potholes: UG2 Reef potholed to a depth of 2m – 4m and is potentially mineable with additional secondary development, provided that the normal channel width occurs.
Severe potholes: UG2 Reef potholed to a depth of more than 4m and thus unmineable.
Unclassified/Other potholes: The Footwall 13 norite / UG1 Pyroxenite contact not intersected.

Table 8. Pothole Re - classification (Knoetze 1996).

Number of Boreholes	% Minor Potholes	% Moderate Potholes	% Severe Potholes	% * Other Potholes	% Total Potholes
51	16	16	8	5	45

*potholes not classifiable: Footwall 13 norite / UG1 Pyroxenite contact not intersected

According to the classification system of Knoetze (1996), at least 16% of the previously classified unmineable potholed UG2 Reef can now be mined without any major disruptions in the mining operations. Thus by implication it is suggested that 29% (45% - 16%) constitutes unmineable potholes.

Thus the losses according to the classification of Knoetze (1996) are as follows:
Geological Loss: potholes + faults/dykes/poor ground
: 29% + 10% = 39%
Rock Engineering Pillar Loss: 15%
Total Loss of Reserves = geological losses + rock engineering losses
= 39% + 15% = 54 %

Thus global extraction rate equates to 46% (100% – 54%)
Although this pothole classification puts a better perspective on the severity of the potholing and predicts a higher global extraction rate, it is by no means a conclusive measurement tool. Aspects such as pothole depth, dimensions on dip and strike, local dips and associated ground conditions ultimately determine the success of mining in potholed vicinities.

New classification incorporating latest information (2002)
Observations on a mine- wide scale have revealed that classifying reef intersections to be potholed **OR** non- potholed solely on the occurrence of the basal pegmatoid and/or Footwall 13 thickness is not entirely valid. There are noted occurrences where the basal pegmatoid has narrowed down or pinched out altogether over relatively short distances due to minor rolling of the UG2 Reef. In such cases the reef is not potholed and can be mined by normal stoping operations.

Therefore in order to predict the nature of the reef (potholed or non – potholed) by merely examining boreholes intersections warrants monitoring other site – specific parameters in addition to the basal pegmatoid telltale sign.

The following parameters have been chosen within the No. 12 Shaft area:
- **UG2 Channel Width: average = 62cm**
- **Occurrence of basal pegmatoid**
- **F/W 13 Width: ranges from 4.2m – 8.6m**

The geological boreholes (Table 2) were then re - assessed in terms of these defining criteria.

These criteria indicate values for the norm and a minimum of 2 (of the 3) deviations from the norm have been deemed a potholed intersection (Table 9).

Table 9. New borehole classification system.

Borehole Number	Channel width cm	Presence of Pegmatoid	F/W 13 width m	Distance to Chromitite Leaders m	Comment
BH 12/ 09331	72.0	Yes	>0.73	5.25	Hole stopped in F/W 13
BH 12/ 09330	62.0	Yes	>5.48	3.93	Hole stopped in F/W 13
BH 12/ 09041	65.0	Yes	7.86	3.94	
BH 12/ 09043	69.0	Yes	7.08	1.86	
BH 12/ 09045	69.0	Yes	7.62	3.30	
BH 12/ 09046	64.0	Yes	7.75	1.56	
BH 12/ 09334	59.0	Yes	>6.4	3.57	Hole stopped in F/W 13
BH 12/ 09050	58.0	Yes	6.93	2.10	
BH 12/ 09052	72.0	Yes	6.58	3.70	
BH 12/ 09053	**51.0**	**No**	**6.89**	**8.00**	**Potholed**
BH 12/ 09349	68.0	No	5.20	Faulted	Faulted intersection
BH 12/ 09336	47.0	Yes	6.08	1.55	
BH 12/ 09509	48.0	Yes	7.24	2.07	
BH 12/ 01724	**62.0**	**No**	**Absent**	**3.42**	**Potholed**
BH 12 /01725	54.0	Yes	8.22	1.76	
BH 12/ 01727	59.0	Yes	7.98	1.89	
BH 12/ 01728	60.8	Yes	>4.88	2.02	Stopped in F/W 13
BH 12/ 01730	**54.0**	**No**	**5.54**	**1.98**	**Potholed**
BH 12/ 01731	**78.0**	**No**	**1.55**	**4.13**	**Potholed**
BH 12/ 01733	69.3	Yes	> 4.06	2.24	Stopped in F/W 13
BH 12/ 01734	72.0	Yes	8.15	2.14	
BH 12 /01735	70.5	Yes	> 2.80	1.86	Stopped in F/W 13
BH 12/ 09345	62.0	Yes	4.6	1.83	Stopped in F/W 13
BH 12/ 01737	78.9	Yes	6.19	1.98	
BH 12/ 01738	**69.6**	**No**	**>3.22**	**2.00**	**Potholed**
BH 12/ 01739	58.2	Yes	> 5.66	2.29	Stopped in F/W 13
BH 12/ 09507	**113.0**	**No**	**0.11**	**5.93**	**Potholed**
BH 12/ 01742	**63.5**	**No**	**> 1.98**	**2.91**	**Potholed**
BH 12/ 01743	**67.0**	**No**	**8.74**	**1.86**	**Potholed**
BH 12/ 01744	**80.3**	**No**	**0.70**	**2.49**	**Potholed**
BH 12/ 01740	**115.0**	**No**	**4.01**	**4.42**	**Potholed**
BH 12/ 01746	**52.9**	**No**	**8.58**	**2.91**	**Potholed**
BH 12/ 01747	60.6	Yes	Absent	1.99	IRUP at F/W 13 position
BH 12/ 02428	73.1	Yes	3.20	2.27	
BH 12/ 02429	**69.2**	**No**	**5.48**	**1.85**	**Potholed**

Borehole Number	Channel width cm	Presence of Pegmatoid	F/W 13 width m	Distance to Chromitite Leaders m	Comment
BH 12/ 09526	51.0	Yes	>3.49	4.11	
BH 12/ 02430	107.0	Yes	2.87	5.79	
BH 12/ 02432	65.9	Yes	8.37	2.20	
BH 12/ 02433	57.70	Yes	7.85	1.71	
BH 12/ 02434	68.9	Yes	7.65	2.24	
BH 12/ 09506	**71.0**	**No**	**2.40**	**2.63**	**Potholed**
BH 12/ 02435A	**56.0**	**No**	**7.77**	**1.60**	**Potholed**
BH 12/ 02436	**51.6**	**No**	**6.61**	**2.11**	**Potholed**
BH 12/ 02437	65.2	Yes	7.12	2.65	
BH 12/ 02438	64.4	Yes	6.70	2.17	
BH 12/ 09042	**100.0**	**No**	**Absent**	**3.22**	**Potholed**
BH 12/ 09051	**-**	**No**	**5.58**	**1.70**	**Potholed**
BH 12/ 02440	56.7	Yes	7.83	2.20	
BH 12/ 02441	**36.0**	**Yes**	**7.25**	**1.69**	**Potholed**
BH 12/ 02442	**57.9**	**No**	**7.84**	**2.13**	**Potholed**
BH 12/ 02443	**78.8**	**No**	**4.48**	**1.95**	**Potholed**
BH 12/ 09340	**34.0**	**No**	**absent**	**2.30**	**Potholed.**
BH 12/ 02445	**48.0**	**No**	**4.26**	**2.41**	**Potholed**
BH 12/ 02446	**62.0**	**No**	**8.61**	**2.85**	**Potholed**
BH 12/ 09343	46.0	Yes	> 0.45	1.29	

Slabbert 1994: classified all the boreholes shown in bold red as well as those in bold black as potholed intersections. Thus total potholed intersections = 45% (23/51)

Knoetze 1996: concluded that the minor potholes can be mined and thus implied that potholed intersections would constitute 29%.

New classification 2004: in terms of the new defining criteria, only those boreholes shown in bold black have been classified as potholed intersections.
Thus total potholed intersections = 25% (14 boreholes)

According to this new classification, the potholed reef intersections by definition have progressively decreased from 45% (Slabbert 1994) to 29% (Knoetze 1996) to 25.0%. This suggests that a large proportion of the mineral resource which was previously deemed unmineable (believed to be abandoned in the form of potholes) can in fact be mined. This new percentage increase in the mineral resource is not merely equal to the percentage derived from the decrease in potholes, because factors such as the geometry, dimensions and dips of the potholes strongly influence the success rate of mining. However, it can be concluded that a decrease in pothole frequency will be associated with improved anticipated extraction rates.

8.2. Elimination of regional barrier pillars

Barrier pillars are extensive regional pillars which span the entire shaft block contained within the designated shaft boundaries. These planned pillars form an essential component of any underground mine design plan. These pillars offer flexibility in terms of their relocation and dimensions and can be easily substituted by natural pillars (unmined ground) such as potholes and dykes (Budavari, 1983).

Although the total loss (geological and rock engineering) in mineral resources equates to 39.83%, (Table 4: 6% + 5.3% + 7.5% + 21.03%), only 6% constitutes definite geological losses in the form of major fault zones, dolerite dykes, replacement and dunite bodies whilst the other losses are likely to change as the mining operations progress.
However the loss in potential mineral resources due to potholes has been calculated to be 21.03% all of which are assumed to be unmineable.

As a result this unmined ground can act as natural barriers pillars thus eliminating the need to plan and design barrier pillars. Should this be the case, then the following changes would occur.

From Table 6:
Loss due to rock engineering pillars = 720 977m^2
Elimination of Barrier Pillars = 462 504m^2 (4.8% of the mineral resource)
Thus Net Loss due to rock engineering pillars = 258 473m^2

Based on the estimated extraction rate of 53.28%, the gain in mineral resource is:
(53.28%) (462 504m^2) = 246 422 m^2 which equates to 3% (246422/9578411)

This implies that by the elimination of regional barrier pillars, the estimated mineral resource can be increased by 3% (inclusive of dip and extraction correction factors).

Chapter 9. Benchmarking

Benchmarking is a technique commonly practiced in all industries in order to measure performances and monitor progress between companies operating similar businesses.
Benchmarking is considered an important management tool, which is applicable to all sectors of the business. In the mining industry several complex considerations and intricate processes are involved commencing with the primary extraction of ore from underground to the eventual recovery and refining of the final product.

An important factor which requires careful investigation prior to the commencement of any mining operations is the geological characteristics of the orebody. The orebody in itself forms the backbone of the entire operation.

9.1. Geological features

Due to the close proximity to No. 8, 14 and 20 Shaft areas (Figure 2), it is evident that one could benchmark the UG2 Chromitite Layer within the No. 12 Shaft area against these shaft areas in terms of regional geological characteristics. The chosen geological features are major faults, potholes, dykes and ultramafic replacement pegmatoid bodies.

These structural features strongly influence the extraction rates and therefore the mineral resources and reserves of a shaft's area. Thus any information concerning the occurrences and abundances of such features also tends to provide some indications of the likely extraction rates to be anticipated. The bench marking exercise has revealed a less than 5% discrepancy in the loss of potential mineral resources due to these geological features in the adjoining shafts areas when compared to that of the No.12 Shaft area. There are also favorable comparisons with regards to channel widths, in-situ grades, reef dips, footwall thicknesses and distances to the overlying chromitite leaders all of which are important parameters which could impact on the success of the mining operations. The UG2 Chromitite Layer is presently being mined from the No. 8 Shaft area and there are positive indications regarding the future mining of this reef horizon at the adjoining shafts.

These regional similarities in the geological characteristics across the adjoining shafts areas, provides an encouraging outlook regarding the mineability of the UG2 Chromitite Layer within the area of the No. 12 Shaft.

9.2. Metallurgical Test Work

The mineralogical characteristics and metallurgical properties of the UG2 Chromitite Layer are important features which must be investigated in order to assess the recovery rates. Bench scale laboratory flotation test work (Knoetze, 1996) conducted on 8 composite borehole samples have indicated limited variability.

There was however one important finding which verifies the findings of McLaren (1978).
It was observed that potholed reef is associated with low recovery rates. Potholed reef contains fine grained PGM mineral grains which have an affinity for silicate gangue minerals. Therefore efficient extraction of the PGEs by flotation processes becomes difficult.

The results from the metallurgical test work conducted on UG2 borehole core samples obtained from the No. 20 Shaft area tends to correlate with the findings concerning the metallurgical test work conducted at the No. 12 Shaft area.

These findings are:

- similar recovery rates have been achieved.
- lower recovery rates must be expected in weathered areas and replacement pegmatoid zones.
- apart from the geological and rock engineering problems associated with abnormal reef (weathering, replacement pegmatoid), it is also not economical, from a grade and recovery perspective, to mine in such areas.

Chapter 10. Financial Analysis

This chapter will address the financial viability of mining the UG2 Reef within the No. 12 Shaft area of Impala Platinum.
There are four common financial criteria intrinsic to a discounted cash flow model which can be used in the evaluation of mineral projects and management decisions (Du Toit et al, 2001). These criteria are: the payback period, net present value (NPV), profitability index (PI), internal rate of return (IRR)
These financial criteria are described in detail in Appendix V.

10.1. The input parameters and discounted cash flow (DCF) model.

The net present value (NPV) and internal rate of return (IRR) will be the chosen financial evaluation criteria to assess the viability of mining the UG2 Chromitite Layer within the No. 12 Shaft area.

The following chosen parameters will be used to build the discounted cash flow model shown in Table 10.

[a] Platinum Price: $825.0/oz

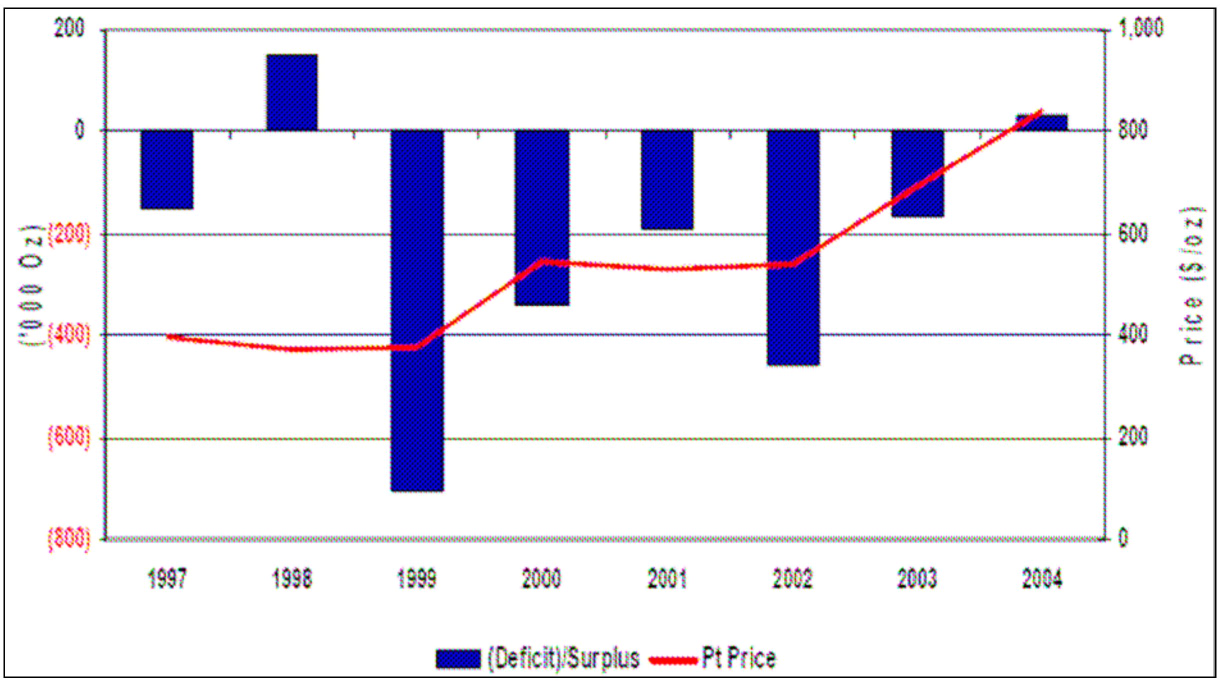


Figure 8. Platinum price and supply/demand trends (Gilmour 2005).

From Figure 8, it is apparent that the price of platinum monitored over an eight year period has been increasing due to an increase in the demand for this metal. Furthermore market research indicates that the demand for platinum as well as its price will continue to increase in the future (Figure 9). Based on these forecasts and the eight year monitoring period, it was decided to use a constant platinum price which is $825/oz. [($775 + $875)/2].

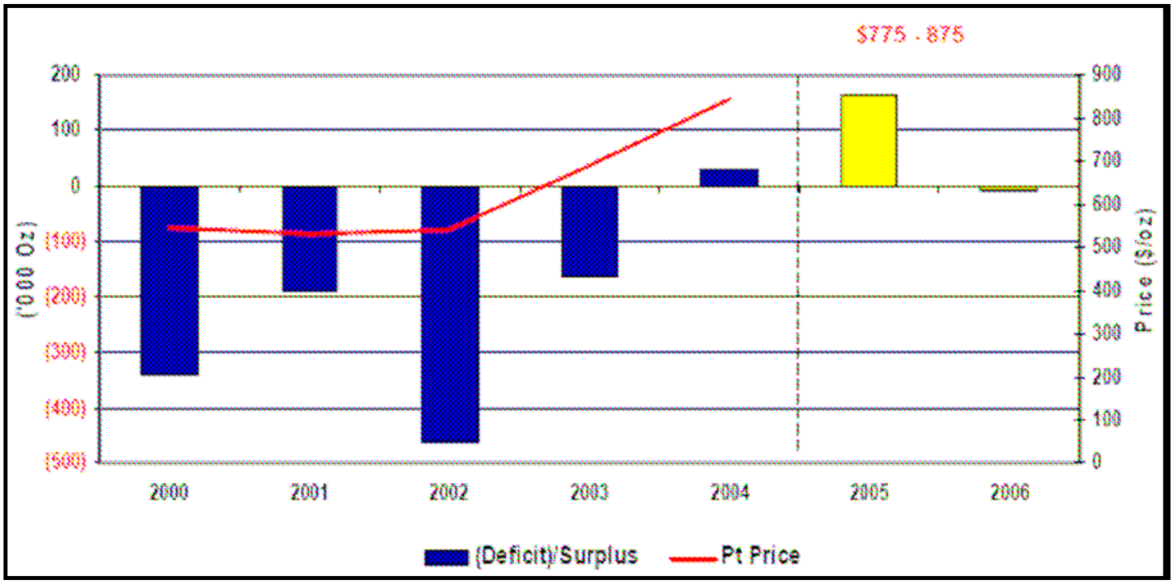


Figure 9. Forecast for platinum supply/demand and its anticipated price (Gilmour 2005).

Furthermore, the precious metals house Johnson Matthey (JM), forecasts a platinum price range from $760/oz to $880/oz for 2005, which is similar to that shown by Gilmour (2005).

[b] Palladium Price: $188.0/oz

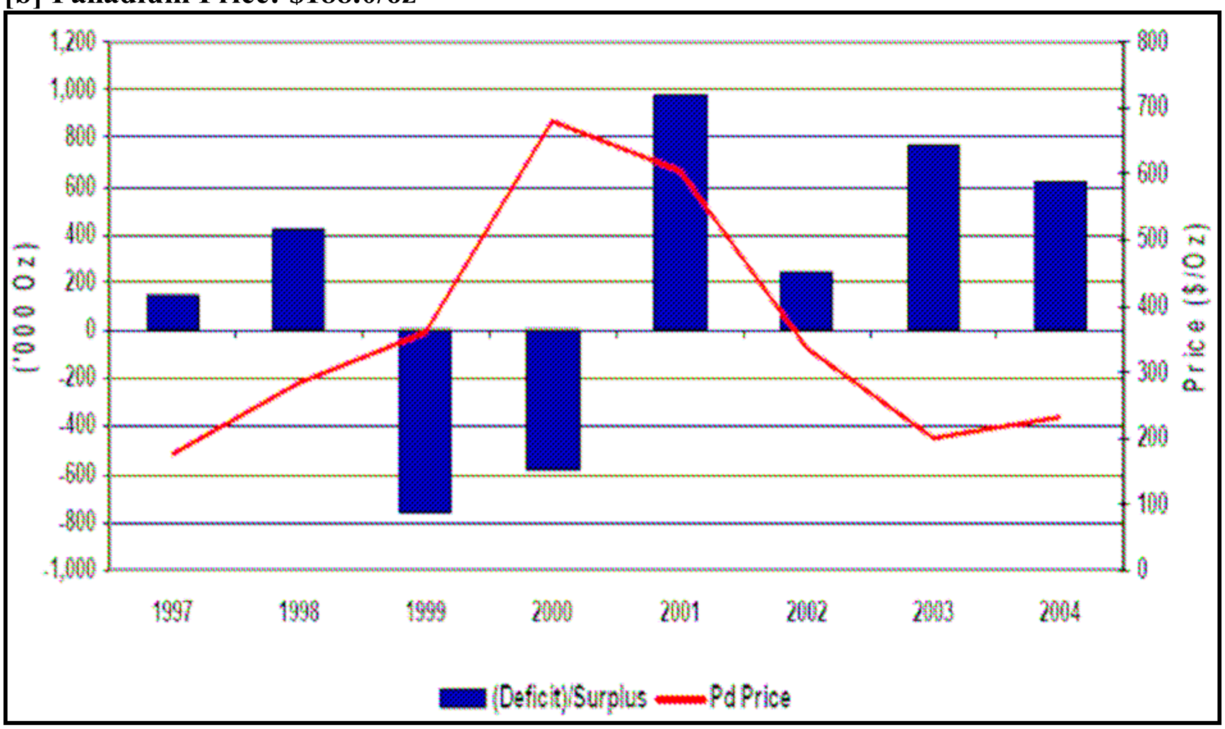


Figure 10. Palladium price and supply/demand trends (Gilmour 2005).

Figure 10 shows two definite trends with regards to the supply/demand and price of palladium. Between 1997 and 2000, there was a worldwide shortage in the supply of palladium and thus a steady increase in the price of the metal. However, between 2000 and 2003, palladium supplies have increased and this has led to a decrease in its price.

Based on the eight year monitoring period (Figure 10) and market forecasts (Figure 11), the palladium price that will be used in the financial analysis of the project is \$188/oz (\$150 + \$225).

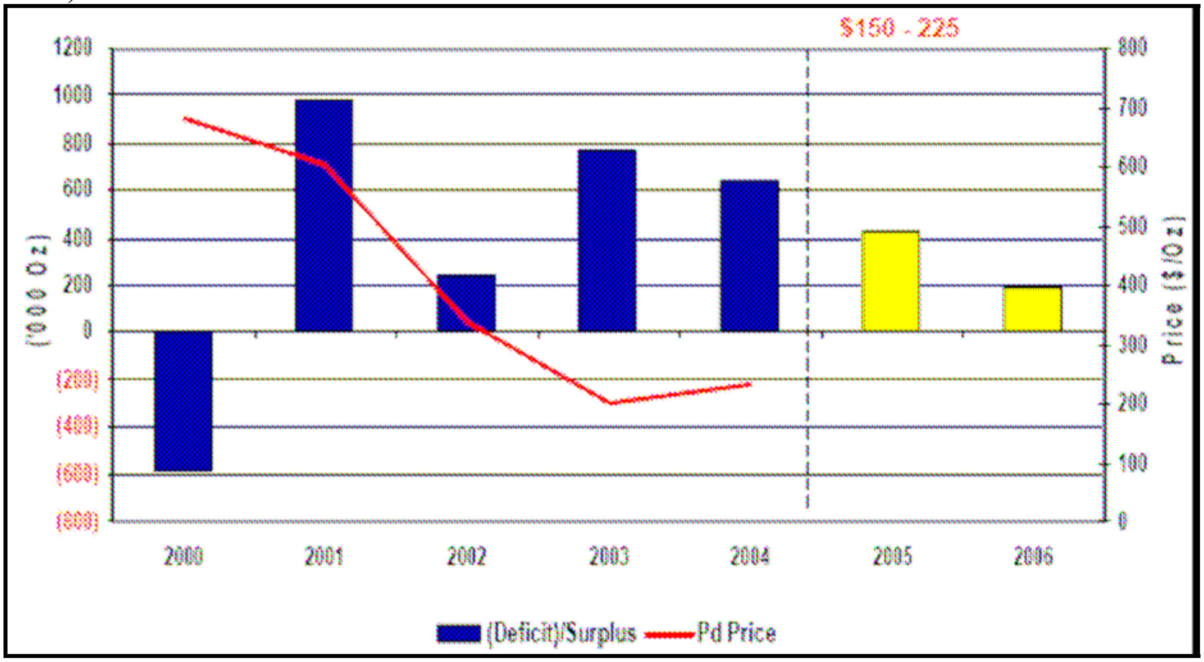


Figure 11. Forecast for palladium supply/demand and its anticipated price (Gilmour 2005).

Furthermore, the precious metals house, Johnson Matthey predicts \$160/oz to \$250/oz for palladium for 2005, which is similar to that shown by Gilmour (2005).

[c]. Rhodium Price: \$1250/oz

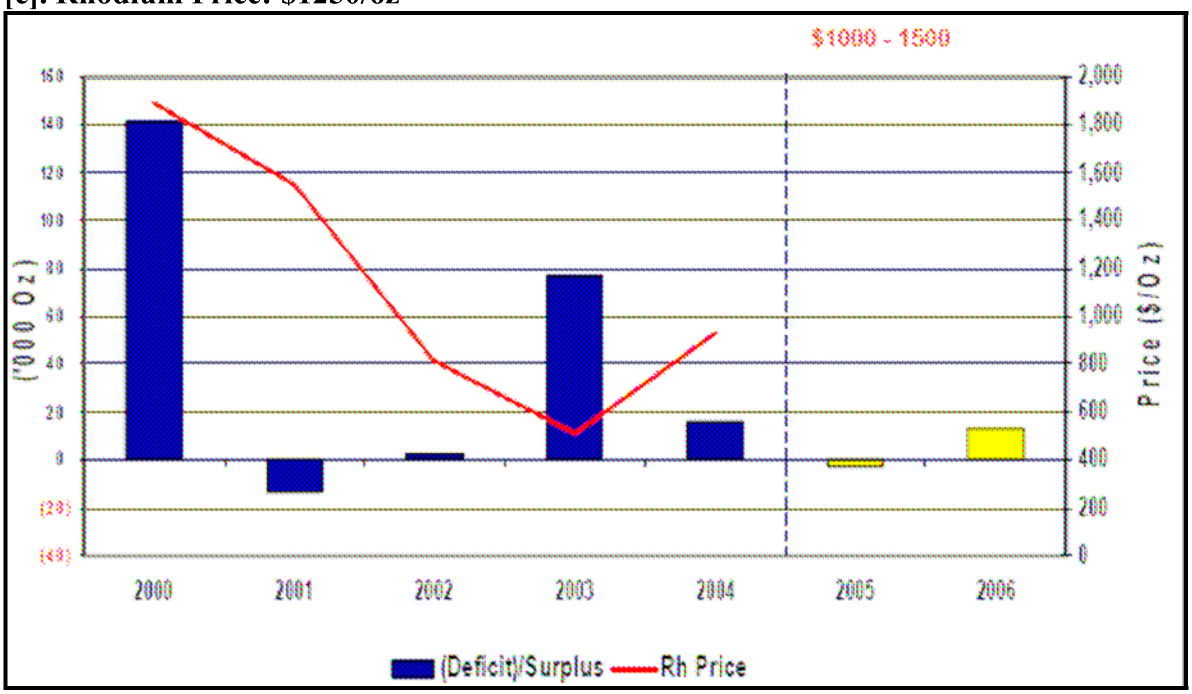


Figure 12. Forecast for rhodium supply/demand and its anticipated price (Gilmour 2005).

Based on the historical and forecast price for Rhodium, a projected price of $1250/oz ($1000 + $1500/2) is used in the financial model.

Other prices used for the metals which have a smaller contribution to the total revenue are:
[d] Ruthenium: $68/oz [Matthey, (2005)]
[e] Iridium: $150/oz [Matthey, (2005)]
[f] Gold: $415/oz [Matthey, (2005)]
[g] Rand/Dollar Exchange Rate: R6.57 [http://www. oanda .com/convert/fxhistory].

The graphical representation of the average rand/dollar exchange rates on a yearly basis for a total period of 17 years (1994 to 2010) is given in Figure 13. It includes historical and current data as well as forecasts (http://www. oanda .com/convert/fxhistory).
Appendix VI shows the detailed monthly rand/dollar exchange rates for this period.

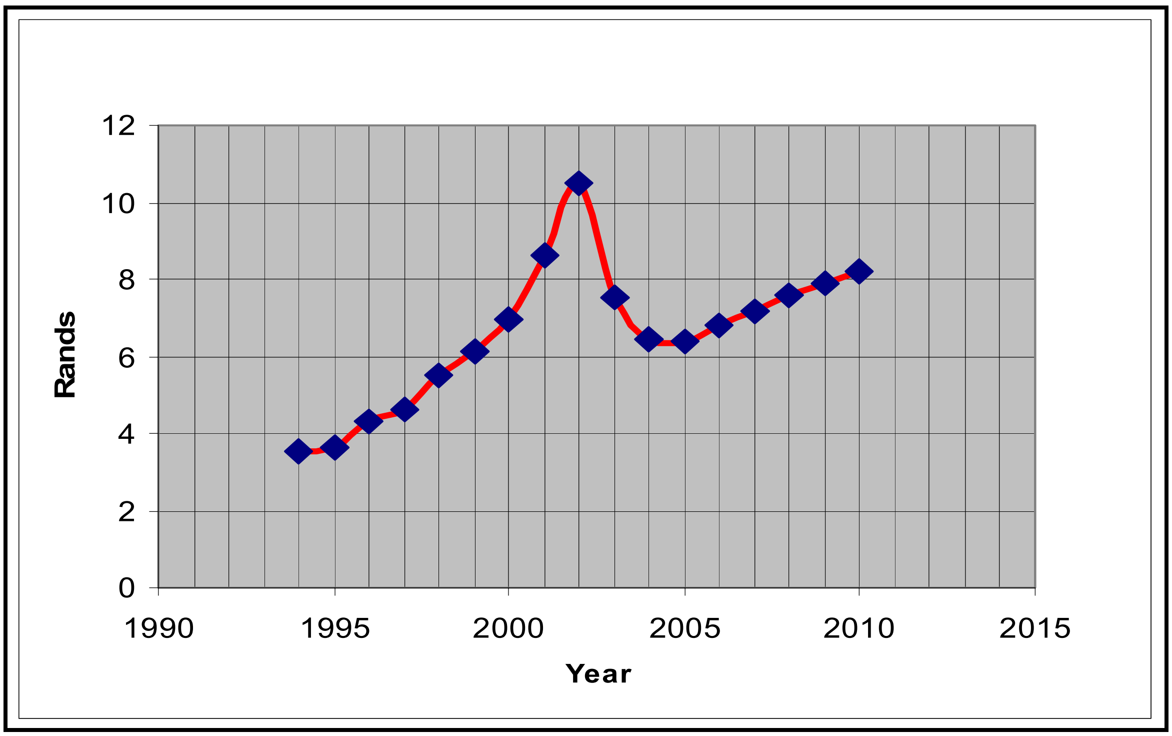


Figure 13. Rand – Dollar Exchange Rates (produced from data found at http://www. oanda .com/convert/fxhistory).

The average rand/dollar exchange rate during the period of interest is R6.57 (Appendix VI), and it was decided to use this exchange rate in the financial calculations.

[h] Estimated Mill Grade: 4.13g/t
The estimated mill grade of 4.13g/t (derivation described in Section 6.3) is used in the financial model.

[i] Taxation: 29%, which is the latest company tax rate.

[j] Discount Rate: 10%

Discounting determines the present value of a future amount, assuming an opportunity to earn a certain return on the money (Gitman 2003).
The discount rate can be calculated by using the capital asset pricing model.

This is a mathematical model and is presented by the formula: **Rf + β [Rm – Rf]** (Du Toit et al, 2001).
where Rf ≡ risk free interest rate
β ≡ beta coefficient (volatility/ risk of the security/ commodity/ project)
Rm ≡ expected rate of return of the market portfolio or market rate of return

Based on the market rate associated with short term government bonds, Rf approximates to 8%. β shows the relationship between the expected rate of return and the systematic risk associated with ordinary shares as quoted on the JSE or determined from the security market line.
An alternative way of deciding on the appropriate discount rate is to add an appropriate project related risk to the risk free interest rate. In this case a risk element of 2% is deemed appropriate for the project (low risk rating) given the certainty of estimations.
This would give a discount rate equal to 10% (8% + 2%) which is used in the base case financial model.

Realizing that the choice of the discount rate entails an element of intuitive judgment, a table has been prepared listing the net present values of the project at different discount rates (Table 12).

[k] Royalties: 22%

Impala Platinum has an agreement to pay the Royal Bafokeng Nation a royalty of 22%. (Implats Annual Report 2004).

[l] Overall Recovery Rate: 80 %

The overall recovery rate of the UG2 Chromitite Layer is slightly lower than that for the Merensky Reef. The recovery rate for the Merensky Reef is 83.2 % (Implats Annual Report 2004). It is assumed that the recovery rate for the UG2 Chromitite Layer is 80%.

[m] Working Costs: R270/ton

The working costs have been based on costs associated with mining, milling, concentrating, smelting, refining, services and other costs. For the purposes of the discounted cash flow model the following estimations have been made which are based on a combination of general market related and mine wide averages (Implats Annual Report 2005): -
Mining cost: R160/ton; Concentrating and smelting cost: R60/ton; Services and other costs: R50/ton.

[n] Capital Costs: R110 million

Capital costs generally comprise the following items: shaft sinking, shaft equipping, infrastructure, pre – production underground mine development, accommodation and amenities. However this project will be carried out within an existing shaft area, therefore the entire currently existing underground and surface infrastructure (main transport haulages and cross- cuts); utility pipes (compressed air, water, electrical cables) and

amenities will be used. This in turn represents an enormous saving in terms of capital expenses. It is thus envisaged that only the following estimated costs, based on market related averages (Mining Weekly, September 2005), will be incurred.
Engineering Requirements – R60 million
Ventilation Requirements – R40 million
Pre – production underground development – R10 million

This capital expenditure of R110 million will be spent over a period of 3 years (estimated time taken to commence stoping operations) as indicated in Table 10 below:

Table 10. Estimated capital expenditure.

Year	Engineering Costs		Ventilation Requirements		Pre – production underground development	
1	50%	R30m	20%	R8m	60%	R6m
2	30%	R18m	50%	R20m	20%	R2m
3	20%	R12m	30%	R12m	20%	R2m
Total	**100%**	**R60m**	**100%**	**R40m**	**100%**	**R10m**

Hence the capital expenditure is: Year 1: R 44 000 000
Year 2: R 40 000 000
Year 3: R 26 000 000

Total estimated capital expenditure = R110 million

[o] Tonnage profile

With regards to a shaft's tonnage capacity, it is understood that the tons derived from mining the UG2 Chromitite Layer has to be phased in with the rest of a shaft's mining activities such that the entire operation is at optimum levels. The tonnage profile shown in Figure 14 is not based on a well constrained mine plan but shall be used for demonstrative purposes of this project.

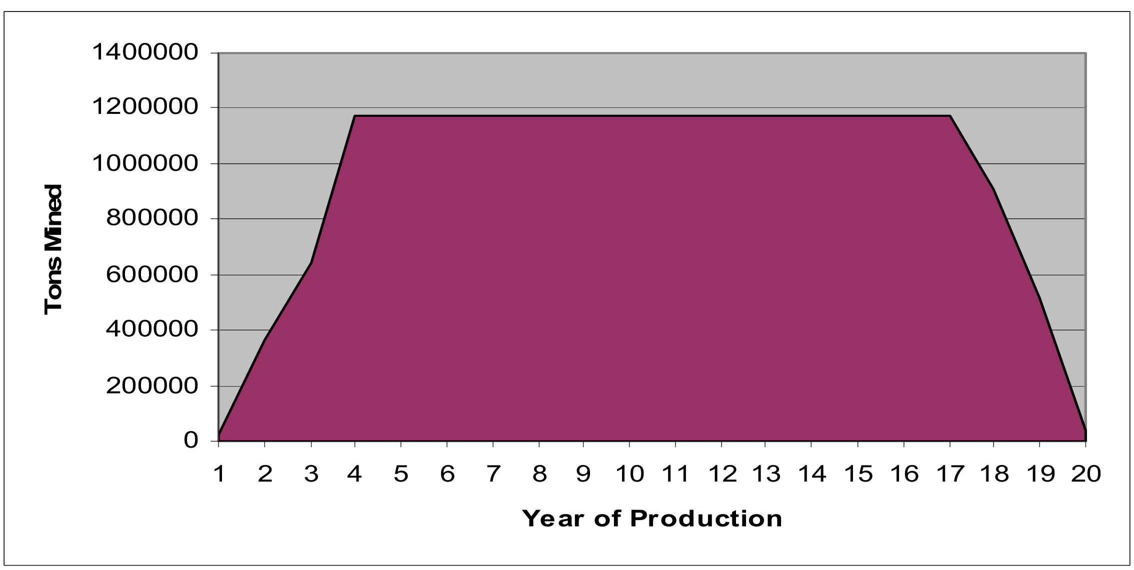


Figure 14. Estimated tonnage profile for the UG2 Chromitite Layer.

For approximately the first 3 years of the project, the mining will predominately be reef development operations (raises and winzes). Thereafter, between Years 3 and 4, there would be a steady increase in the amount of tons mined due to intensive stoping operations. The peak production phase is reached in Year 4. This steady state is maintained until Year 17; thereafter there is a rapid decrease in the tons being mined with the operations eventually ceasing 20 years after its commencement.

Table 11. Discounted cash flow model based on the estimated and assumed input parameters.

Year	Capex. Rand-Million	Tons Mined	Mill Grade g/t	PGM Rec. %	5PGM + Au Rec. oz	Revenue Rand	Working Cost Rand	Royalties @ 22% Rand	Cashflow before tax Rand	Tax 29% Rand	Cashflow after tax Rand	Undisc. accum. cashflow	Discount cashflow @ 10%	Disc. accum cashflow
2005	44	18934	2.08	80 %	1002	3946577	6626900	13106288	59574035	17276470	42297565	42297565	38452332	38452332
2006	40	359759	2.08	80 %	19842	78145048	125915650	14024207	63746395	18486454	45259940	87557505	72361575	110813906
2007	26	643779	2.08	80 %	30486	120063316	225322650	20784470	94474864	27397711	67077153	154634659	116179308	226993214
2008		1173951	4.13	80 %	116875	460292394	410882850	8909918	40499627	11744892	28754735	125879924	85977682	312970896
2009		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	87670774	54436653	367407549
2010		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	49461625	27919798	395327347
2011		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	11252475	5774299	401101646
2012		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	26956674	12575487	388526158
2013		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	65165824	27636671	3608894[illegible]
2014		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	103374973	39855527	3210339[illegible]
2015		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	141584123	49624371	2714095[illegible]
2016		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	179793272	57287677	2141219[illegible]
2017		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	218002422	63147536	1509743[illegible]
2018		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	256211571	67468514	835058[illegible]
2019		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	294420721	70481980	130238[illegible]
2020		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	332629870	72389951	593660[illegible]
2021		1173951	4.13	80 %	121000	476538008	410882850	11839455	53815704	15606554	38209149	370839020	73368523	1327345[illegible]
2022		908865	4.13	80 %	93678	368932534	318102750	9166027	41663757	12082490	29581268	400420287	72019108	2047537[illegible]
2023		511236	4.13	80 %	52694	207524322	178932600	5155884	23435838	6796393	16639445	417059732	68192599	2729463[illegible]
2024		37876	4.13	80 %	3904	15374878	13256600	381985	1736294	503525	1232768	418292500	62176515	3351228[illegible]
2025		18934	4.13	80 %	1952	7685815	6626900	190952	867963	251709	616254	418908754	56607379	391730194
Total	110	18934697				7456958993	6627143950	129802713	590012330	171103576	418908754			
Net Present Value @ 10% R391,730,194														
Internal Rate of Return 16.57 %														

* figures shown in red indicate negative balances

* abbreviations used: Capex – capital expenditure; PGM – platinum group metals; Rec – recovered; Undisc – undiscounted; Accum – accumulated; Disc - discounted

Discussion of Table 11

The tons to be mined on an annual basis have been scheduled as per the tonnage profile shown in Figure 14. Initially less tons are mined and it would be development operations through the reef horizon (raises and winzes). Though it is reef tons, the dimensions of the required excavation introduces dilution which is significantly higher than the stope dilution (calculated in Section 6.3) resulting in a low head grade. This low head grade has been estimated as follows:

Assumption – reef development excavations (such as raises, winzes) will be mined at an estimated height of 240cm (in order to accommodate compressed air, water and ventilation pipes). Since the average channel width has been estimated to be 66cm, this implies that 174cm (240 – 66) will constitute footwall waste. For demonstrative purposes assume that the 174cm has a trace value of 0.1g/t, thus the dilution introduced into the development operation is estimated as follows:

Average cm.g/t: 567 cm.g/t (8.59 g/t over 66cm channel width)

Additional cm.g/t: 17.4 cm.g/t (0.1g/t * 174cm)

New reserve grade (5PGE +Au) over 240cm stope width: [584cm.g/t]/240cm = 2.43 g/t
This indicates a grade decrease of 6.16 g/t (8.59 – 2.43g/t) which introduces approximately 72% (6.16/8.59) dilution.

Performing calculations similar to those shown in Table 4, however now based on 72% dilution and 240cm vertical height, results in an estimated head grade of 2.08g/t.
It is assumed that the reef development phase would last approximately 3 years, therefore this low head grade of 2.08g/t has been assigned to the first 3 years of the project.
Approximately 3 years after the commencement of the project, stoping operations contribute significantly to the increase in tons and head grade. Peak production is reached in the year 2008. The head grade is 4.13g/t over a stope width of 100cm. This optimum production profile is maintained in this steady state until the year 2021. Thereafter there is a downward trend in terms of tons mined with the eventual closure being reached in 2025.

From Table 11 and graphically shown in Figure 15 it is evident that the payback period (time taken to recover the initial capital investment amount) is between 9 and 10 years (between 2014 and 2015) after the commencement of the project.

However due to its numerous shortcomings (described in Appendix V) among which is its failure to consider the time value of money it is more appropriate to determine the discounted payback period (Gitman, 2003).
Applying the calculated discount rate of 10%, the payback period is now reached between 15 and 16 years (between 2020 and 2021) after the start of the project.

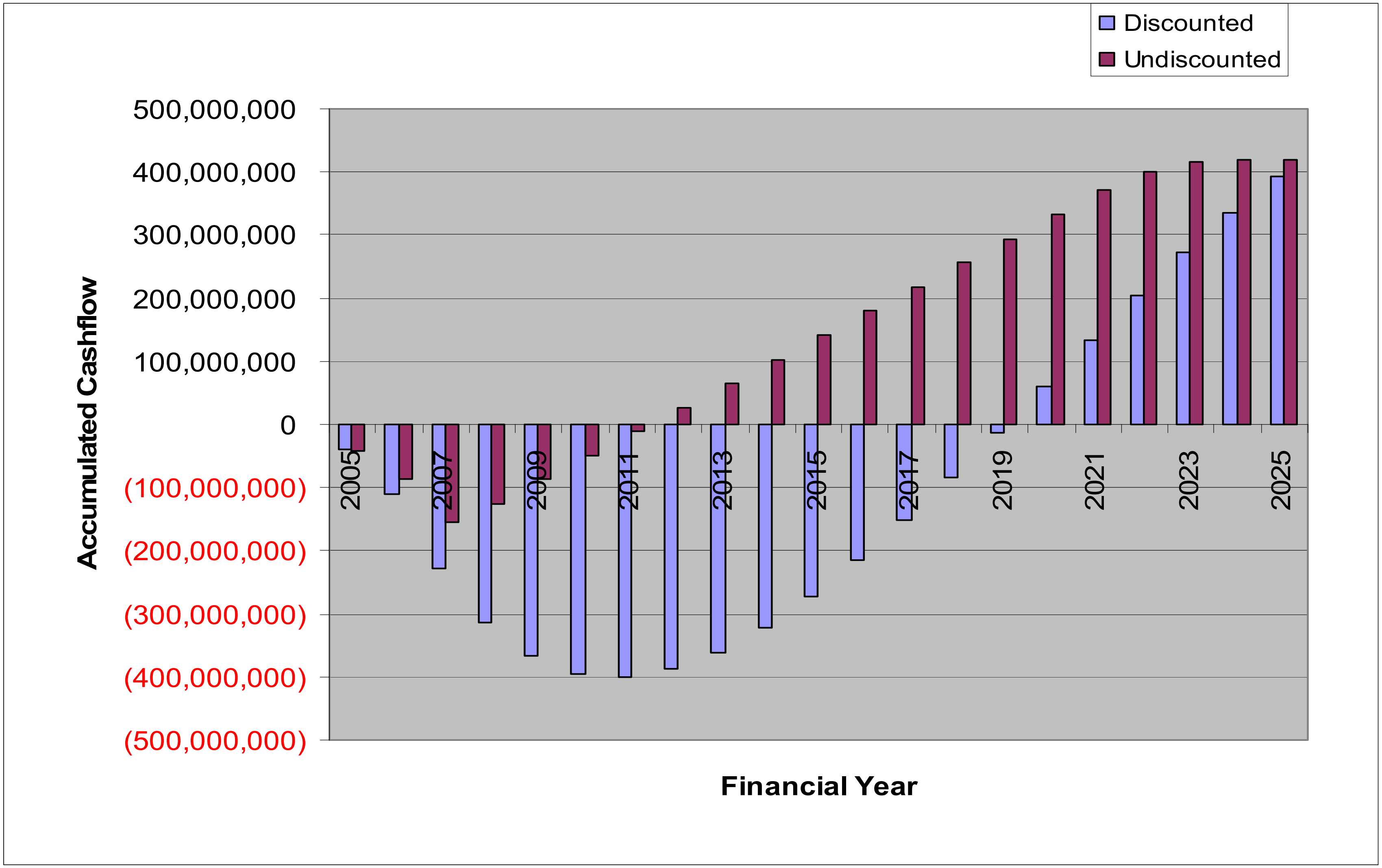


Figure 15. Discounted and undiscounted accumulated cash flow graphs.

The net present value has an inverse relationship with the discount rate.

Table 12. Discount rates and related net present values.

NPV @	1%	2%	3%	4%	5%
Rand	2,552,798,115	2,110,343,426	1,741,671,775	1,433,875,502	1,176,428,261
NPV @	6%	7%	8%	9%	10%
Rand	960,723,505	779,706,496	627,580,113	499,569,029	391,730,194
NPV @	11%	12%	13%	14%	15%
Rand	300,800,139	224,071,640	159,293,853	104,591,252	58,397,679
NPV @	16%	17%	18%	19%	20%
Rand	19,402,560	13,493,065	41,212,557	64,533,259	84,110,830
NPV @	21%	22%	23%	24%	25%
Rand	100,499,323	114,167,771	125,513,927	134,875,670	142,540,482
NPV @	26%	27%	28%	29%	30%
Rand	148,753,356	153,723,389	157,629,300	160,624,058	162,838,764
NPV @	31%	32%	33%	34%	
Rand	164,385,920	165,362,178	165,850,668	165,922,965	

- figures shown in red indicate negative balances.

Figure 16 shows the graphical representation of the decrease in the net present value with an increase in the discount rate.

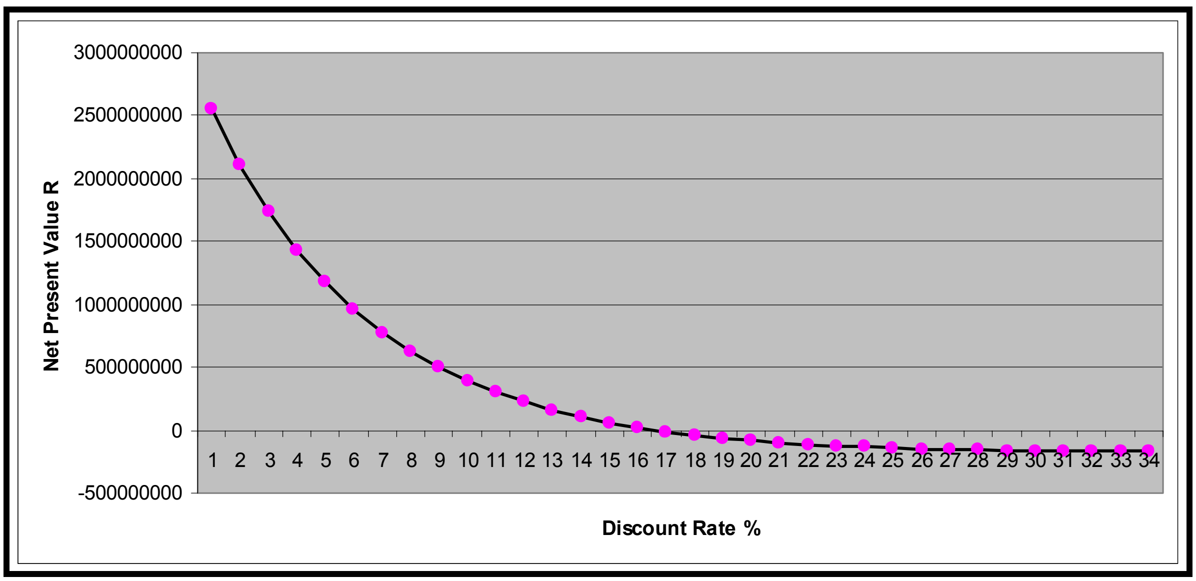


Figure 16. Internal Rate of Return.

The internal rate of return is defined as that discount rate at which the NPV equals to zero. From Figure 16, it has been calculated that the internal rate of return for the project is 16.57%.

Chapter 11. Sensitivity Analysis

Sensitivity analysis is a means of evaluating the effects of uncertainty/risk by determining how an investment alternative profitability varies as the parameters are varied.
Sensitivity analysis is a means of identifying those critical variables that if changed could considerably affect the profitability measure.
The analysis involves changing the values of the individual variables and assessing the effect on the net present value (NPV). Only one parameter is changed at a time whilst the others remain constant. The chosen parameters are:

- sales revenue (which is influenced by changes in exchange rate, grade, tonnage and metal price)
- working costs
- capital costs

Sales revenue, working costs and capital expenditure are important variables, which provide important information regarding profit and loss scenarios, and are thus utilized in the sensitivity analysis.

Table 13. Optimistic and pessimistic scenarios and the resultant effect on the net present value.

Percentage Deviation	Working Costs	Capital Expenses (CAPEX)	Sales Revenue
+15%	-575162838	402408754	1537452603
+10%	-243805641	407908754	1164604653
+5%	87551556	413408754	791756703
0 – Base Case	**418908754**	**418908754**	**418908754**
-5%	750265951	424408754	46060804
-10%	1081623146	429908754	-326787145
-15%	1412980347	435408754	-699635095

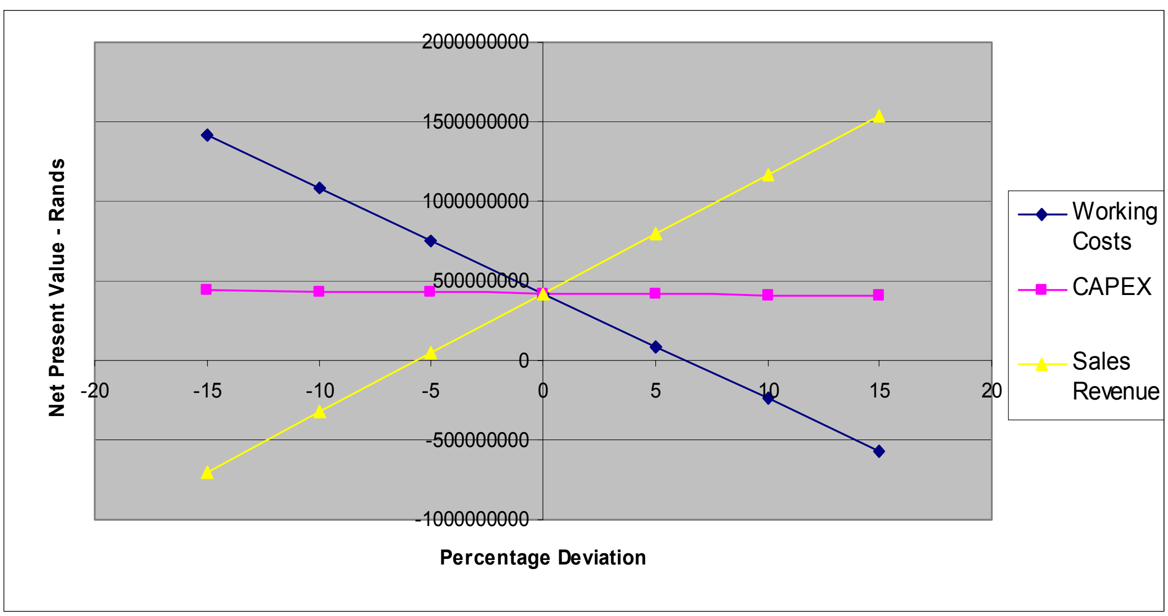


Figure 17. Sensitivity Analysis.

The sensitivity of the net present value to the specific variable is indicated by the slope of the variable. The steeper the slope, the more sensitive is the net present value to that respective variable. Different scenarios are reflected for percentage deviations from the base case (Table 13). The base case represents the undiscounted accumulated cash flow obtained from Table 11.

From Figure 17, it is clear that the net present value is most sensitive to the sales revenue, followed by the working costs and is least sensitive to the capital expenses (Capex). Consequently the items which have the most influence on the NPV and hence determines the viability of the project must be addressed in the risk analysis.

The sales revenue is strongly influenced by the following aspects:

- metal prices
- exchange rates
- grade of the deposit
- mineral resource

Chapter 12. Risk Management

Risk is defined as a measure of the probability and resultant consequence of a deviation from an expected value (Nicholas 2001). The exploration for and development of mineral deposits involves significant risks, which even a combination of careful evaluation, experience and knowledge may not eliminate. Feasibility studies and mine planning thus requires values for a large number of variables such as costs, prices, inflation, capital investment, grades, tonnages, rock properties, dilution, continuity of mineralization and mill recovery rates.

These parameters are unknown at the time of commencing the feasibility study and hence must be estimated from the available data. Due to the uncertainty of these variables, a certain degree of risk is associated with each aspect.
Risk Management involves the following coherent steps (Kerzner 2001):

- Identifying the risks
- Quantifying the risks
- Risk analysis
- Presenting the results
- Beyond presentation

12.1. Risk Identification

The commercial viability of mining the UG2 Chromitite Layer within the No. 12 Shaft area is dependent on several factors which have been identified in the sensitivity analysis. These factors are:

- grade variation within the deposit
- estimation of the mineral reserve
- metal prices
- exchange rates

The risks associated with each of these factors, either alone or in combination, cannot be entirely predicted and their impact may result in the company not receiving an adequate return on invested capital.

12.2. Risk Quantification

There is no hard data available which can be used to assess the potential damage associated with the risk factors concerning the UG2 Reef within the No. 12 Shaft area. In such cases, a qualitative approach is used to measure the risk.

Kerzner 2001, proposed the following common qualitative risk rating system:

- **High risk:** substantial impact on cost, schedule or technical. Substantial action is required to alleviate issues. High priority management attention is required.
- **Moderate risk:** some impact on cost, schedule or technical. Special action may be required to alleviate issues. Additional management attention may be needed.
- **Low Risk:** minimal impact on cost, schedule or technical. Normal management oversight is sufficient.

Figure 18 shows the qualitative ranking system of low, moderate and high risk in relation to the important components of risk namely **probability** of an occurrence of an event and the related **impact** thereof (Kerzner, 2001). Furthermore, percentages of probability can be used to describe uncertainty statements (Kernzer 2001) as shown below.

Table 14. Uncertainty statements and probability (Kernzer 2001).

Statement	Assigned Probability
Almost no chance	< 5%
Chances are slight/highly unlikely	5% < 15%
Probably not unlikely/improbable/doubtful	15% < 45%
Good chance	45% < 50%
Better than even	50% < 55%
Believable/probably/likely	55% > 85%
High likely/ Almost likely	85% - 100%

The risk associated with each factor which will impact on the mining of the UG2 Chromitite Layer within the No. 12 Shaft area will be qualitatively assessed (as per criteria in Table 14) and then ranked according to its magnitude (Figure 18).

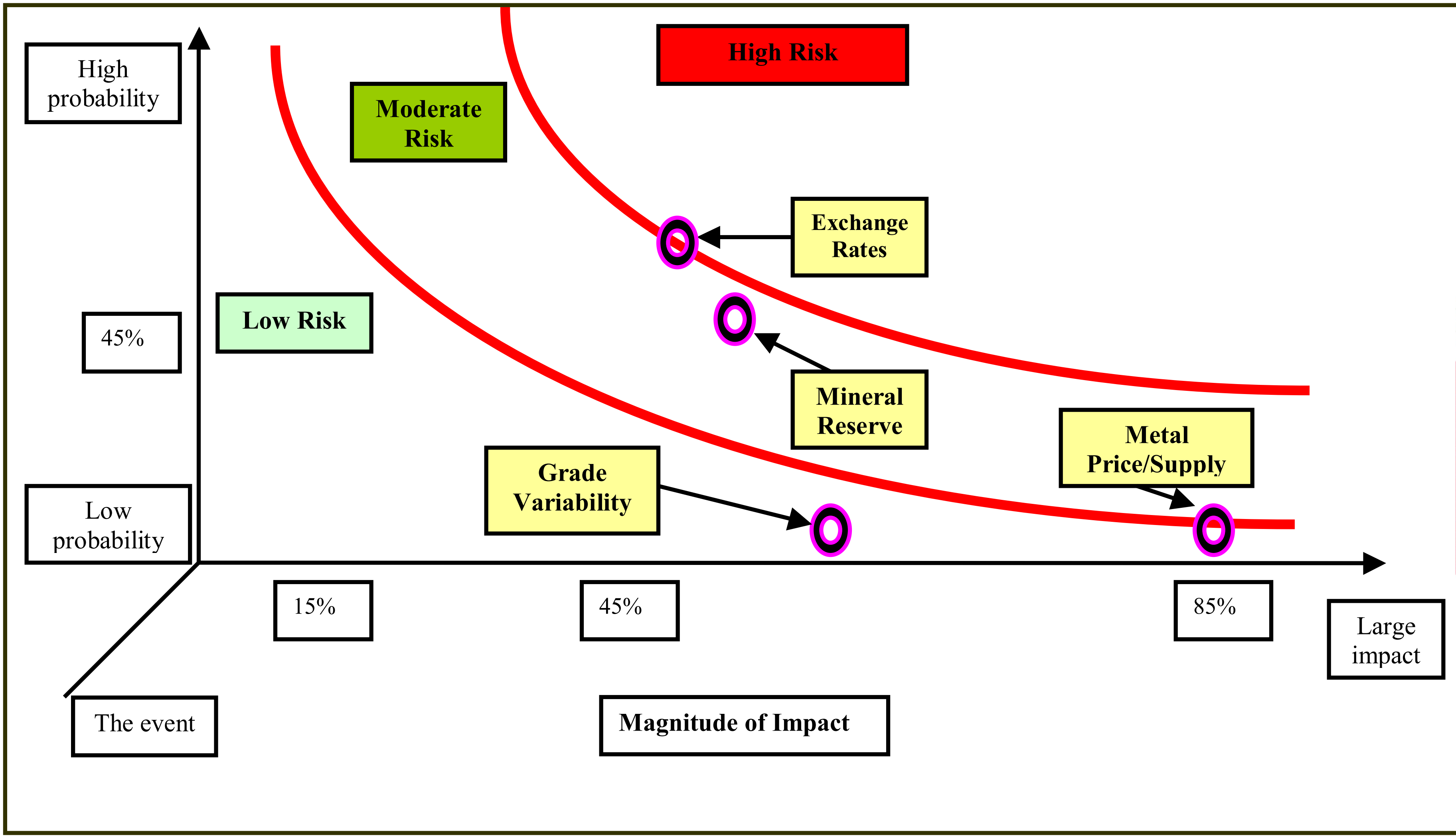


Figure 18. Risk Components and Quantification (adapted from Kerzner 2001).

Grade variability of the UG2 Chromitite Layer

Unexpected changes in the economic environment are extremely detrimental to the sales revenue of a business especially when combined with an adverse change in the anticipated grade of the reef. It is thus imperative that unpredictability is anticipated and catered for prior to the commencement of any reef development or mining operations. The spatial distribution of the UG2 exploration boreholes (Figure 5) across the No. 12 Shaft area indicates satisfactory coverage in terms of detecting any regional areas of intense grade discrepancies. The *in situ* mineral resource grades as determined from the exploration boreholes are graphically shown in Figure 19. For further comparative purposes the exploration borehole grades have been benchmarked against the average UG2 *in - situ* grade at Impala Platinum.
The average *in – situ* UG2 mineral resource grade (5PGE + Au) at Impala Platinum is 9.21g/t (Implats Annual Report 2004). The average *in situ* grade calculated from the exploration boreholes is 8.59 g/t (based on 5PGE + Au), which compares favorably to 9.21g/t indicating an average discrepancy of 0.62g/t.

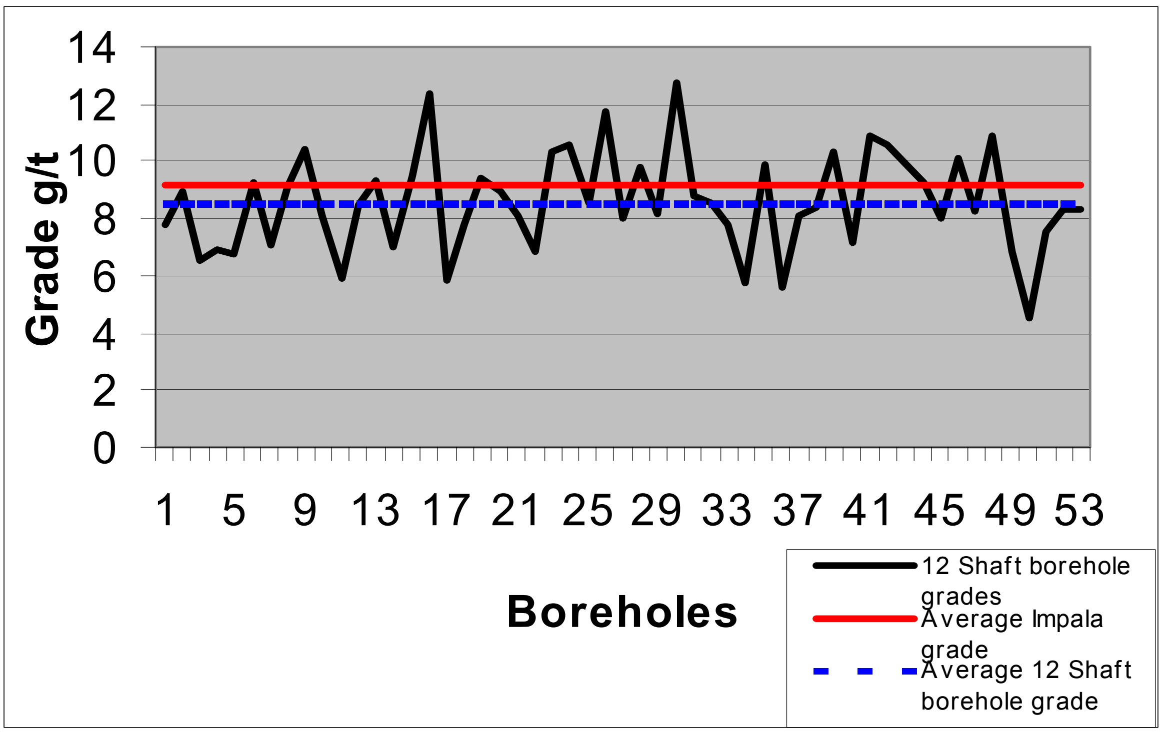


Figure 19. UG2 Reef *in- situ* grades (5PGE + Au).

From Figure 19, it is clear that the grade of the UG2 Chromitite Layer within the No. 12 Shaft area is not expected to vastly differ from that currently obtained on a mine – wide basis for Impala Platinum.
Using the upper probability limits from Table 14, it is apparent that the probability of grade variability is less than 5%. However, should the event occur then the impact would result in a substantial influence on costs, schedules and technical aspects.
At a 5% probability the magnitude of the impact can be greater than 50%. However this risk maybe ameliorated by simultaneous mining from various stopes within the shaft area to ensure that the Run of Mine material remains of a consistent grade. Therefore the risk associated with grade variability is ranked as low (Figure 18).

Estimation of the mineable mineral resource

The estimation of the mineral resource depends on several factors such as:
- detailed interpretation and understanding of the geological structures and characteristics of the ore body.
- the applicable extraction and dip correction factors catering for the geological and mining losses.

The geological structural features and pillar requirements which could strongly impact on the estimation of the mineral reserve have already been identified (Table 6).
The losses in mineral resources associated with each feature have been estimated within reasonable accuracy and was used in determining the global extraction rate.

With reference to the UG2 geological structural plan (Figure 5), it is apparent that the ore body can be divided into different risk probability categories on the basis of geological features. The anticipated geological structural features tend to decrease from the south to the north of the shaft area. This implies that the southern portions have a better than even chance (55% probability from Table 14) of intersecting more complex structural features as compared to that of the northern reaches of the shaft. It is unlikely (45% probability from Table 14) that any more complex features could be intersected in the northern parts of the shaft. The average of these two probabilities is 50%, which suggests that there is an almost likely chance that the estimated mineral reserve is likely to change. This change will impact significantly on costs, schedules and technical concerns. Therefore the risk associated with the estimation of the mineral reserve lies in the region of moderate risk (Figure 18).

Metal Prices

Metals prices are affected by supply, demand and monetary exchange rates. There are however several other unpredictable factors which could affect the supply and demand and hence the price of the metal. Such factors include the discovery of new deposits, introduction of new affordable alternatives, government restrictions, legislation and restrictive practices.

Figures 9, 11 and 12 show the supply/demand and the expected future price trends for platinum, palladium and rhodium (Gilmour 2005; JM 2005).
These metals are in great demand because they are extensively used in the automobile, jewellery, dental and electronic industries.
Currently, all indications are that the automotive industry will be the major driver of platinum and palladium demand in the medium to long term (Figures 20 and 21).
A significant potion of this future demand will come from both light and heavy duty diesel emission control technologies that are dominated by platinum formulations (Gilmour 2005; Matthey 2005).
Supported by jewellery demand, the market is forecast to remain in equilibrium which in turn will be supportive of a firm pricing regime.
A combination of rekindled automotive demand together with new applications in a more stable price environment should boost palladium demand in the longer term.

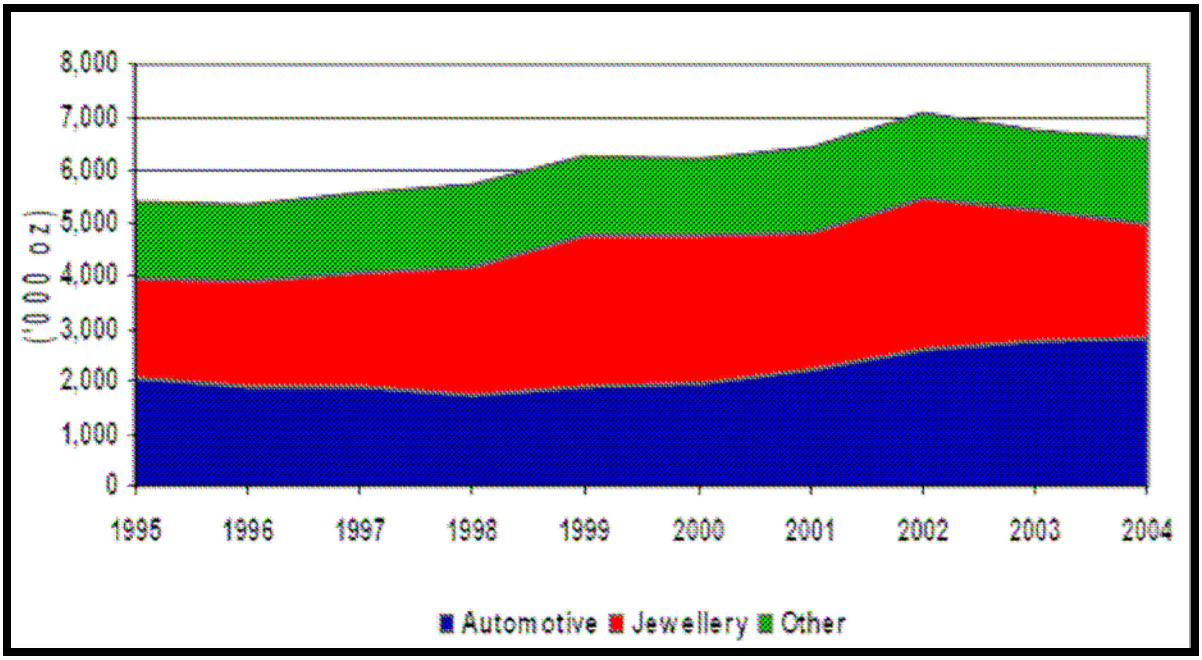


Figure 20. Platinum Demand (Gilmour 2005).

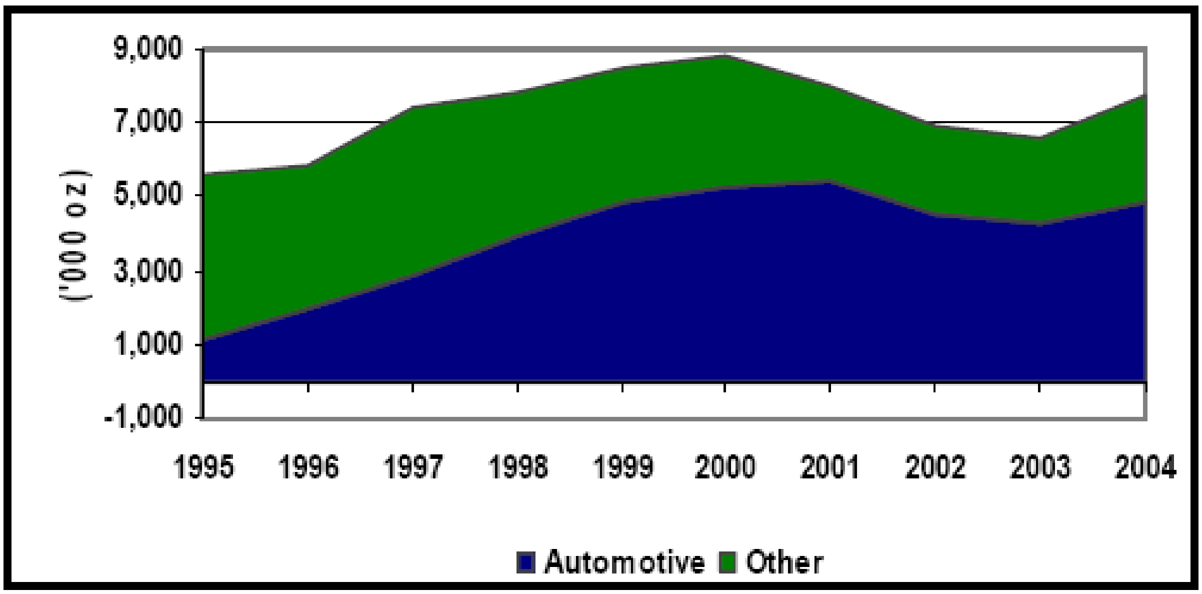


Figure 21. Palladium Demand (Gilmour 2005).

Thus it can be concluded that it is highly unlikely (<15% probability) for the metal price or demand to drop to uneconomic levels. However, should this occur, then the magnitude of the impact on the company could reach significant proportions.

A less than 15% probability with a high impact thus renders this factor to be classified as low to moderate risk (shown on Figure 18).

Fluctuations in Exchange Rates

Companies which enter into sales contracts in currencies other than their own are prone to serious and abrupt reversal of fortune if the exchange rate moves against them either because of currency market transactions or because of Government devaluation or strengthening of local currency.
The metals produced are sold in US dollars. Thus dollar weakness and rand strength are of some concern as this could impact on the future prices and availability of platinum and palladium.

For the financial analyses of the project, a flat exchange rate is used which is a calculated average for the period from 1994 and projected to 2011. However, it is apparent from Figure 13 that there is a gradual increase in the Rand's exchange rate and thus a flat exchange rate is actually introducing a conservative interpretation. There is a good chance that this steady upward trend would continue. A 'good chance' of an event occurring assumes a probability of 45% to 50% (Kernzer 2003). This specific event will also have a high impact (>45%) on the project. Consequently, the risk associated with this parameter could be classified as moderate to high. Incidentally, this could imply upward potential for the project.

Chapter 13. Conclusion

The assessment of the UG2 Chromitite Layer within the No. 12 Shaft area was of an analytical nature broadly encompassing all the major factors which needs to be addressed prior to reaching a decision on the economic viability of mining an orebody. The project work has included reviews and additional investigative studies in the following fields namely: exploratory drilling, geological mapping and structural interpretation, assaying of borehole core, geostatistics, estimation of the mineral resource, project optimization, benchmarking, financial analyses, sensitivity analyses and risk management. The procedure in each of these domains has involved the collection of data, analysis and deductions.

The UG2 Chromitite Layer has been significantly explored by underground geological boreholes drilled out of the existing Merensky Reef workings. The drilling density of the exploratory boreholes into the No. 12 Shaft block was sufficiently adequate to allow correlation of the various lithologies and geological structural features across this specific lease area. This level of detailed correlation and interpretation has assisted in the compilation of a geological model for the UG2 Chromitite Layer within the No. 12 Shaft area.
The UG2 Chromitite Layer intersections encountered in the boreholes were assayed according to the 6 hour fire assay method. The average *in situ* grade of the orebody was calculated to a sufficently high degree of confidence. The global extraction rate applied to the orebody has catered for losses in the mineral resource due to geological structural features and rock engineering pillars. In order to increase the potential value of the project, suitable optimization techniques were investigated which could increase the global extraction for the UG2 Chromitite Layer within the No. 12 Shaft area. The benchmarking exercise has incorporated the regional geological characteristics of the orebody across the neighboring shafts. The metallurgical test work performed on borehole core samples has also revealed favorable results. The input parameters used to construct the discounted cash flow model have been based on historical market related averages, estimations and calculated assumptions. The estimated discounted cash flow model has revealed a positive net present value and this confirms the potential worth of the project. The factors which will strongly influence the viability of the mineral project have been isolated in the sensitivity analyses. The risks associated with these specific project threatening factors have been satisfactorily investigated and quantified.

All areas of the investigation have produced favorable results. Thus it can be concluded that based on the calculations, market averages and assumptions combined with personal views contained herein it would be economically viable to mine the UG2 Chromitite Layer within the No. 12 Shaft area of Impala Platinum.

References

Amos, Q.G. and Sagrabb, T. (1994). Managing Financial Risk in the Mining Industry, 4th Proceedings Large Open Pit Mining Conference (The Australasian Institute of Mining and Metallurgy: Melbourne). 295 – 299.

Amos, Q.G. (2001). Resources and Risk – A Lender's View in Mineral Resource and Ore Reserve Estimation – The AusIMM Guide to Good Practice (The Australasian Institute of Mining and Metallurgy: Melbourne). 511- 517.

Anheausser, C.R. (1987). The nature and geotectonic setting of Archaean and early Proterozoic layered intrusions in Africa: A review. *Indaba on the tectonic setting of layered intrusives.* Geol. Soc. Of South Africa, Pretoria, 1- 4.

Appleyard, G.R.., Gilfillan, J.F. and Northcote, G. (2001). Mineral Resource and Ore Reserve Estimation - An Overview and Outline. 1- 10.

Appleyard, G.R.., and Smith, C.L. (2001). Non – Resource Inputs to Estimation of Ore Reserves – The Modifying Factors in *Mineral Resource and Ore Reserve and Estimation – The AusIMM Guide to Good Practice,* The Australasian Institute of Mining and Metallurgy: Melbourne. 325 – 332.

Balakrishna, M. (2004). A Geological Evaluation of the UG2 Reef Horizon within the No. 12 Shaft area – in house report.

Beamish, E. and Van Loon J. C. (1977). Analysis of Noble Metals Overview and Selected Methods. Academic Press, New York.

Boxer, R.W. (1972). Theory and Problems of Statistics.

Brooks, C. (2001). Analysis? Which analysis, of what and why. 20th IGES, Santiago, Chile, unpublished manuscript.

Buchanan, D.L. (1987). Geological characteristics of current platinum group element exploration targets. *Indaba on the tectonic setting of layered intrusives.* Geol. Soc. Of South Africa, Pretoria, 22- 26.

Budavari, S. (1983). Rock Mechanics in Mining Practices. The South African Institute of Mining and Metallurgy. Monograph Series No. **5**. 173 – 195.

Carlos, C., Flynn, D., Uliana, E. and Wormald, M. (2001). Financial Management, *4th edition*. 229- 246.

Cawthorn, R.G. (1999). The platinum and palladium resources of the Bushveld Complex. S. Afr.J. Sci. **95**. 481 – 489.

Cawthorn, R.G., Lee, C.A., Schouwstra, R.P. and Mellowship, P. (2002). The relationship between PGE and PGM in the Bushveld Complex. The Canadian Mineralogist, Vol.**40**. 311 - 328.

Chan, T.K. and Finch, I.J. (2001). Determination of Platinum Group Elements and Gold by Inductively Coupled Plasma Mass Spectrometry.1 – 9.

Clark, I. and Harper, W.V. (2000). Practical Geostatistics 2000.

Cousins, C.A. (1957). The chromitite seams of the Bushveld Complex and their economic potentialities. Unpbl. Internal Memorandum.

Cousins, C.A. (1969). The Merensky Reef of the Bushveld Igneous Complex, 239 – 251. *In*: Wilson, H.D.B., Magmatic Ore Deposits. Monogr. Econ. Geol. **4**. 366.

Du Toit, A.L. (1986). The Impala Platinum Mines. *In* Mineral Deposits of Southern Africa 2. (C.R. Anhaeusser and S. Maske, eds.) Geology Soc. S. Afr., Johannesburg, South Africa (1091 – 1106).

Du Toit, G.S., Neuland, E.W., Oost, E.J. and Begemann, E. (2001). Capital Investment Decisions, *Principles and Applications* – revised edition. 99 – 131.

Ferguson, J. and Botha, E. (1963). Some aspects of igneous layering in the basic zones of the Bushveld Complex. Trans. Geol. Soc. S. Africa, **66**, 259 – 78.

Gain, S.B. (1985). The geologic setting of the platiniferous UG2 chromitite on the farm Maandagshoek, eastern Bushveld Complex. *Econ. Geol*. **80**. 925 – 943.

Gain, S.B. (1986). The upper group chromitite layers at Maandagshoek, eastern Bushveld Complex. In Mineral Deposits of Southern Africa 2. (C.R. Anhaeusser & S. Maske, eds.) Geological Society of South Africa, Johannesburg, South Africa (1197 – 1208)

Gentry, D.W. and O' Neil, T.J. (1992). Mine Feasibility Studies, in *SME Mining Engineering Handbook* (The Society for Mining, Metallurgy and Exploration Inc: Colorado). 393 – 404.

Geological Society of South Africa. (1969). Symposium in the Bushveld Igneous Complex and other layered intrusions, Special Publication No.**1**

Gilmour, B. (2005). Automotive Sector – the driver of future PGM demand, PDAC.

Gitman, L.J. (2003). Principles of Managerial Finance 10th edition. 467 – 489.

Golenya, F. (2004). Mineral Resource/Reserve Estimation- No. 20 Shaft Base Case – *in house report* 2 – 39.

Golenya, F. (2004). Geology and Valuation Report – 20 Shaft Project. Vol. **1** – Principal Report. 3, 4, 124 - 127

Haffty J., Riley L. B. and Goss W. D. (1977). A Manual on Fire Assaying and Determination of the Noble Metals in Geological Materials. U.S.Geological Survey Bulletin **1445**.

Hahn, U.F. and Ovendale, B. (1994). UG2 chromitite layer potholes at the Wildebeestfontein north mine, Impala Platinum Limited. *Proc.* 15th CMMI Congress, S. Afr. Inst. Mining Metall.195 - 200

Hiemstra, S.A. (1985). The distribution of some platinum group elements in the UG2 chromitite layer of the Bushveld Complex. Econ. Geol**. 80**, 944 – 957.

Joint Ore Reserves Committee of the Australian Institute of Mining and Metallurgy. (1989) The Australian Code for Reporting of Mineral Resources and Ore Reserves (The JORC CODE). 683 – 698.

Kerzner, H. (2001). Project Management. A Systems Approach to Planning, Scheduling, and Controlling. 7th Edition. 903 – 938.

Knoetze, P. (1996). Mining strategy for the UG2 Chromitite Layer at No. 12 Shaft. 2 – 4.

Laird, A.M. (1997). How to Develop a Project, in *Proceedings Mindev 1997 – The International Conference on Mine Project Development*. 3 – 9.

Lee, C.A. (1996). A review of mineralization in the Bushveld Complex and some other layered intrusions. *In* Layered Intrusions (R.G. Cawthorn, ed.) Elsevier, Amsterdam, The Netherlands. 103 – 145.

Lenahan W. C. and Murray-Smith R. (1986). Assay and Analytical Practice in the South African Mining Industry. SAIMM.

Mackenzie, B.W. (1982). Economic Guidelines for Exploration Planning. Department of Geology. Rhodes University. Grahamstown. 39 – 59, 417 – 489.

Matthey, J. (2005). SA Mining – January 2005.

Mining Weekly, (2005). Mining Projects. 88 – 103.

McCarthy, P.L. (1997). Mining Dilution and Losses in Underground Mining in *Proceedings Resource to Reserves Inputs Seminar,* The Australasian Institute of Mining and Metallurgy: Melbourne. 5 – 8.

McCarthy, P.L. (2001). Cost Estimation, Internal AMC Paper. 333 – 336.

McLaren, C.H. (1978). The UG2 layer of the Bushveld Complex comes under scrutiny. Metals and minerals processing.

McLaren, C.H. and de Villiers, J.P.R. (1982). The platinum group chemistry and mineralogy of the UG2 chromitite layer of the Bushveld Complex. *Econ. Geol* **77**. 1348 - 1366.

McOuat, J. (1991). The Prefeasibility Study Assessment, in *Proceedings Sixth Mineral Economics Symposium* (Canadian Institute of Mining, Metallurgy and Petroleum: Montreal)

Monei, V. (2004). Potholes in Impala Platinum Mines – in house report. 6.

Morrissey, C.J. (1988). Exploration for platinum. In: Prichard, H.M., Potts, P.J., Bowles J.F.W. and Cribb, S.J. (eds.), Geoplatinum **87**. Elsevier, 1 – 12.

Naldrett, A.J., Cameron, G., von Gruenewaldt, G. and Sharpe, M.R. (1987). The formation of stratiform platinum group element deposits in layered intrusions. 313 – 397.

Nicholas, J.M. (2001). Project Management for Business and Technology. Principles and Practice, 2nd Edition. 306 – 326.

Odendaal, N.J. (1992). Platinum Group Metals. *South Africa's Mineral Industry*. Johannesburg, Minerals Bureau. Dept. of Minerals and Energy. Johannesburg, Minerals Bureau. Dept of Mineral and Energy Affairs.

Pahla, T. (1997). Definition of ore at Impala Platinum Mines, Rustenburg, South Africa – in house report. 3 – 5.

Rozman, L.I. (1998). Measuring and Managing the Risk in *Resources and Reserves in Proceedings Ore Reserves and Finance,* The Australasian Institute of Mining and Metallurgy: Melbourne. 43 – 55.

Schmidt. E.R. (1952). The structure and composition of the Merensky Reef and associated rocks on the Rustenburg platinum mine. Trans. Geol. Soc. S. Africa. **55**. 233 – 279.

Shillabeer, J.H. (2001).Lessons Learned Preparing Mining Feasibility Studies in Mineral Resource and Ore Reserve Estimation – The Aus IMM Guide to Good Practice*,* The Australasian Institute of Mining and Metallurgy. 435 – 440.

Slabbert, M.J. (1994). An Evaluation of the UG2 Chromitite Layer at Bafokeng North Mine, Bulletin No. **435**. Addendum No. **1** – *in house bulletin*. 8 – 13.

South African Mineral Resources Committee. (2000). South African Code For Reporting Of Mineral Resources and Mineral Reserves. (The SAMREC CODE)

Stephenson, P.R. and Vann, J. (2001). Common Sense and Good Communication in Mineral Resource and Ore Reserve Estimation – *The AusIMM Guide to Good Practice* (The Australasian Institute of Mining and Metallurgy: Melbourne). 13 – 20.

Sundquist, L. (2001). Information about Fire Assay Correction Factors.

Taylor, H.K. (1997). Mine Valuation and Feasibility Studies, Mineral Industry Costs, 2nd edition, Northwest Mining Association.

Theron, J. (2003). Second phase UG2 progress at advanced stage of planning – mining Weekly - in *house bulletin*.2 - 3.

Thomas, C. (1993). A discussion on the methodology for integrating geological surface and underground borehole data into the Geovaluation System on Impala. *Bull. Geol. Dept. Impala Platinum Ltd* (unpubl.) **450**. 4.

Verbaan, B. (2004). Financial Management & Project Investment Analysis. 72 – 102.

Vermaak, C.F. (1985). The UG2 layer – South Africa's slumbering chromitite giant: Chromium Review **5**. 9 – 23.

Vermaak, C.F. (1995). The Platinum Group Metals. Publ. Mintek, Johannesburg.

Viljoen, M.J. and Schurmann, L.W. (1988). Platinum - group metals. *In* The Mineral Resources of South Africa (M.C.G. Wilson and C.R. Anhaeusser, eds). Council for Geoscience, Pretoria, South Africa (532 – 568)

Viljoen, M.J. and Schurmann, L.W. (1998). Platinum Group Metals *in* The Mineral Resources of South Africa (M.G.C. Wilson and C.R. Anhaeusser, eds): Handbook, Council for Geoscience **16**. 532 –556.

Von Gruenewald, G. (1977). The mineral resources of the Bushveld Complex: Minerals, Science and Engineering **9**, 83 – 95.

Von Gruenewald, G. (1979). A review of some recent concepts of the Bushveld Complex, with particular reference to sulphide mineralization. Can. Mineral. **17**. 1244 - 1269

Von Gruenewaldt, G., Hatton, C. J., Merkle, C.J. and Gain, S.B. (1985). Platinum chromitite associations in the Bushveld Complex. Econ. Geol. **81**. 1067 – 1079.

Wagner, P.A. (1929). The Platinum Deposits and Mines of South Africa. Oliver and Boyd, Edinburgh, U.K. 326.

Walraven, P. (1987). Geochronological and isotopic studies of Bushveld Complex rocks from the Fairfield borehole at Moloto, northeast of Pretoria. South Africa Jour. of Geol **90**. 352 – 360.

White, M.E. (2001). Feasibility Studies – Scope and Accuracy, in *Mineral Resource and Ore Reserve and Estimation – The AusIMM Guide to Good Practice.* The Australasian Institute of Mining and Metallurgy: Melbourne. 421 – 434.

Willemse, J. (1969). The geology of the Bushveld Complex, the largest repository of magmatic ore deposits in the world. Econ. Geol. Mon. **4**. 1 – 22.

Young, G.J. (1946). Elements of mining 4th edition.

Appendices

Appendix I – Stratigraphic nomenclature used at Impala Platinum Mines (Leeb – Du Toit, A. 1986)

Below are the descriptions of the different units which constitute the informal stratigraphic column (Upper Critical Zone) as used on Impala Platinum, with special reference to their occurrence within the No. 12 Shaft area.

Footwall 16

This unit forms the immediate footwall to the UG1 Chromitite and is typically used for the footwall drive development in the cases of the UG2 Reef being mined.

The upper portion of Footwall 16 is a mottled anorthositic layer, containing numerous thin chromitite layers towards the upper 2.5m. These chromitite stringers are irregular and vary in thickness from a few millimeters to several centimeters. The stringers often create poor ground conditions especially when it occurs in the immediate hangingwall above the drives.

UG1 Chromitite

The UG1 Chromitite Layer rests on Footwall 16 and is known to split into two or more layers ranging in thickness from 0.85m to 1.0m with lens – like layers of either anorthosite or pyroxenite between them. This unit has no mineralization of economic importance. However it poses as a potential fracture plane along which the rocks easily separate.

UG1 Pyroxenite

This unit is found overlying the UG1 Chromitite and is a brownish green pyroxenite which ranges from 3.5m to 6m in thickness. Its physical appearance is similar to the pyroxenites of the Merensky and Bastard Reefs.

Footwall 13 norite

The thickness of this unit is in the order of 4.2m to 8.6m with a characteristic 1cm basal chromitite layer.

Footwall 13 norite consists of uniform, brownish – grey, medium to coarse grained, spotted anorthosite, made up of white anorthite with dark pyroxene 'spots.' It is the footwall to the UG2 Pegmatoid.

UG2 Pegmatoid

This unit ranges in thickness from 20 – 45cm as encountered in boreholes drilled into the No. 12 shaft block. This is a very coarse grained rock consisting of large dark brown pyroxene crystals and minor interstitial white anorthite. Stratigraphically, this rock lies directly below the UG2 Chromitite Layer and above Footwall 13. In general, it has been observed that the absence of this unit is indicative of potholed or rolling reef.

UG2 Chromitite Layer

This is a black, fine grained rock, consisting almost entirely of chromite grains. It is generally between 55 and 67cm thick displaying a sharp upper contact with the UG2 Pyroxenite and an irregular gradational lower contact with the underlying pegmatoid.

UG2 Pyroxenite

This rock is dark brown, coarse to medium grained, hard and competent which lies directly above the UG2 Chromitite Layer. It comprises predominately pyroxenes and varies between 5m to 6.5m in thickness.

The Intermediate Chromitite Layer (ICL) and Leader Chromitite Layers (Triplets)

These chromitite layers occur within the UG2 Pyroxenite. The Leader Chromite Layers typically consists of 3 layers varying from 5 to 20cm in thickness and are inter -layered with disseminated chromite bearing pyroxenite. These 'Triplets' as they are locally known are typically developed over 70cm to 90cm. The ICL has not been encountered in all the borehole intersections. However, when present it occurs between the Triplets and the UG2 Chromitite Layer. The ICL is a single approximately 0.5cm to 4cm thick chromitite layer.

Footwall 12

This footwall is a medium to coarse grained mottled anorthosite comprising whitish anorthite containing irregular darker pyroxene mottles. This unit forms the immediate hangingwall to the UG2 Chromitite Layer and is rarely exposed in the UG2 workings other than above potholed reef or as a result of faulting. This lithology ranges in thickness from 8 to 12m with a characteristic 1cm basal chromitite layer.

Footwall 11

The mottled Footwall 12 grades into a spotted anorthosite, which is known as Footwall 11. Footwall 11 has an average thickness of 5m.

Footwall 10

Being a typical Harzburgite, Footwall 10 has a very distinctive layered appearance. It is about 0.50m thick.

Footwall 9

This unit is a very persistent, mottled anorthosite with an approximate thickness of 2.2m.

Footwall 8

Footwall 8 is a spotted anorthosite in the order of 2.5m.

Footwall 7

This is an anorthositic norite and varies in thickness from 50m in the south of the lease area to 36m at No. 12 Shaft. Footwall 7 is clearly identified by its marker horizon locally known as the Olivine Platy Layers (OPLs). The OPLs form a very distinctive layer of dark greenish black serpentinized olivine and clinopyroxene within a lighter colored brown norite. OPLs vary in thickness from 20cm to 140cm and are potential parting planes along which the rocks break. The contact between Footwall 7 and the overlying Footwall 6 is sharp and visibly very distinctive.

Footwall 6

This mottled, anorthositic layer varies in thickness from 5cm to 1m as observed in the underground workings within the No. 12 Shaft area. A 1cm chromitite stringer is the distinctive marker zone in Footwall 6. The chromitite stringer is regarded as a geological hazard and results in poor hangingwall conditions.

Footwall 5

This unit consists is a uniform anorthositic norite layer of approximately 1.2m in thickness. Footwall 5 appears pyroxenitic in hand specimen.

Footwall 4
This unit is not well developed in the northern shafts of Impala Platinum. It appears rather as a shear plane as observed in the underground workings of the No. 12 Shaft area.

Footwall 3
Footwall 3 is an anorthositic norite with a thickness of about 1m and appears very similar to other spotted footwall lithologies.

Footwall 2
Within the No. 12 Shaft area, Footwall 2 is of a very distinctive nature. It occurs as a 1 – 2cm thick mottled, anorthosite layer. It is often used as a marker zone to distinguish between footwalls 1 and 3, which are very similar in hand specimens.

Footwall 1
This unit is between 3m to 5m thick. Footwall 1 is noritic and grades upwards into an anorthositic norite. The overlying Footwall 1a is a 0.2m to 1m thick mottled variety of Footwall 1 as identified within the No. 12 Shaft area.

Merensky Reef
This is the portion of the Merensky Unit and underlying footwall, which can be mined for platinum group elements. Three types of Merensky Reef can be identified depending on the footwall unit directly underlying the reef.

Merensky Pyroxenite
This rock is coarse to medium grained, dark brown, hard and competent which lies directly above the Merensky Chromitite Layer. It comprises predominately pyroxene crystals and varies between 1.2m to 1.8m in thickness.

Middling 1 – is a norite of 20 – 80cm thickness forming the hangingwall to the Merensky Pyroxenite.

Middling 2 – is a spotted anorthosite of approximately 6 – 8m in thickness.

Middling 3 – is a white large mottled anorthosite ranging in width from 3 – 4m.

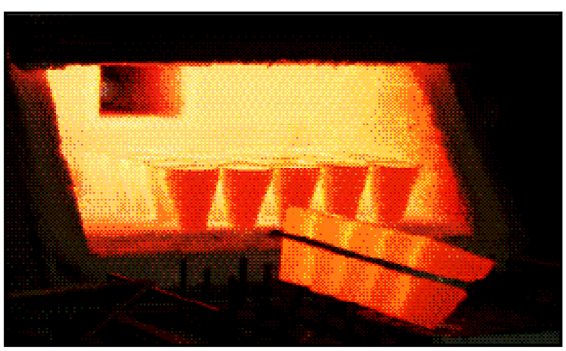
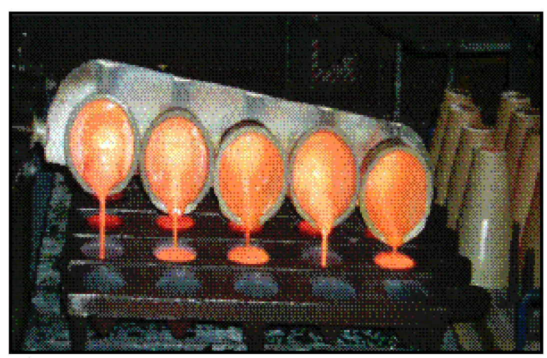
Bastard Reef/Pyroxenite – is a pyroxenite layer with no economic mineralization and lies about 10 to 13m above the Merensky Unit.

Appendix II – Assay Techniques (Chan and Finch, 2001).

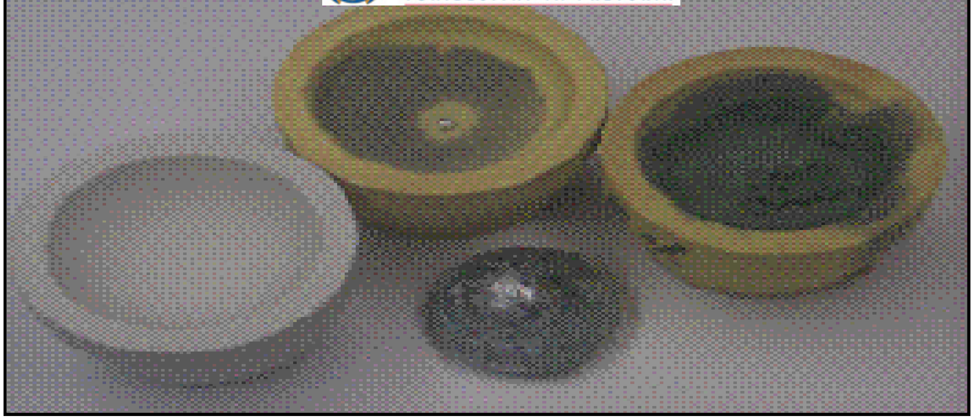
Fire Assay Lead Collection Technique (Chan et al, 2001).
The basic principle of any fire assay technique using lead as the collector is that a certain amount of sample must be intimately mixed with a chemically compatible flux (comprising litharge (PbO), soda ash, borax, silica, flour and silver). The mixture is then fused at approximately 1050^{0}C for about 40 minutes during which time the litharge is reduced by the flour to elemental lead which collects the precious metals. The gangue reacts chemically with the rest of the flux components to form slag, which can be separated from the lead fairly easily once the 2 phases have cooled. When fusion is complete, the pot is removed from the furnace and the melt is poured into a conical cast – iron mould.

The lead forms a button at the bottom of the container which is allowed to cool and is then cleaned of slag. The lead button is then knocked into a cubic shape, placed into a pre – heated cupel and heated in a furnace set at 950^{0}C. The cupellation process in which lead is converted to lead oxide, which immediately percolates into the cupel leaving a dore bead or prill of precious metals takes about 30 to 40 minutes. The cupel is then removed from the furnace and slowly cooled to room temperature.

The prill must then be exposed to about 1300^{0}C in a high temperature cupellation furnace, so as to drive off the residual Pb and Ag. The period of time required for this event to occur can vary from 20 minutes, 90 minutes to 6 hours during which time the prill is left in the high temperature furnace to achieve its end, the composition of which will vary depending on the cupellation time. The final prill is then weighed and its mass related to the aliquot of ore sample taken to obtain the ppm or g/ton value.

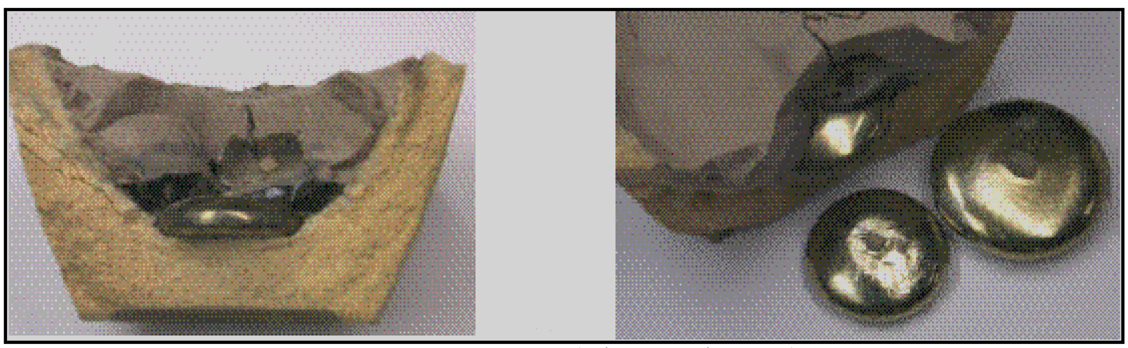
Fusion melt poured into mould (Chan et al, 2001).



Cupels, prill and Pb button (Chan et al, 2001)

Fire Assay Nickel Sulphide Collection Technique (Chan et al, 2001).
This technique is extremely efficient for collecting all the PGE's and Au. The ground geological material is thoroughly mixed with a flux consisting of a mixture of soda ash, borax, silica, sulphur and nickel carbonate ($NiCO_3$) or nickel oxide (NiO). This charge is fused in a new fireclay pot in a gas fired fusion furnace set at a temperature of 1200^0C for a period of 75 minutes. During fusion, the sulphur combines with the nickel carbonate or nickel oxide to form nickel sulphide liquid which collects the PGE's and Au sulphides formed at the same time.

The nickel sulphide button obtained is cooled, cleaned of slag and weighed. It is then ground to a nominal particle size of 80% less than 75µm in a ring mill dedicated for this purpose. The pulped NiS is stored in an air tight plastic vial to prevent moisture absorption.



NiS button formed and slag layers above (Chan et al, 2001).

Separation of PGE and Gold (Chan et al, 2001).
A weighed portion of the NiS pulp is then treated with hydrochloric acid at about 100^0C to dissolve the NiS. The undissolved PGE and Au sulphides are then collected on a cellulose nitrate membrane filter. The membrane filter containing the PGE and Gold sulphides is digested with aqua regia in a sealed borosilicate test tube. The resultant solution is diluted with nitric acid and thoroughly mixed. The final solution is then analyzed for ruthenium, rhodium, palladium, osmium, iridium, platinum and gold using the Inductively Coupled Plasma Mass Spectrometry (ICP – MS) technique.

Recovery for all the PGE's is quantitative. Gold is sometimes lost due to the slight solubility of gold sulphide in hydrochloric acid.

The Fire Assay Method (Chan et al 2001)

- Before heating in an oven, a weighed sample is mixed with flux reagents in fireclay crucibles.
- Fire assay melts (fuses) this powered rock/flux mixture to generate a reaction which separates precious metals from gangue minerals. These reactions are usually designed to finish at 1300^0C.

  Fusion combines high temperature acid-base and reduction reactions in a design which balances:

- reaction temperature patterns through full (fluid) melt
- chemical release reactions
- composition of reaction products
- and other reaction factors for optimum precious metal recovery.

  Optimum recovery requires that 100% of precious metals collect when a rain of lead metal descends to the bottom of the melt during fusion.

- After fusions are poured and cooled, a glassy slag (top layer) and lead button (bottom layer) forms.

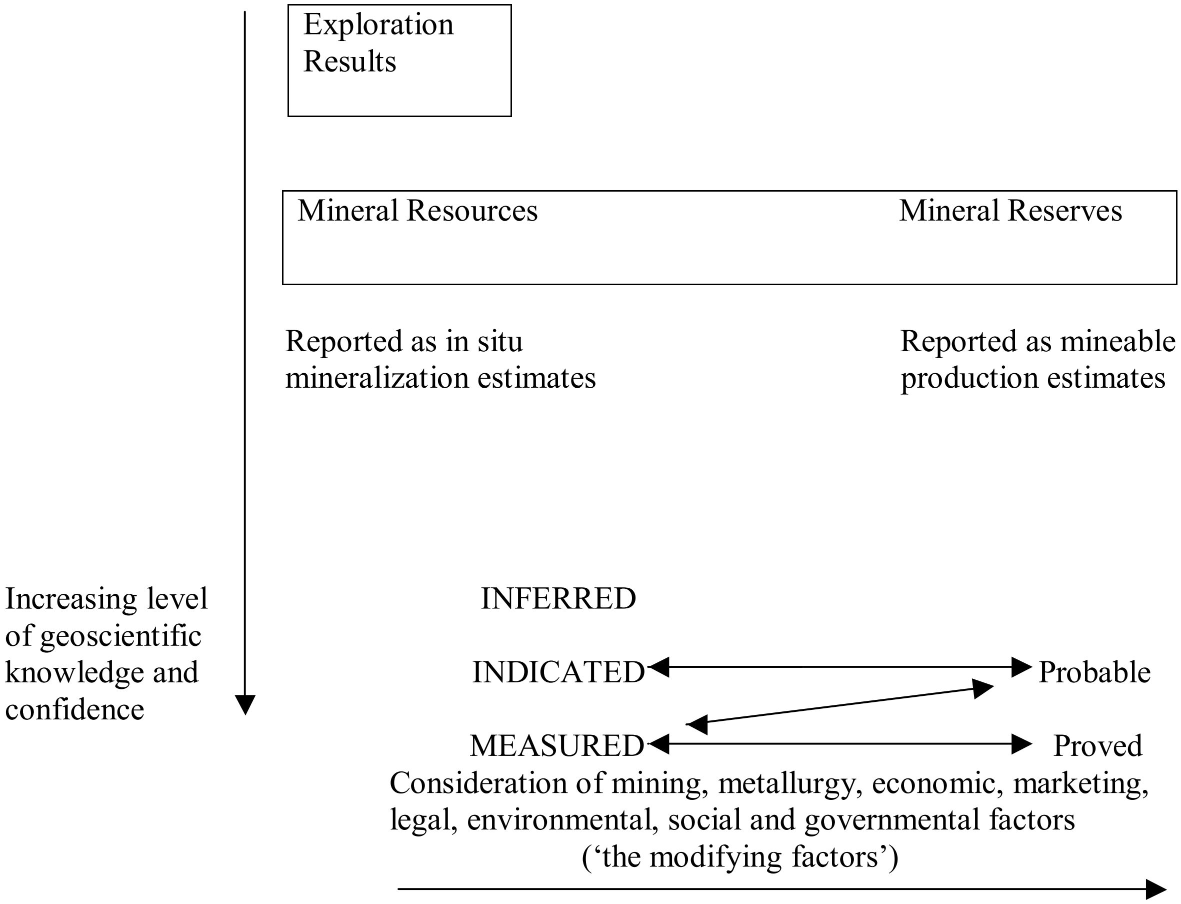
  With correct assay design, all of the precious metals alloy in the lead button and the waste forms the slag. Ideal slags shatter upon cooling to separately cleanly from the lead.

- The slag is broken and discarded; the lead reaction product (button) is 'cupeled' to remove lead and any remaining base metal impurities.
- cupellation uses oxidation to remove impurities according to the following reaction: $2PbO + O_2 \rightarrow 2PbO$
- oxidized products (lead and limited amounts of other base metals) absorb into a cupel; non – oxidized metal does not absorb.
- cupellation has only a limited capability to accurately cope with impurities; the bulk of impurities must be removed by correct fusion design.
- Recovered precious metal is measured by either: (a) Parting and Weighing – selectively dissolving precious metals then weighing the precious metal products (gravimetric finish) to calculate concentrations of gold, silver and other precious metals.

  (b) By digesting the product and measuring metal concentration with modern instruments such as Atomic Absorption, or Inductively Coupled Plasma (ICP) methods.

Appendix III – (applicable sections of SAMREC, 2000).

The SAMREC Code which shows the relationship between Mineral Resources and Mineral Reserves is diagrammatically outlined below.



Mineral Resource- refers to material of economic importance such that its quality and quantity warrants reasonable and realistic prospects for eventual extraction. Geoscientific information is used in order to evaluate the deposit in terms of quality and quantity.
Mineral resources are subdivided into categories based on the level of confidence in respect of geoscientific evidence. These classes are Inferred, Indicated and Measured Reserves.

Inferred Resource – is that portion of a resource for which tonnage, grade and mineral content is estimated with a low level of confidence. The inference is derived from information obtained from outcrops, trenches, pits workings and drill holes.

Indicated Resource – here the tonnage, densities, shape physical characteristics, grade and mineral content is estimated with a reasonable level of confidence. These parameters are based on exploration, sampling and testing information obtained from outcrops, trenches, pits workings and drill holes.

Measured Resource – tonnage, grade, shape and mineral content is estimated with a high degree of confidence. The data is based on detailed and reliable exploration and sampling methods. Owing to the level of detail, one is able to confirm geological grade and continuity.

Mineral Reserve – is the economically mineable material derived from a measured and/or indicated resource. This category requires careful consideration of factors affecting extraction such as mining method, metallurgical aspects, economic, marketing as well as legal, environmental, social and governmental factors.

In some cases, measured mineral resources could convert to probable mineral reserves due to uncertainties associated with the modifying factors.

Probable Reserve – is economically mineable material derived from a measured and/or indicated mineral resource. It is estimated with a lower level of confidence as compared to the proven mineral reserve. It is inclusive of diluting materials and allows for losses which may occur during mining operations.

Proved Reserve – this is economically mineable material derived from a measured mineral resource. It is estimated with a high level of confidence.

Appendix IV- Rock engineering pillars (Budavari, 1983).

Rock Engineering Losses occur in the form of:

Bracket Pillars
All the geological structures e.g. faults and dykes are regarded as potential hazards to mining operations and as such, for the reasons of safety, will require clamping pillars to be left on either side of the feature. The physical dimensions of these pillars can only be confirmed upon the intersection of the feature in the underground workings. However, it must be noted that a loss in potential reserves will be anticipated.

Shaft Boundary Pillars
These pillars are a legal requirement and are put in place in order to regulate ventilation, provide adequate protection against the inrush of water from one shaft into the next as well as to ensure that the pillars inner core remains intact and solid. The pillars are left on either side of the actual boundary.

Shaft Protection Pillar
A shaft is one of the major capital investments of a mine and therefore elaborate steps must be taken to ensure its unhindered operation throughout the shaft's productive life. The rock mechanics and mining engineers have decided that a conventional shaft pillar will be left in place. Depending on the stress regimes, it may be decided to extract the shaft pillar or portions thereof at a convenient stage. However for the purpose of ore reserve calculations, the reef constituting the pillar has to be written off for the foreseeable future. This loss in reserves must be accounted for in the extraction rate.

Barrier Pillars
These are regionally extensive pillars which span the entire shaft block contained within the designated shaft boundaries. From the entire category of prescribed rock engineering pillars, the barriers pillars appear to offer the most flexibility in terms of relocation and physical dimensions. Often, as the stoping operations progress, natural pillars e.g. potholes, dykes are encountered or created e.g. areas abandoned due to poor ground conditions. Ideally these features tend to substitute for planned pillars in the original mine design.

Grid Pillars
These pillars are commonly left as 'in-stope'pillars and form a crucial part of the mine design support pattern. Depending on the type of stoping layout, these pillars are fairly fixed and not intended to be mined.

Appendix V – Financial evaluation criteria (Du Toit, et al 2001).

The following financial criteria are commonly used to evaluate projects:

The Payback Period

This is the expected number of years in which the initial investment of a capital project is recovered from the projects net cash flows after tax.

The payback period serves as a basis or criterion for the acceptance or rejection of investment projects. The maximum acceptable payback period is usually included in the investment policy of a firm.

Generally speaking, the shorter the payback period of a project, the less the uncertainty about the recovery of the investment and the better the liquidity in terms of cash being available for alternative investment opportunities.

The advantages associated with the payback period technique are:

- The technique is simple to use.
- It serves as a criterion of risk if it is assumed that risk (e.g. risk of technological obsolescence) increases over time.
- It serves as a criterion of liquidity, because the faster the initial investment is recovered, the sooner the generated cash is made available for alternative use.

However this technique also has several shortcomings such as:

- It does not consider the cash flow after the payback period has been reached and is thus not a reliable measure of overall project profitability. The emphasis here is on short – term profitability rather than profitability over the entire life of the project
- It does not take into account the time value of money, despite the fact that different cash flow patterns may have a major influence on the total value of the project
- It does not consider the cost of capital in any way
- It makes no distinction between projects of different sizes or those with different capital requirements
- There is no theoretical basis whatsoever for a maximum acceptable criterion. The payback period is generally determined arbitrarily by a company's management as part of its investment policy.

Despite its disadvantages, the payback period is often used for the initial screening of investment proposals for the purpose of ranking projects according to liquidity and risk.

The Net Present Value (NPV)

This is the difference between the present value of all expected net cash inflows and the present value of all expected net cash outflows (initial investment) calculated over the expected life of the project.

The successful application of this evaluation method requires the following established parameters:

- An appropriate project life.
- Accurate estimation of all relevant after – tax net cash flows.
- Appropriate discount rate (cost of capital).
- Differences between the present values of all cash inflows and outflows based on the discount rate.

Generally, decisions based on the NPV of projects are as follows:

- **Accept all projects with a NPV>0.** In these projects, the initial investment amount and the cost of financing is recovered from the project cash flows. Furthermore the project helps increase the value of the firm by creating a positive NPV.

- **Reject projects which show a NPV<0**. A negative NPV means that the project is not generating sufficient cash to cover all expenses, the interest payments and the initial investment associated with the project. If such a project is accepted, the value of the firm would by decreased by an amount equal to the negative present value.

- **Reject projects with NPV = 0.** In this case, the net cash flows of the project discounted at a rate equal to the firm's cost of capital are just sufficient to recover the total investment amount and the financing costs, making no contribution to the value of the firm.

The Profitability Index (PI)
The profitability index is the ratio of the present value of all net cash inflows to the present value of all net cash outflows, where the latter normally represents the initial investment.
The PI is a measure of a project's profitability relative to each R1 invested in the project. Using only the PI as a decision criterion could have a negative impact on potentially valuable projects because the PI is essentially a ratio and does not consider the size or extent of a project.
If the PI is used as decision making criteria, the following basic rules should apply:

- Accept all projects where PI >1.
- Reject projects which have a PI < 1.
- Projects with PI = 1 make no contribution to the value of the firm and should be rejected.

The Internal Rate of Return (IRR)

The internal rate of return (IRR) is the discount rate that equates the present value of expected net cash inflows and the present value of net cash outflows (normally the initial investment) resulting in NPV = 0.

When the IRR is used as the criterion for financial evaluations, the rules are as follows:

- **IRR = k**, where k = the cost of capital (rate of interest).

Where IRR = k, it means that the initial investment and the financing costs concerned are recovered in full, but the project does not add any additional value to the firm. This is equivalent to the case where the NPV = 0.

- **IRR > k**

In this case the project will in addition to the recovery of the investment amount and the financing costs involved, add value to the company. This is equivalent to the case where NPV > 0.

- **IRR< k**

An IRR < k implies that the initial investment and the financing costs are not fully recovered. Accepting such a project would bring about a decrease in the existing value of the company. This is equivalent to a NPV < 0

In summary, the NPV, PI, and IRR all lead to the same accept – reject decisions.
All projects showing a NPV > 0, PI >1 and IRR > k are acceptable and adds to the total worth of the company.

Appendix VI – Rand/Dollar Exchange Rates (http://www. oanda.com/convert/fxhistory).

Year	Month	Exchange Rate	Year	Month	Exchange Rate
1994	Jan -94	3.41	**1995**	Jan -95	3.54
	Feb -94	3.45		Feb -95	3.56
	Mar -94	3.46		Mar -95	3.60
	Apr - 94	3.59		Apr - 95	3.60
	May -94	3.63		May -95	3.64
	Jun - 94	3.63		Jun - 95	3.66
	Jul - 94	3.67		Jul - 95	3.64
	Aug- 94	3.60		Aug- 95	3.64
	Sep - 94	3.56		Sep - 95	3.66
	Oct -94	3.54		Oct -95	3.65
	Nov-94	3.53		Nov-95	3.65
	Dec-94	3.56		Dec-95	3.66
1996	Jan -96	3.64	**1997**	Jan -97	4.64
	Feb -96	3.75		Feb -97	4.46
	Mar -96	3.93		Mar -97	4.44
	Apr - 96	4.22		Apr - 97	4.44
	May -96	4.37		May -97	4.47
	Jun - 96	4.35		Jun - 97	4.50
	Jul - 96	4.40		Jul - 97	4.56
	Aug- 96	4.53		Aug- 97	4.56
	Sep - 96	4.50		Sep - 97	4.69
	Oct -96	4.58		Oct -97	4.85
	Nov-96	4.66		Nov-97	4.86
	Dec-96	4.69		Dec-97	4.86
1998	Jan -98	4.94	**1999**	Jan -99	5.95
	Feb -98	4.94		Feb -99	6.12
	Mar -98	4.98		Mar -99	6.20
	Apr - 98	4.90		Apr - 99	6.11
	May -98	5.05		May -99	6.18
	Jun - 98	5.27		Jun - 99	6.08
	Jul - 98	6.23		Jul - 99	6.10
	Aug- 98	6.32		Aug -99	6.12
	Sep - 98	6.12		Sep - 99	6.05
	Oct -98	5.81		Oct -99	6.10
	Nov-98	5.66		Nov -99	6.13
	Dec-98	5.91		Dec -99	6.14

Year	Month	Exchange Rate	Year	Month	Exchange Rate
2000	Jan -00	6.12	**2001**	Jan -01	7.77
	Feb -00	6.31		Feb -01	7.81
	Mar -00	6.46		Mar -01	7.88
	Apr - 00	6.64		Apr - 01	7.88
	May -00	7.02		May -01	7.97
	Jun - 00	6.91		Jun - 01	8.05
	Jul - 00	6.88		Jul - 01	8.19
	Aug- 00	6.95		Aug- 01	8.30
	Sep - 00	7.16		Sep - 01	8.65
	Oct -00	7.47		Oct -01	9.28
	Nov-00	7.68		Nov-01	9.73
	Dec-00	7.64		Dec-01	11.64
2002	Jan -02	11.55	**2003**	Jan -03	8.68
	Feb -02	11.46		Feb -03	8.27
	Mar -02	11.48		Mar -03	8.03
	Apr - 02	11.05		Apr - 03	7.67
	May -02	10.12		May -03	7.65
	Jun - 02	10.13		Jun - 03	7.87
	Jul - 02	10.07		Jul - 03	7.53
	Aug- 02	10.56		Aug- 03	7.38
	Sep - 02	10.57		Sep - 03	7.29
	Oct -02	10.29		Oct -03	6.95
	Nov-02	9.62		Nov-03	6.72
	Dec-02	8.93		Dec-03	6.49
2004	Jan -04	6.94	**2005**	Jan -05	5.95
	Feb -04	6.74		Feb -05	6.01
	Mar -04	6.62		Mar -05	6.03
	Apr - 04	6.54		Apr - 05	6.15
	May -04	6.76		May -05	6.33
	Jun - 04	6.39		Jun - 05	6.75
	Jul - 04	6.10		Jul - 05	6.69
	Aug- 04	6.45		Aug -05	6.45
	Sep - 04	6.53		Sep - 05	6.48
	Oct -04	6.38		Oct -05	6.51
	Nov-04	6.04		Nov -05	6.54
	Dec-04	5.72		Dec -05	6.57