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The viability of the Kalplats Platinum Group Element deposit

By

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ABSTRACT

The Kalplats platinum group metal deposit is located in the Northwestern Province of South Africa, 80 km southwest of Mafikeng, in the Stella Layered Intrusion. The Stella Layered Intrusion intruded into the Kraaipan Greenstone Belt and is dated at 3.03 billion years. The Kraaipan Greenstone Belt is host to the 80 000 ounce per annum Kalgold Gold Mine.

Platinum Group Element mineralized layers in the Stella Layered Intrusion are interpreted to occur in the overturned western limb of folds, formed by an eastward vergent compressional event. Three major reefs have been identified, namely the Lower Grade (LG) reef, the Mid Reef and the Main Reef. High grade reefs occur within these three. The average Pt:Pd ratio of the Main Reef is 1:1. Highest total precious metals content is concentrated in the Upper Main and Lower Main Reefs and the average grade for these two reefs is 4g/t. Open pit mining is suggested.

The total inferred precious metals resource equates to 84Mt at an average grade of 1.4 g/t Pt+Pd+Au, for 3.9million ounces. Platinum, palladium and gold occur as fine grains. Maximum recoveries of approximately 72% are possible, from sulphide ore, using a two-stage mill-float circuit. The estimated reserve (non-JORC compliant) is 26 Mt at an average grade of 2.01g/t Pt+Pd+Au, for 1.68 million ounces.

A financial evaluation was done on the viability of the Kalplats deposit, using a discounted cash flow model. Future projections used were a R/$ exchange rate of R6-50 to the dollar and long-term metals prices of US$ 800/oz Pt, US$ 200/oz Pd and US$ 400/oz Au. The result of the discounted cash flow model was negative and indicated no return on capital and a negative Nett Present Value (NPV) of –R206 million at a discount of 13%.

Factors impacting negatively on the viability of the project, include grade, metallurgical recovery, smelter fees, government royalties, metals prices and the Rand-US$ exchange rate.

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Chapter 1 INTRODUCTION

1.1 Locality of the Project Area

The Stella Layered Intrusion is situated approximately 80km southwest of Mafikeng and 20 km north of Stella. A locality plan is indicated in Figure 1.1.
Topography is relatively flat, with magnetite quartzite units outcropping as low ridges. The area is covered by Kalahari sand, which varies in thickness from 1 to 8m depth.

Land usage is mainly agricultural and crops include sunflowers, maize and peanuts. Cattle farms are popular in the area, with a number of game farms in operation too. Summer rainfall is dominant with highest rainfall from December to March.

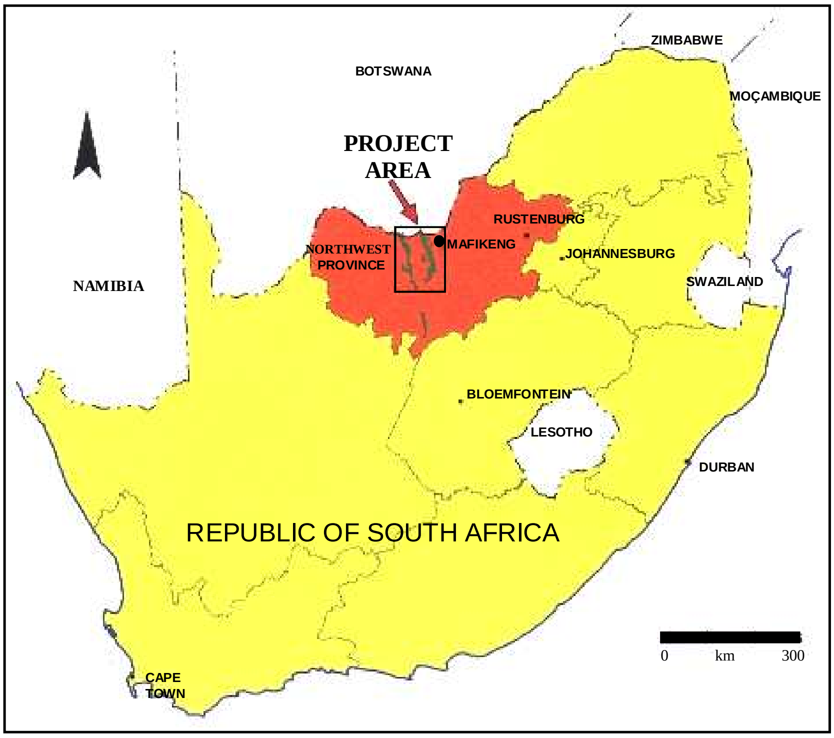

Figure 1.1: Locality of the Kalplats Project Area

1.2 Statement of the Problem

Determination of the economic viability of exploiting the Kalplats platinum-group mineral ("PGM") deposit and estimation of the current value of the project.

1.2.1 The Subproblems

a) What are the geological, dimensional (shape, size, tonnage), mineralogical and metallurgical characteristics of the ore bodies to be investigated in the study?
b) Which mining method(s) will be the most effective for mining these ore bodies?
c) What is the Nett Present Value of the project and how is the financial viability influenced by different risk-related factors?
d) What alternatives exist to mining the ore bodies?
e) How does this project compare to other similar projects?

1.3 The Delimitations

The study will not attempt to give an in-depth mineralogical or petrological review of the ore bodies.
The study will not attempt to design the complete mining operation, but will take the format of a study at pre-feasibility level.
The study is based on information obtained during exploration work programmes between 2000 and 2003.

1.4 Assumptions

a) Homogeneous distribution of mineralisation is present in the reefs. This was partially tested by comparing selected samples from the reefs in terms of grade.
b) Metallurgical test work as conducted on selected core and bulk samples is representative of ore to be treated in a metallurgical processing plant.
c) Future projections of metals prices and R-US$ exchange rate are accurate and in line with generally held industry views.

1.5 Importance of the Study

Given the exploration expenditure incurred to date, the objective of this study is to provide clear guidance in deciding whether further expenditure can be justified. This study may also provide a framework for the evaluation of similar projects.

Chapter 2 REVIEW OF THE RELATED LITERATURE

The exploration for new mineral deposits builds on our understanding of other similar deposits. In this chapter, other Platinum Group Element (PGE) deposits will be reviewed.

2.1. Ni-Cu-PGE sulfide Deposits

A review of literature by Maier (2001) revealed the following: deposits that contain Ni, Cu and PGE's are normally associated with mafic or ultramafic rocks. Those deposits that are more Cu and Ni rich (Norilsk and Uitkomst) contain more than ten percent sulfides, while the PGE dominant deposits (Bushveld Reefs, Great Dyke and Stillwater) contain less than ten percent sulfides.

Most Cu-Ni-PGE sulphide deposits are thought to have formed through segregation of an immiscible sulphide melt from a silica magma, in response to processes such as magma mixing, rapid cooling, differentiation and contamination (Naldrett et al, 1989, Li et al., 2001). When the sulfur content of the magma reaches saturation, small amounts of sulphide melt will segregate, at the same time as silicates crytallise from the magma. These sulphides and silicates are denser than the surrounding magma and will sink to the base of the magma chamber. Ni, Cu and PGE's will preferentially partition into the sulphide melts. Where sulfides are more concentrated, they form ores. The ores may be PGE reefs (<3% sulphide) or massive Cu-Ni-(PGE) sulphides.

Locating PGE-rich layers within a layered intrusion is often difficult, because the ore zones are generally thin compared to the thickness of the intrusions. Tectonic settings of sulphide ores include archaean greenstone belts, rifted plate margins and intrusions in cratonic areas (Maier, 2001)

2.2 Arctic Platinum – Cu-Ni-PGE deposits

Arctic Platinum consists of mining licences and claim blocks covering 305km^2 and is 100% owned by Gold Fields. According to the Gold Fields website (2005), a JORC classified resource of 12.6 Moz has been estimated at an average grade of 2.33g/t Pt+Pd+Au, using a 1g/t cut-off. The resource is contained in two projects, namely the Suhanko and SK Reef projects. Suhanko is the most advanced and contains an open pittable resource of 118.9 Mt.

The Cu-Ni-PGE mineralisation at Suhanko occurs in pyroxenite and gabbro layers of the Portimo Complex at the base of the intrusion. The resource at Suhanko is JORC classified as 7.5 Moz. The inferred resource at SK Reef is 5.1 Moz at 3.19g/t Pt+Pd+Au. The project is currently in the feasibility stage, but indications are that development of the project is dependent on an increase in the Pd price, as the average ratio of Pt:Pd is 1:4.

A summary of the resource in the Suhanko and SK Reef projects follows:

	Tons (Mt)	2PGE+Au g/t	Ounces (000)	Pd (g/t)	Pt (g/t)	Au (g/t)	Cu (%)	Ni (%)
Suhanko	118.9	1.97	7,538	1.47	0.35	0.15	0.23	0.09
SK Reef	49.4	3.19	5,064	2.45	0.67	0.07	0.10	0.08
Total	**168.3**	**2.33**	**12,601**	**1.76**	**0.45**	**0.12**	**0.19**	**0.09**

Table 2.1 Arctic Platinum Resource

Gold Fields announced, on 19 October 2005, the intention to form a joint venture with North American Palladium to further explore and develop the Arctic Platinum project. North American Palladium will be granted an option to acquire up to 60% interest in the project, for up to $45 million.

2.3 Russian Cu-Ni-PGE mining

Approximately 40% of annual global Pd production and 15% of Pt production is from Russian deposits. Historical production, reserve and sales figures are hard to come by as the information was deemed confidential under Russian law. Only recently (2004), a bill was signed to authorize the publication of this type of data.

Norilsk Nickel dominates Russian PGE production and as such will be discussed further. Norilsk has two PGE producing divisions, the Polar division, which operates seven Cu-Ni-PGE mines in northern Siberia and the Kola division, and process lower grade Ni-Cu deposits, with a small amount of by-product PGE's.

The deposits mined by the Polar division at the Norilsk-Talnakh mines, occur as large sheets associated with a sequence of layered igneous intrusions. Head grades vary between 10-11g/t PGE. The three types of ore that are mined are:

1) Massive Sulphides occurring in lens-shaped ore bodies, which are the richest in Ni, with PGE grades of between 12-14g/t and Pt:Pd of 3:1 to 4:1 and thickness between 1 and 40m. In general, Pt and Pd are enriched and Rh, Ru, Ir and Os depleted in the massive ore, when compared to the more disseminated ores.
2) Cu-rich ores, forming a halo around the massive sulphide ores, which are richer in Cu, with PGE grades up to the same levels as in the massive sulphide ores, and
3) Disseminated ores, associated with the intrusive bodies and less rich in metals content, with PGE grades of between 5 and 15g/t and thickness of 40 to 50m.

The Polar division consists of five underground mines on the Talnakh deposit and two open cast operations on the Norilsk-I deposit. PGE reserve and resource and production figures for the deposits are not yet publically available, but annual production was estimated by Johnson Matthey (2004) to be in the order of 2.7 Moz Pd, 650 koz Pt and 60 koz Rh. These figures are in line with the more recently published half-year results by Norilsk Nickel on their website, which indicated annualized production for 2005 of 3 Moz Pd and 700 koz Pt.

2.4 Uitkomst Complex, South Africa

Theart et al. (2001) described the occurrence of platinum group minerals in the Ni-Cu-Co-bearing ore of the Uitkomst Complex. The complex is regarded as a satellite intrusion of the Bushveld Complex and the Massive Sulphide Body (MSB) is currently being mined as a joint venture between African Rainbow Minerals (ARM) and LionOre. The layered ultramafic intrusion (Gauert et al., 1995) was dated at 2,044 Ma (De Waal et al.,2001), and is divided into the Main and Basal Groups. Both disseminated and massive mineralisation occurs.

The MSB comprises three zones: the Upper Stringer Zone, the Massive Sulphide Zone and the Lower Stringer Zone. PGE enrichment occurs in the western end of the MSB ore body and in the lower stringer zone. The average Pt:Pd ratio is 1:2.5 (Theart et al., 2001). The total reserve was estimated at 2.1 Moz PGE and resource at 4.6 Moz PGE.

2.5 Fiskenaesset Complex, Greenland

Crocket's (1981) studies of the Fiskenaesset Complex indicated that it is a layered Archean complex, which consists of anorthosite, gabbro, ultramafic units and chromitite. PGE concentrations are very low in the main part of the intrusion, but higher in the basal ultramafic rocks, in ultramafic lenticular channels, in chromitites and in rocks with disseminated sulphides.

The average Pt:Pd ratio and Pt and Pd values in different reefs are displayed below, as per samples by Page et al. (1980):

Layer Unit	**No samples**	**Pt (ppb)**	**Pd (ppb)**	**Pt:Pd (rounded)**
Ultramafic	16	4.9	10	1:2
Lower leucogabbro	14	12	11	1:1
Middle gabbro	5	10	5.1	2:1
Upper leucogabbro	12	4.2	1.1	4:1
Anorthosite	19	2.4	1.4	2:1

Table 2.2: Fiskenaesset Complex Resource

The age of the complex was determined at 2.82 billion years.

2.6 Stillwater Complex, USA

Information obtained from the Stillwater Mining Company website (2004) indicated the following. The Stillwater complex in Montana, dated at 2.7 Ga (De Paolo et al. (1979), is composed of a succession of mafic to ultramafic rocks in a large complex magmatic intrusion. Cooling of the complex was slow, giving rise to the formation of well-segregated layers of heavy mafic minerals and lighter siliceous noritic, gabbroic and anorthositic suites. The J-M Reef is one such layered sequence. The reef was deposited horizontally, but has been faulted and uplifted at angles of 50 to 90 degrees in the northern portion. The reef strikes along approximately 28 miles.

Mineralisation in the J-M Reef consists of Pd, Pt and minor Rh, as well as Fe, Cu, Ni and trace amounts of Au and Ag. The most recent proven and probable reserve (2003), declared 40.4 Mt at an average grade of 18.5 g/t, containing 23.6 Moz Pt+Pd. Two mines are operational on the J-M Reef, the Stillwater Mine and the East Boulder Mine. Pt:Pd ratios are in the order of 3.4:1 and 3.7:1 at the respective mines. A summary of the proven and probable reserves follows:

	Tons (000)	**Pt+Pd Grade (g/t)**	**Ounces (000)**
Stillwater Mine	17,480	20.53	11,460
East Boulder Mine	22,908	16.49	12,139
Total	**40,388**	**18.04**	**23,599**

Table 2.3: Stillwater Reserves

Although the deposit is more Pd-rich, underground mining of the J-M Reef is viable as a result of the high metals content contained in the reef. Norilsk (NMC) acquired a 56% interest in the Stillwater Mining Company in September 2003.

2.7 Great Dyke, Zimbabwe

The PGE resources in the Great Dyke comprise the Main Sulphide Zone and the Lower Sulphide Zone, according to Wilson et al. (2001). The Great Dyke is a layered intrusion, running roughly north-south through Zimbabwe for approximately 550km. Mineralisation occurs in four elliptical bodies with a total strike length of 350 km (Cawthorn, 1999). Mining occurs at both the Mimosa mine and the Ngezi mine. The age of emplacement of the Great Dyke was determined to be 2,580 Ma (Armstrong and Wilson, 2000). Resources in the Great Dyke are estimated at 2.6 Mt, containing 143 Moz Pt and 87 Moz Pd, from Vermaak (1995). Pt:Pd ratio is 1.5:1 and average grades are in the order of 3g/t PGE.

2.8 Bushveld Complex, South Africa

The 2,054 Ma old Bushveld Complex is the largest known layered intrusion in the world (Cawthorn, 1999). 75% of the world's Pt and 50% of the world's Pd resource is contained in the Bushveld Complex, according to Cawthorn. PGE's present in the Bushveld include platinum, palladium, rhodium, iridium, osmium and ruthenium, with Pt and Pd being most abundant. Mineralisation occurs in three different ore bodies contained in magnetite or chromitite layers, namely the Merensky Reef, the UG2 Reef and the Platreef. Pt:Pd:(other metals) ratios vary from 5:3:2 in the Merensky Reef, to 4:5:1 in the Platreef, to 5:3:2 in the UG2 Reef.

The more than 300km surface extent of the mineralized layers was explained by an ore forming model where magma was repeatedly injected into a shallow chamber and cooled slowly to form segregated mineral layers. Subsequent magmatic replenishment of the chamber led to a repetition of the crystallization sequence, giving rise to duplication of the mineralized layers. The Merensky and UG2 Reefs outcrop in the western and eastern limb of the Complex, while the Platreef is developed in the northern Potgietersrus limb.

Reserves and resources for the Bushveld Igneous Complex, as estimated by Crawthorn to a depth of 2km, are displayed below:

Region	Proven and Probable Reserves Moz (Tonnes)		Inferred Resources Moz (Tonnes)	
Northern Bushveld				
Platreef	9.9	11.3	136	136
	(308)	(351)	(4,230)	(4,230)
Eastern Bushveld				
Merensky Reef	10.9	4.8	286	165
	(339)	(149)	(8,896)	(5,132)
UG2 Chromitite	38	32.7	306	301
	(1,182)	(1,017)	(9,518)	(9,362)
Western Bushveld				
Merensky Reef	66.2	30.6	114	56
	(2,059)	(952)	(3,546)	(1,742)
UG2 Chromitite	78.3	36.7	97	53
	(2,435)	(1,142)	(3,017)	(1,648)
Totals	203.3	116.1	939	711
	(6,323)	(3,611)	(29,206)	(22,115)

Table 2.4 Bushveld Igneous Complex Pt and Pd Reserves and Resources, after Cawthorn (1999)

2.9 Lac Des Iles, Canada

Information reviewed here was obtained from the North American Palladium Ltd website (2005). This 2.7Ma mafic/ultramafic Lac des Iles Intrusive Complex (LDI-IC) is located in western Ontario and intruded into the Wabigoon Greenstone Belt (Goodwin, 1977). The Roby Zone deposit is hosted within the complex and is a zone of disseminated sulfide in gabbro (Watkinson et al.

1979). Mineralisation at the Roby zone has been delineated over 900m by 850m and is open below 904m depth.

North American Palladium geologists have subdivided the LDI into three main intrusive ore bodies, the North LDI Intrusion (ultramafics), the Mine Block Intrusion (host to Roby ore zone) and the Camp Lake Intrusive. Mineralisation occurs as three different types: the PGE-Ni-Cu rich breccias, the mineralized dykes or sills, as at Roby Zone and in a 15 to 25m thick unit of high-grade Pd mineralisation along the east-central portion of the Roby Zone.

The resource was estimated as follows (Whyte, 2001):

Tons (Mt)	Pt (g/t)	Pd (g/t)	Au (g/t)	Cu (%)	Ni(%)	Pt+Pd+Au (Moz)
145.6	0.17	1.57	0.12	0.06	0.05	8.46

Table 2.5: Lac des Iles resource

This indicates a Pt:Pd ratio of 1:9. Current measured and indicated resource is 5.8 Moz Pd.

Chapter 3 DATA AND THE TREATMENT OF DATA

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3.1 Data needed and Means for Obtaining Data

Metallurgical, mineralogical, spatial and physical properties data was required. These were obtained from reports on test work completed, drilling reports and interpretations of these data.

Data on mining limitations and associated costs, processing costs and capital requirements for the project were obtained through interviews with mining engineers, metallurgical and mining specialist consultants and data on similar operations as owned and managed by Harmony Gold and others, where available.

3.2 Specific Treatment of Data for Each Subproblem

3.2.1 Requirements for subproblem 1 (Ore body characteristics)

a) Morphology of the ore body:

Aeromagnetic and groundmagnetic survey information was used to delineate magnetically susceptible lithological units and to plan the drilling program. Vertical sections were constructed from the borehole information, supplemented with density and magnetic susceptibility results. From these sections, 3-D ore body models were constructed and used to calculate ore body volumes and tons, internal waste volumes and tons, as well as overburden thickness.

b) Mineralogical information:

Ore body mineralogy was compared to metal content grades (g/t), as obtained from geochemical analyses of borehole samples, in order to test for correlation between specific mineralogy and grade. The confinement of metals to certain mineralogical assemblages was investigated, as this may have bearing on the processing of the ore and liberation of metals from the ore. Grades obtained from geochemical analyses of borehole samples, were used to calculate resource volumes and metal ounces contained in the ore.

c) Metallurgical information

Metallurgical characteristics of the ore-bearing rock were obtained from testing of borehole samples and a 500t bulk sample. Following initial testing of the diamond drill samples, a pilot plant was erected at Mintek to process the bulk sample and simulate commercial recovery plant processes. Results from test work were used to predict recoveries from different ores and processing problems to be expected in the commercial scale plant.

3.2.2 Requirements for subproblem 2 (Mining method)

a) Ore body model:

The ore body models were constructed using borehole information, vertical sections through the ore bodies and mapping of the bulk sample site. The boundaries of the mineralized package were delineated; and visual and other indicators of mineralisation identified.

b) Mining limitations:

Information on potential mining methods was obtained through interviews with mining engineers and mine planners and from published data on the topic. Different mining method options were considered, taking into account depth extent of mineralisation, ore zone thickness and dip.

3.2.3 Requirements for subproblem 3 (DCF model)

a) Costs of mining and processing:

Comparative mining costs were obtained from the Kalgold open pit mine, 35km southwest of Mafikeng, as the Kalgold gold deposit shows similar characteristics in terms of mineralisation depth, width and dip. Metallurgical processing costs were obtained through scoping studies and interviews with technical specialists involved in the bulk sample treatment at Mintek and also at MDM.

The mining and processing costs were applied to the specific limitations of mining at Kalplats (depth, width, weathered-fresh interface, haulage distances to plant, mill throughput, consumables, etc.) and on-site general and administrative expenses comparable to that of the Kalgold operation were applied. All costs were used for construction of the discounted cashflow model.

b) Capital requirements:

Capital requirements for construction and sustainable capital over the life of mine were obtained from scoping studies and personal interviews with construction and metallurgical consultants. Capital required was calculated using mining volumes and mill troughput volumes. Additional capital items included the establishment of a well-field and dam, infrastructure and pre-stripping of waste. All capital amounts were used in the construction of the discounted cashflow model.

c) Metals prices:

Information on metals prices was obtained from historical moving averages as published in the press and future projections made by mining analysts. As changes in metals prices have a component of volatility over time, an attempt was made to identify cycles in the metals prices. That said, the metals prices used in the discounted cashflow model are still assumptions, based on past behaviour of these prices and may vary significantly into the future. Metals prices were kept constant over the life of mine.

d) South African Rand – US Dollar exchange rate:

Historical trends of the ZAR:US$ exchange rate were analysed in terms of escalation, tempo of escalation and critical factors that may impact on the exchange rate. A constant exchange rate was used for construction of the discounted cashflow model. Fluctuations in the exchange rate will impact on the cash flows generated from the project.

e) Government tax and royalties:

Government tax on mining companies (non-gold) is 29%. The mining charter and mining act quotes royalties payable on revenues obtained from the sale of metals. These royalties will be payable as from 2009. Royalties on mining of the Kalplats deposit were negotiated and agreed upon in principle between Harmony and the DME and constitute 1% on revenue from gold and platinum sales and 0% on revenue from palladium sales. It is however still to be seen if these terms will be acceptable to the DME should the project enter the production phase.

f) Discount rate:

The discounted cashflow model is used to calculate nett present value and period of payback of capital. A discount rate of 13% was used, based on project specific risks.

<u>3.2.4 Requirements of Subproblem 4 (Alternatives to mining)</u>

a) Sale of project:

Discussions with colleagues and top management highlighted alternatives to a production decision. Benefits of immediate sale of the project were considered vs. proceeding with a full bankable feasibility study before production or sale. The decision was made to proceed with sale of the project.

<u>3.2.5 Requirements of Subproblem 5 (Benchmarking)</u>

a) Platinum-group metals (PGM) producers:

Information was obtained about various PGM operations, from published sources and this information was used to benchmark the ore body characteristics and development status of the Kalplats deposit.

Chapter 4 GEOLOGY

This chapter will give an overview of the regional, local and reef geology, discovery and exploration history of the Stella Layered Intrusion.

4.1 Regional Geology

The locality of the project area was indicated in Figure 1.1 (Chapter 1). The mineralisation reported on here is related to the Stella Layered Intrusion, which occurs within the Kraaipan Greenstone Belt. This belt comprises a succession of magnetite quartzite, magnetite-rich banded iron formations (BIFs), cherty BIFs, intercalated schists and amphibolites. Intrusive granites and diabases also occur.

The Stella Layered Intrusion was dated at 3,033 Ma (Maier et al., 2003), making it the oldest occurrence of PGE mineralisation on earth.

The terrain is structurally complex with abundant shearing, brittle faulting, thrusting, folding and disruption of stratigraphy. Small-scale structures are easily identifiable in diamond drilling core and resemble the regional scale structural characteristics (Andrews, 2002).

An eastern belt, the Goldridge Sector (internal company terminology), hosting the SACS Goldridge Formation, and a western belt, the Stella Sector (internal company terminology), have been identified from outcrop and aeromagnetic imagery, during initial stages of exploration over the Kraaipan Greenstone Belt, using government magnetic survey information and field mapping conducted by West Rand Consolidated geologists during 1992. These strike NNW-SSE for approximately 120km. The Amalia Sector (internal company terminology) is regarded to be a southern extension of the Stella Sector and has a strike of approximately 80km. It hosts the former Amalia Gold Mine, now known as Blue Dot Mining (Mathe, 1995).

The Goldridge Sector hosts the Kalahari Goldridge Mine (Kalgold), a 3 million ounce BIF-hosted gold deposit (Dabrowski, 2003). Current design production throughput is 140 000 t/month. On-lease exploration identified the depth extent of gold mineralisation to be more than 250m, as defined through drilling. The Kalgold Mine was under scrutiny during December 2003, when Afrikander Leases attempted to buy the property. This sale, however, did not realize and Harmony is continuing production on the property. (Harmony Gold Annual Report, 2005)

Outcrops are limited and magnetic units were delineated using aeromagnetic and groundmagnetic surveys, thereby enabling the company to define the extent of the greenstone belts. Figure 4.1 illustrates the extents of the Goldridge and Stella Sectors and figure 4.2 the Kromdraai Formation Map (AAP, 1992).

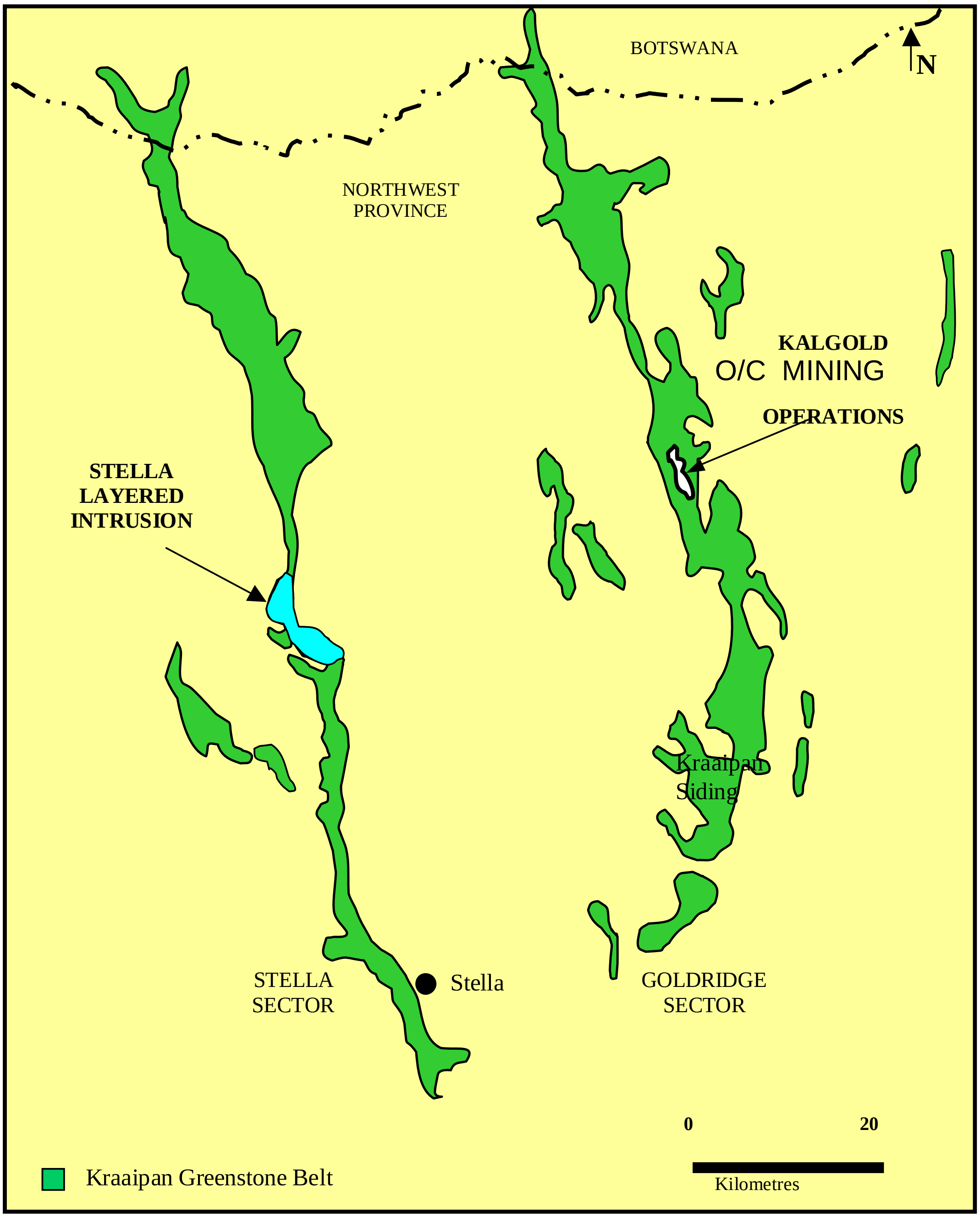


Figure 4.1: Locality of the Stella Layered Intrusion

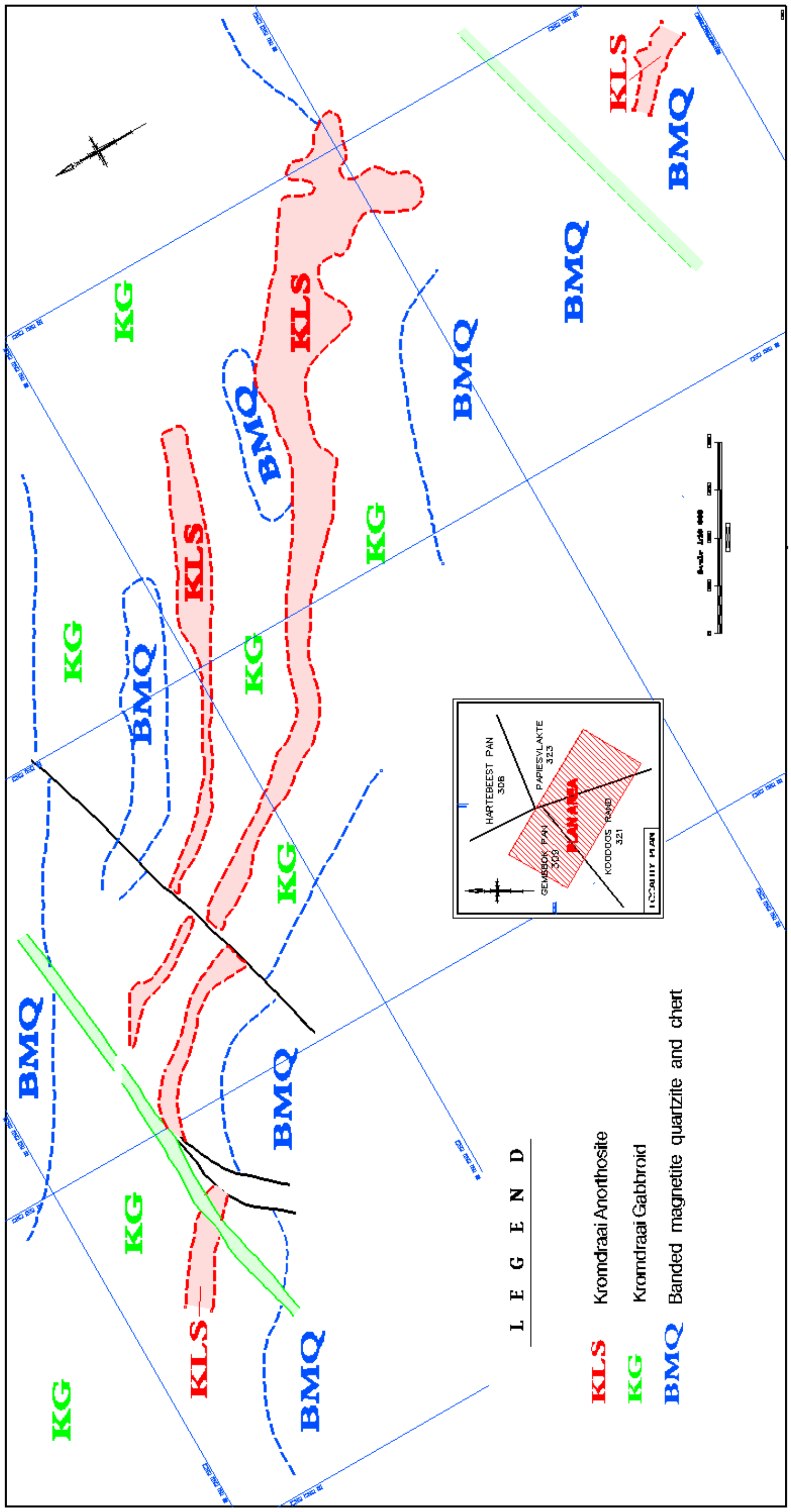


Figure 4.2 KROMDRAAI PGM PROJECT
GEMSBOK PAN 309 / KOODOOS RAND 321 / PAPIESVLAKTE A323
GEOLOGY

4.2 Mineralisation in the Stella Layered Intrusion

Seven (7) ore bodies were defined in the Stella Layered Intrusion, namely Crater, Orion, Sirius, Vela, Crux, Serpens North and Serpens South (Figure 4.3).

Outcrops are scarce in the area, due to extensive sand cover (Kalahari sand). The horizontal dimensions of the intrusions are estimated from magnetic bedrock signatures and borehole information at approximately 14 by 1.5 km.

The strike of the SLI is broadly north-west, similar to that of the Kraaipan Greenstones. It is postulated that the SLI originally developed as a sill-like intrusion, sub-parallel to shallow dipping strata of the Kraaipan. The SLI was intruded by mafic sills prior to deformation. These sills intruded the Kraaipan succession too and are now represented by highly deformed diabase dykes (Andrews, 2002).

4.3 Discovery History and Regional Exploration

Anglo American Prospecting Services conducted large-scale regional exploration programmes over the area during the early 1990's. These included soil geochemical surveys and drilling over the farm Kromdraai. Samples were routinely analysed for Au, Pt and Pd and low levels of PGE mineralisation were reported over the southern Kalplats deposit area. Analyses results were reported in internal company communications.

West Rand Consolidated Exploration Pty (Ltd) acquired the prospecting licenses over the Kraaipan Greenstone Belt in 1994 and started with regional exploration for gold mineralisation. These programmes included regional mapping, soil geochemical surveys along fence-lines and roads, detailed aeromagnetic surveys and investigations in close proximity to old gold workings.

Soil samples were analysed for Au only, by Scientific Services in Cape Town, using a cyanide leach method and atomic absorption (AA) finish. Those samples that delivered encouraging assay results (>10ppb Au) identified areas for further soil sampling to confirm results. Rock samples from outcrops in the vicinity of the anomalies were also analysed and some elevated Au values were reported (~10g/t). The focus at this stage was to discover gold deposits (Gartz, 2002).

A new assaying laboratory was elected to handle large volumes of samples, Genalysis Laboratory Services (Perth, WA), who then analysed all samples for Au (1 ppb detection limit), Cu (1 ppm) and As (5 ppm). An aqua regia digest method was used with a flame absorption AAS (atomic absorption spectrometry) finish for Cu and As. A graphite furnace AAS finish was used for Au (Gartz, 2002).

WRCE was incorporated into Harmony Gold Mining Company in 1999. With the PGE market booming at that stage, it was decided to re-investigate the Kromdraai area. The farm, Morester, was also targeted, based on aeromagnetic data displaying similarities to the Kromdraai area. These included the presence of highly magnetic lithologies coincident with large-scale structures. Landsat TM data and aerial photographs were used for regional mapping of outcrops and soils (Carroll, 1999).

Local grids were laid out over the identified magnetic anomalies. Groundmagnetic surveys were done on 100m line spacing and soil sampling was also done along these lines at 25m intervals, with samples composited every 50m (thus 2 samples combined to deliver one final sample for a 50m interval). Geochemical soil sample analyses were done on the -150µm fraction, after it was established through orientation work that this was the most representative mineralised soil fraction (Gartz, 2002).

6,800 Soil samples were analysed for Au (1 ppb detection limit), Pt (5 ppb d.l), Pd (10 ppb d.l), Cu (1 ppm d.l.) and As (0.5 ppm d.l.). The analytical method used consisted of an aqua regia partial digest, with analyses for Au, Pt, Pd and As by ICP-MS (inductively-coupled plasma mass spectrometry) and Cu by AAS. Genalysis, also did these analyses. The soil geochemical anomalies were in the order of tens of ppb (parts per billion) Pt/Pd/Au, while background values of 2-4 ppb were common. Dispersion haloes of PGM mineralisation were smaller (~100m around bedrock mineralisation) than those for As and Cu (200m around mineralisation) (Gartz, 2002).

Overburden drilling was done on a wider line spacing and targeted on top of certain magnetic anomalies where encouraging soil sampling results had been obtained. A study of PGE mineralisation in the Rincon del Tigre Complex of eastern Bolivia (Prendergast et al., 1998), indicated that soil geochemistry closely reflects the distribution of metals in underlying weathered and fresh bedrock. The physical environment of the Rincon Complex is sub-tropical and therefore different to the environment of Kalplats. As no tropical weathered soils are present at Kalplats, overburden drilling was done with an open hole (percussion) drill rig and targeted to drill through the overburden (mostly Kalahari sand) into +/- 2 meters of bedrock. Samples were taken at the overburden-bedrock interface, which is the boundary between the Kalahari sand and saprolitic (highly weathered) bedrock. The last two meters in every borehole were also composited and analysed. The results for samples in every hole were compared and showed good correlation between the different elements analysed for (Gartz, 2002).

Overburden samples were analysed in the same way as above mentioned soil samples. Once again, elevated results were followed up by overburden drilling in the vicinity of anomalies and also by closer spaced ground magnetic surveys (50m line spacing). A total of 2,400 samples were analysed, comprising stoneline and bedrock samples. Some drilling samples were analysed for Ni and resultant values were up to 100 ppb Ni. No more Ni analyses were done. Anomalies as reported by the overburden drilling samples were in the range of 10's to 100's ppb Pt/Pd, with background values of <10 ppb (Genalysis laboratory analyses results, Kalplats overburden sampling batches).

Bedrock types were identified as belonging to an intrusive into the greenstone belt sequence and consisted mainly of cumulate gabbro, magnetite gabbro and magnetitite (>90% magnetite) layers. Mineralogical work done by Steffan (2001) and petrographic studies by Duarte et al. (2001) confirmed this assemblage. The intrusion was termed the Stella Layered Intrusion. Further observations revealed the very fine-grained nature of the PGE's, varying from 3μm to approximately 17μm (Duarte et al., 2001) The distribution of PGE's in different minerals is discussed further under the Metallurgy Section (Chapter 7).

4.4 Target selection

Previous work by Anglo American Prospecting Services on the Kromdraai prospect, identified a number of PGE anomalies, later attributed by Harmony to different ore bodies along strike of the aeromagnetic anomalies. These PGE anomalies indicated the existence of mineralisation on the more recently termed Orion, Serpens North, Serpens South and Crux ore bodies (Gartz, et al. 2001)

Reverse circulation boreholes were drilled on the same section lines used for soil sampling and overburden drilling. Drilling was targeted and angled to intersect the strata which underlies soil sampling and overburden drilling anomalies. It was determined that the average dip of the layered intrusion is steep about the vertical (85 – 90 degrees) and this influenced the decision to drill boreholes at inclinations of 50-60° to the east or less frequently to the west.

A number of diamond boreholes were drilled to confirm depth extent and reliability of RC sample values obtained. Drilling done by Harmony in 2000-2001 delineated the northern Crater, Vela and Sirius ore bodies. Figure 4.3 illustrates the distribution of the ore bodies.

A considerable gap exists between the Sirius and Orion ore bodies. Airborne and ground magnetic surveys failed to identify any magnetite-rich successions in this gap and no further work was conducted over this area.

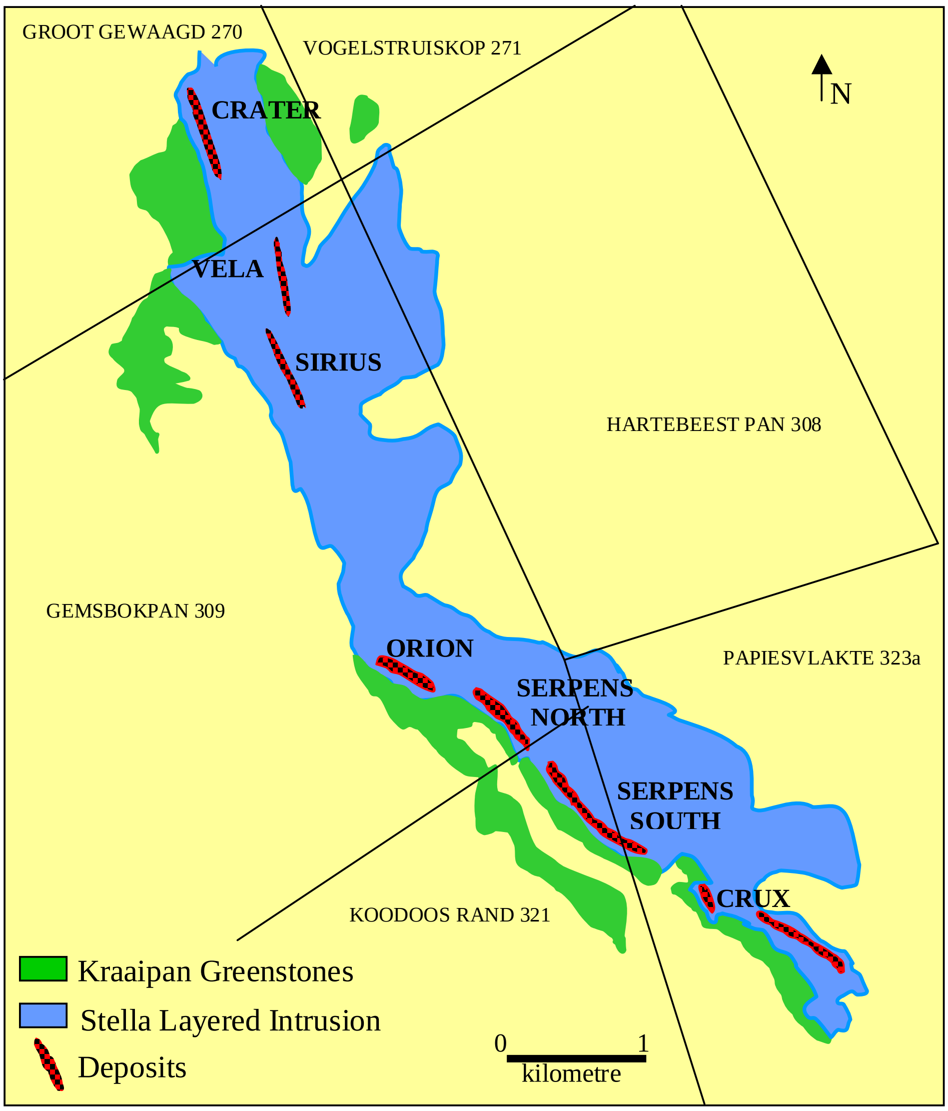


Figure 4.3: Ore bodies of the Stella Layered Intrusion: (Gartz, et al., 2001): The delineation of the SLI and Kraaipan Greenstones was done using aeromagnetic survey data, supported by drilling information, as outcrop is scarce in the project area.

Diamond drilling samples were analysed by Lakefield Research Laboratories, now SGS. The samples were crushed to –2mm and a 500g sub-sample was then milled to 80% -75µm. Analyses consisted of Au, Pt and Pd, using fire assay (Pb collection) and an ICP-OES finish. Representative pulp samples (1,300 samples) were sent for re-assaying to Genalysis Laboratories, which indicated a constant 1% over-reporting of values by Lakefields. The Pb-collection method tends to result in an underreporting of especially Pd (Theart, 2003) and the inconsistency of assay results from the different laboratories may be indicative of such a problem.

No adjustments were made to the results from Lakefields, but consideration should be given to repeating a select number of assays using the Ni sulphide collection method in order to test for underreporting of Pd when using the Pb-collection method. A limited number of samples were analysed for Rh, but resultant values were too low to justify further analyses for Rh.

All anomalous results were followed up by more RC drilling along strike and also laterally displaced, relative to previous boreholes to confirm the anomalies. The best reef continuity was identified in the Crater, Orion and Crux ore bodies. Figure 4.7 shows a schematic view of a typical ore body. Figure 4.8 is a typical cross-section through an ore body (Harmony internal progress reports, 2003).

As infill drilling progressed, data became available for drilling on 100m spaced lines. Interpretations along strike and on cross-sections were done and these were confirmed by more diamond drilling and more RC drilling. Two ore bodies, Crater and Orion, were selected for detailed investigation, based on drill results to date. Detailed drilling of these two deposits commenced (50m line spacing), while exploration drilling continued across other targets and prospects (Harmony internal progress reports, 2003)

Samples from diamond drill holes were sent for metallurgical test work to MINTEK and to Lakefields Laboratory for analyses. These samples were quartered diamond cores at 1m intervals. Reports are available detailing results of this test work. The metallurgical test work proved positive and a full-scale detail-drilling programme was launched, consisting of RC and diamond drilling (Harmony internal progress reports, 2003).

Following more metallurgical test work and analytical results obtained, the decision was made to extract a bulk sample from the Crater ore body. This sample was excavated during December 2002, between 40m and 45m depth over what was interpreted, through shallow drilling, to be the least structurally complex section of the ore body, which would provide the most geological information. Figure 4.4 shows the location of the excavation (Van Niekerk, 2003).

During excavations, pit wall mapping was done as well as mapping the floor of the pit, as depths increased. This mapping provided useful information with regards to the distribution and orientation of different structural elements in the box cut. It is believed that the distribution of structures is representative of the deposit as a whole (Harmony internal meeting, 2003).

A 500 ton bulk sample was taken for test work at MINTEK, after 1.5 million tons of rock was moved. Drilling and blasting was required below depths of 8 – 10m. A pilot plant was built and different permutations of processes were tested. Results from this test work were positive and two potential metallurgical flowsheets were suggested for full-scale plant design (Duarte et al., 2003).

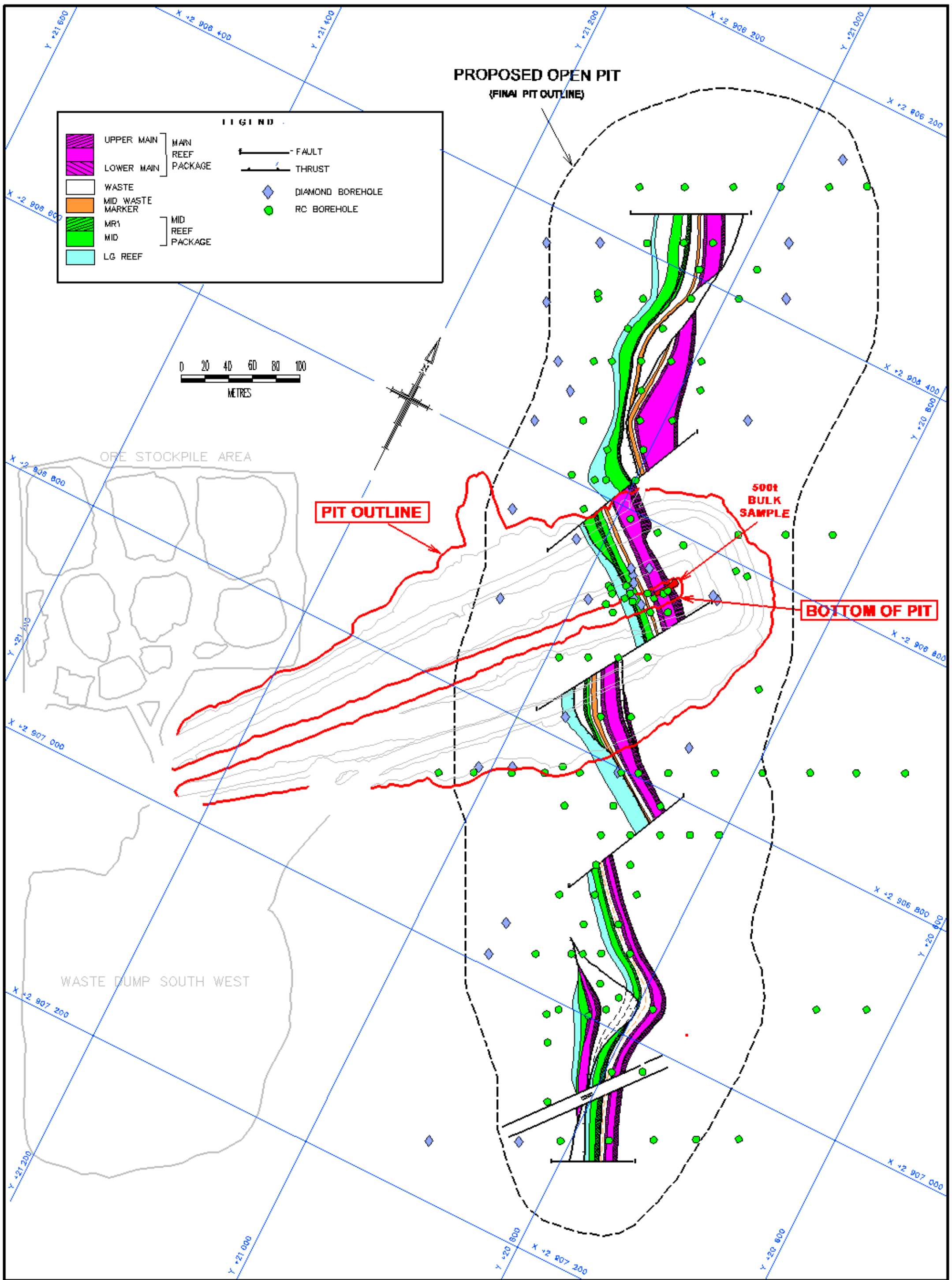


Figure 4.4 **CRATER LEVEL PLAN : 60m below surface**

Crater is located on the farm Groot Gewaagd 270

Subsequent to this test work, more drilling was done on Crater, Orion and the five other ore bodies. As more information became available, the structural interpretation became more complex, with high frequency small-scale faults being identified. Three comparative diagrams, later in this chapter, illustrate the evolution of the understanding of Crater's structural framework, Figures 4.7 – 4.9.

Further mention will be made under the Metallurgy section in this report.

4.5 Geology of the Stella Layered Intrusion

The Stella Layered Intrusion consists of a sequence of gabbro, magnetite gabbro and magnetitite. The layered sequence intruded into the Kraaipan Greenstone Belt. The SLI was intruded by mafic sills, now present as diabase dykes. Subsequent faulting and fracturing enabled the intrusion of fluids to form quartz-albite veins, which vary in width from a few centimeters to approximately a metre.

Figure 4.5 is a schematic illustration of reef succession in the Crater Ore body, further discussed in Section 4.6, while figure 4.6 illustrates the average Pt, Pd and Au grades in each reef.

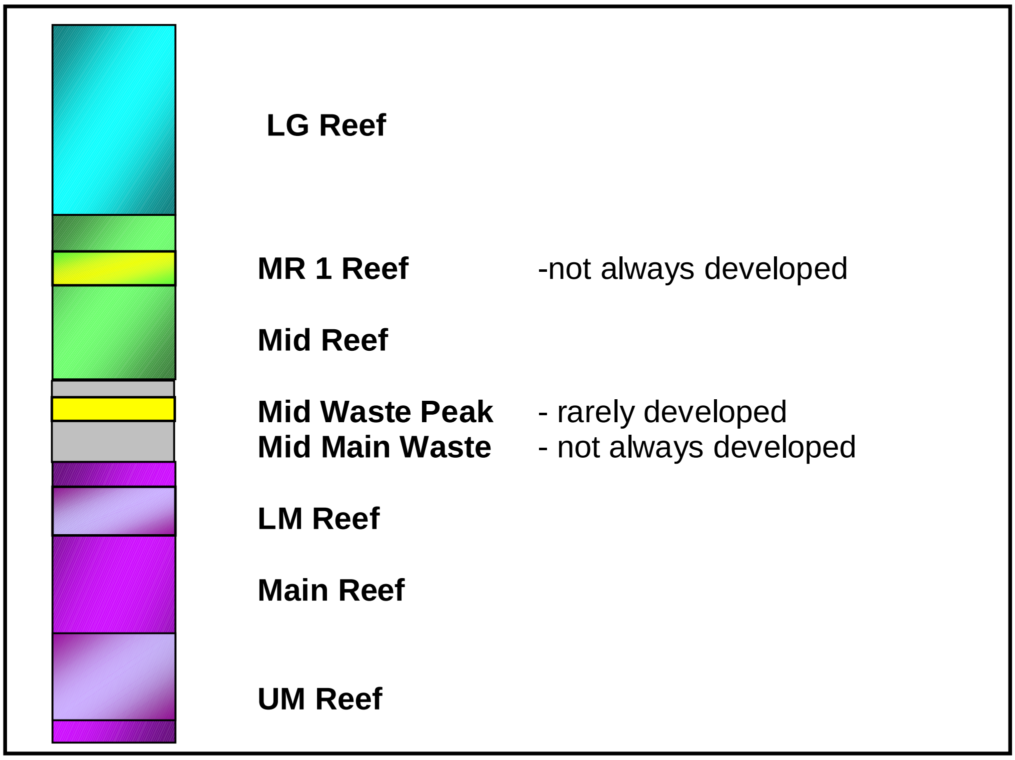


Figure 4.5: Type section indicating different magnetite layers

	Pt	Pd	Au	Pt+Pd+Au	THICKNESS	AVERAGE Pt+Pd+Au
LG REEF	0.6g/t	0.4g/t		1.0 g/t	7.0 m	1.00g/t
MR1 REEF	0.5g/t	2.0g/t		2.5 g/t	2.0 m	
MID REEF	0.3g/t	0.8g/t		1.1 g/t	8.0 m	1.38g/t
LM REEF	1.8g/t	2.0g/t		3.8 g/t	2.0 m	
MAIN REEF	0.5g/t	0.5g/t	-	1.3 g/t	8.0 m	2.32g/t
UM REEF	2.4g/t	1.9g/t	0.2g/t	4.6 g/t	3.0 m	

Figure 4.6: Schematic indicating average PGE grade (g/t) distribution in the PGM layers

4.6 Structure of the Stella Layered Intrusion

The early deformational history of the SLI and the Kraaipan greenstones was dominated by an eastward-vergent compressional event. Progressive deformation during this event first resulted in folds being overturned and resultant steepening of strata. The PGE mineralised layers identified in the SLI are interpreted to occur on the overturned western limb of such a fold. During the latter stages of deformation, brittle-ductile, thrust-sense shear zones developed along overturned limbs of the folds.

The shear zones exhibit a *ramp and flat* thrust geometry and locally cause imbrication of strata (Hilliard, 2001). Deformation during this orogenic event was accompanied by greenschist to amphibolite facies metamorphism. Alteration occurred in the form of carbonatisation, chloritisation, silicification and sericitisation. Assemblages are dominated by epidote, calcite and chlorite. Focussed fluid flow along thrust-sense shear zones resulted in localised carbonate alteration. Thrusting was accompanied by the intrusion of granitoid sheets and associated metasomatic quartz-albite veins.

Thrust imbrication of the PGM mineralised layers has resulted in duplication of reef-packages in some zones and significant disruption and reef-loss in others. Felsic granitoids have locally intruded, thereby diluting mineralisation in the reef zones. Sub-vertical brittle faults, perpendicular and oblique to the stratigraphy occur. They may be characterized by large movements that result in displaced reef packages for either hundreds of metres or smaller displacements that only locally disrupt reef continuity (Hilliard, 2001).

Figures 4.7-4.9 illustrate plan views of the Crater ore body, indicating displacements along faults. These diagrams further indicate the evolution of understanding of the ore body over time, as more information was obtained through drilling.

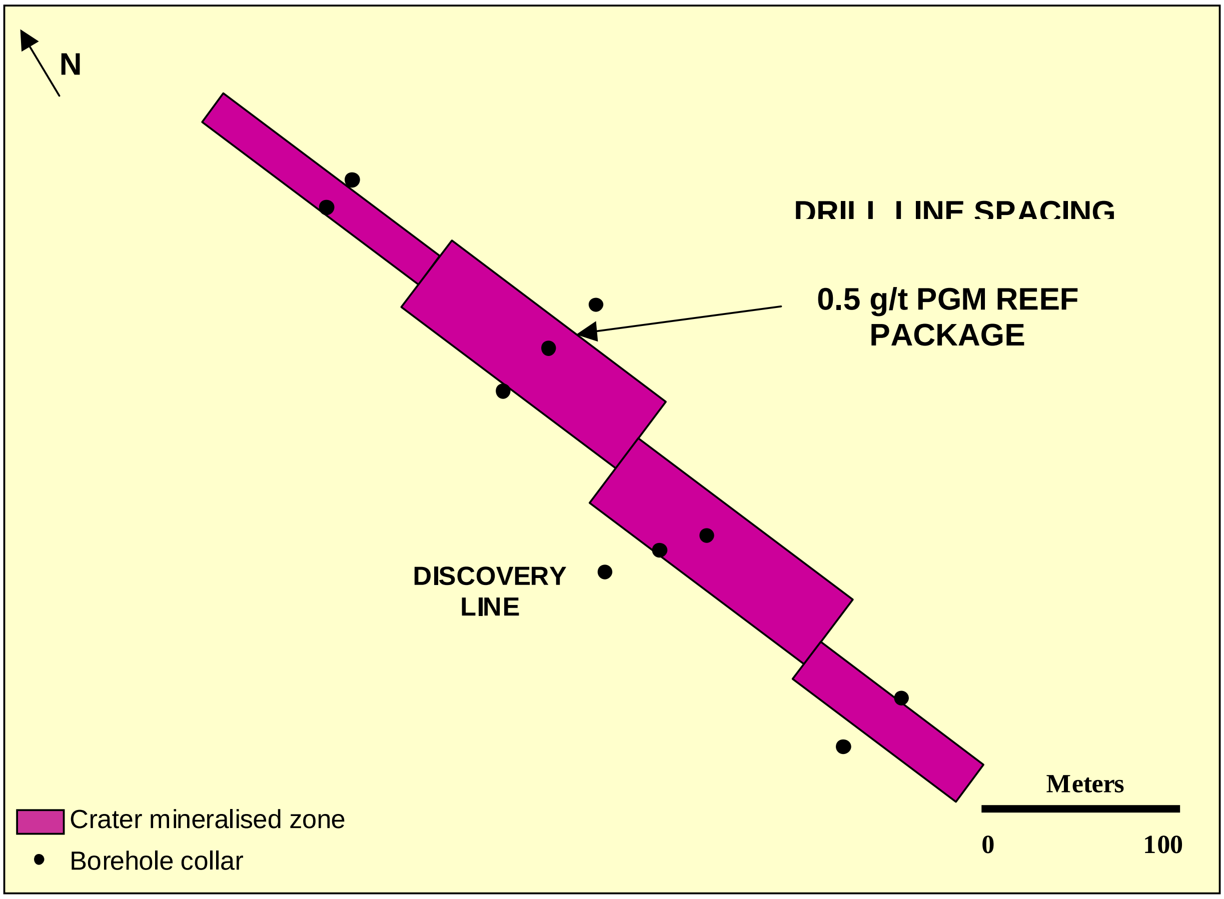


Figure 4.7 Schematic Delineation of the Mineralised Zone in the Crater Ore body:

This schematic was based on information available at October 2001. A limited number of boreholes had been drilled to date and the purpose of delineation was to assist in the targeting of future drilling and sampling surveys.

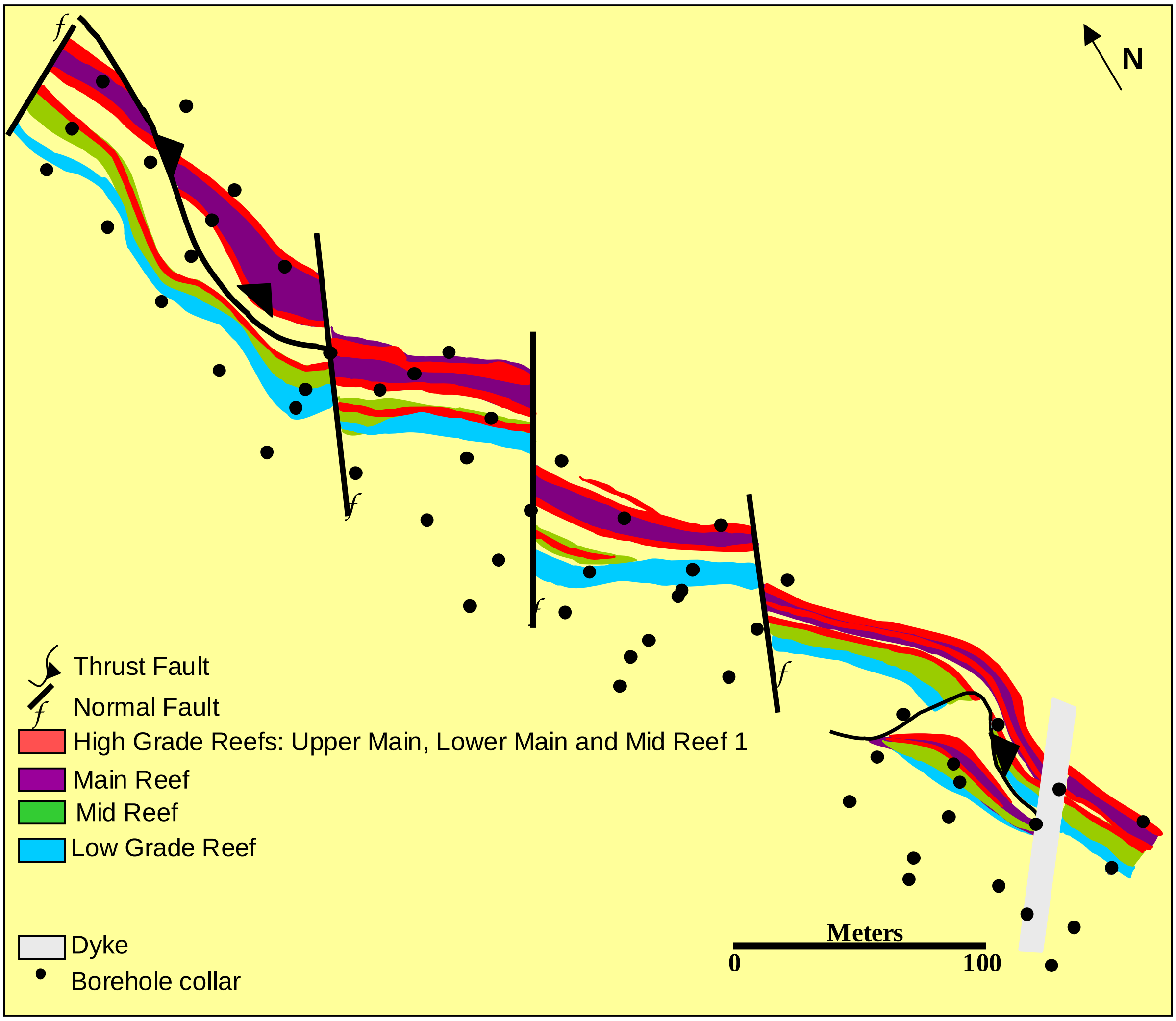


Figure 4.8: Plan of the Crater Ore body based on information at July 2002:

The ore body model evolved as more drilling was undertaken and the model dated July 2002 was based on results from 56 boreholes. Different reefs had been identified, based on grades of and ratios between platinum, palladium and gold mineralisation. Zones of internal waste were also identified and the structural model evolved somewhat, to incorporate thrust faulting

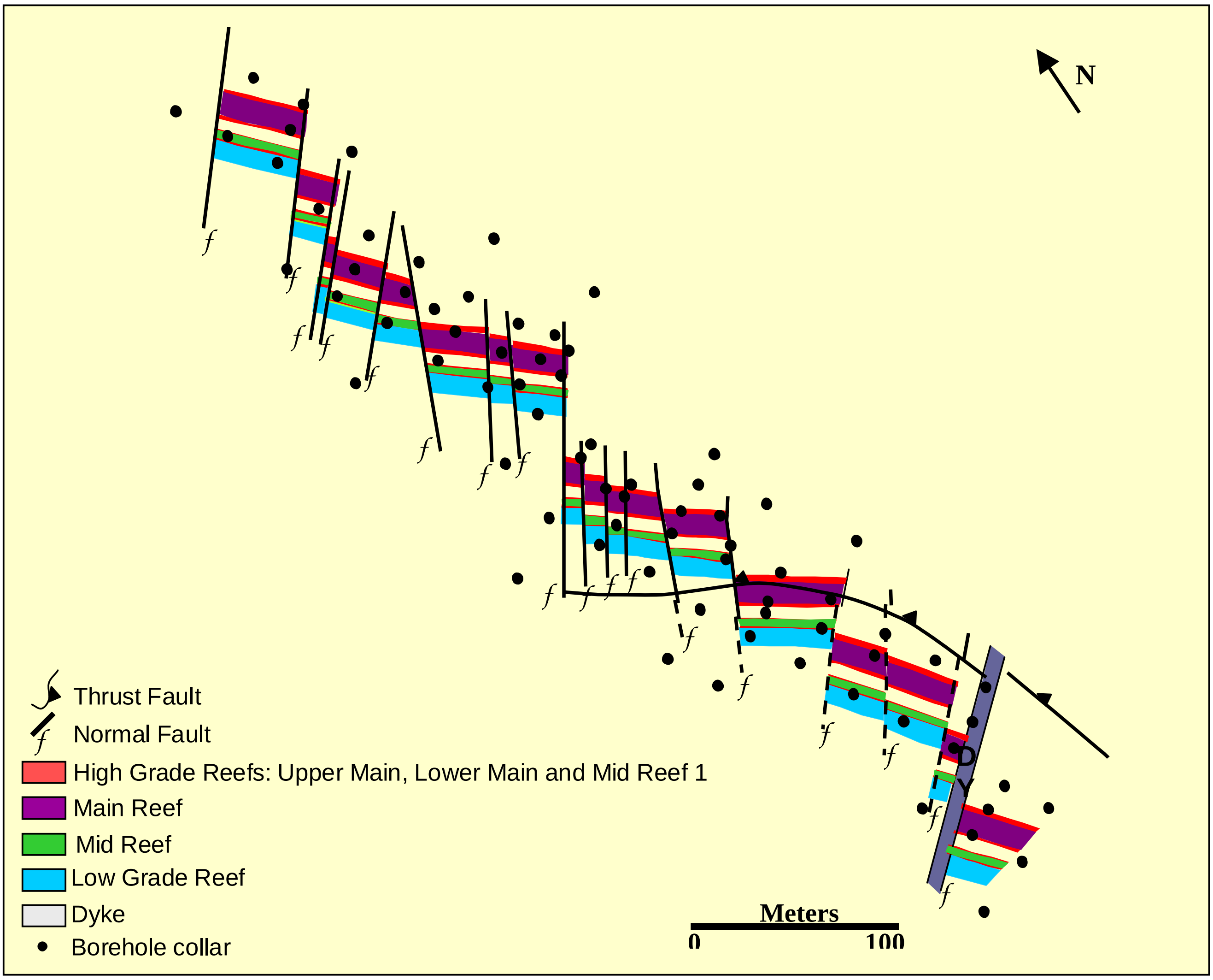


Figure 4.9: Plan of Crater Ore body based on information at December 2003:
As more information became available, the ore body and structural model evolved accordingly. As at December, 2003, a total of 78 boreholes had been drilled and information incorporated in the models.

4.7 Reef Geology

It is possible to discriminate between the different reefs in the ore-bearing package, based on different metal contents, inter-element ratios, as well as on different cumulate textures.

The succession identified, occurring generally from east to west (NE to SW) is the following: Main Reef, including the Upper Main (UM) and Lower Main (LM) Reefs, Mid Reef, including the MR1 high grade Reef and the Lower Grade (LG) Reef. Internal waste, consisting of barren gabbro, occasionally separates the Main Reef package and the Mid Reef package. This zone is called the Mid Main Waste. In addition to this, cross cutting diabases and granitic sheets also contribute to waste. An additional thin reef, the Mid-waste Peak (after elevated gold values, relative to host material), has been locally identified in the internal waste, mainly on the Crater deposit. It is important to note that not all reefs are developed continually or present in all intersections.

The LM Reef is a high-grade section of the Main Reef on the western extremity of the Main Reef, while the UM Reef is a high-grade section on the eastern extremity of the Main Reef. The LM reef is generally approximately 2m wide, with a Pt:Pd ratio of 1:1.2 and an average grade of 3-4g/t Pt+Pd+Au. The Main Reef has a precious metals content of 0.6 – 1.5 g/t Pt+Pd+Au and Pt: Pd of 1:1, while thickness ranges between 3 – 10m. The UM is a 4m wide reef with an average grade of 4.3 g/t Pt+Pd+Au and Pt:Pd of 1:1. This high-grade reef may display grades of higher than 10g/t Pt+Pd+Au over a 1m width, which makes it easily recognizable in the grade histograms.

Coarse chalcopyrite appears for the first time in the eastern half of the UM and gold is associated with this appearance. The UM displays transition to a well-developed cumulate texture and contains euhedral feldspar grains of up to 10mm long. Visible distinction between the Main and UM Reefs is impossible and a lower cut-off of 2g/t Pt+Pd+Au is used to identify the UM.

The Mid-Main Waste (MMW) zone is typically 8 – 10m wide and contains a 1-3m wide magnetite-rich gabbro unit with average grades of 0.5 – 1.5 g/t Pt+Pd+Au. The rest of the MMW zone contains in the order of 0.1 – 0.4 g/t Pt+Pd+Au.

The MR1 reef is a magnetitite, typically 2-4 m wide, with a grade of 2-3 g/t Pt+Pd+Au. The total Mid Reef contains a grade of 1.3 g/t Pt+Pd+Au over a width of 10-15m and has a Pt:Pd ratio of 1:3.

The contact between the LG and Mid Reefs can only be identified through metals ratio changes. The LG Reef is generally approximately 10-15m wide and contains an average grade of 1 g/t Pt+Pd+Au, with Pt:Pd of 1.3:1.

Figure 4.10 schematically illustrates the characteristics of mineralisation in the reefs.

Minor remobilization/redistribution of the PGM's have occurred in the top 20-30m of weathered rock, with only small losses of Pd (and Au). The silicates have, however, undergone extensive changes during weathering, influencing PGE recoveries negatively. Recoveries from the weathered zone are between 10-50%.

Simplified ore body models were created in Datamine and derived from drilling data. The model was updated as new information became available and used for mine planning purposes. Steep dips on brittle faults and dykes were derived from diamond and RC drilling intersections. The ore body models are treated in Chapter 6 - Mining.

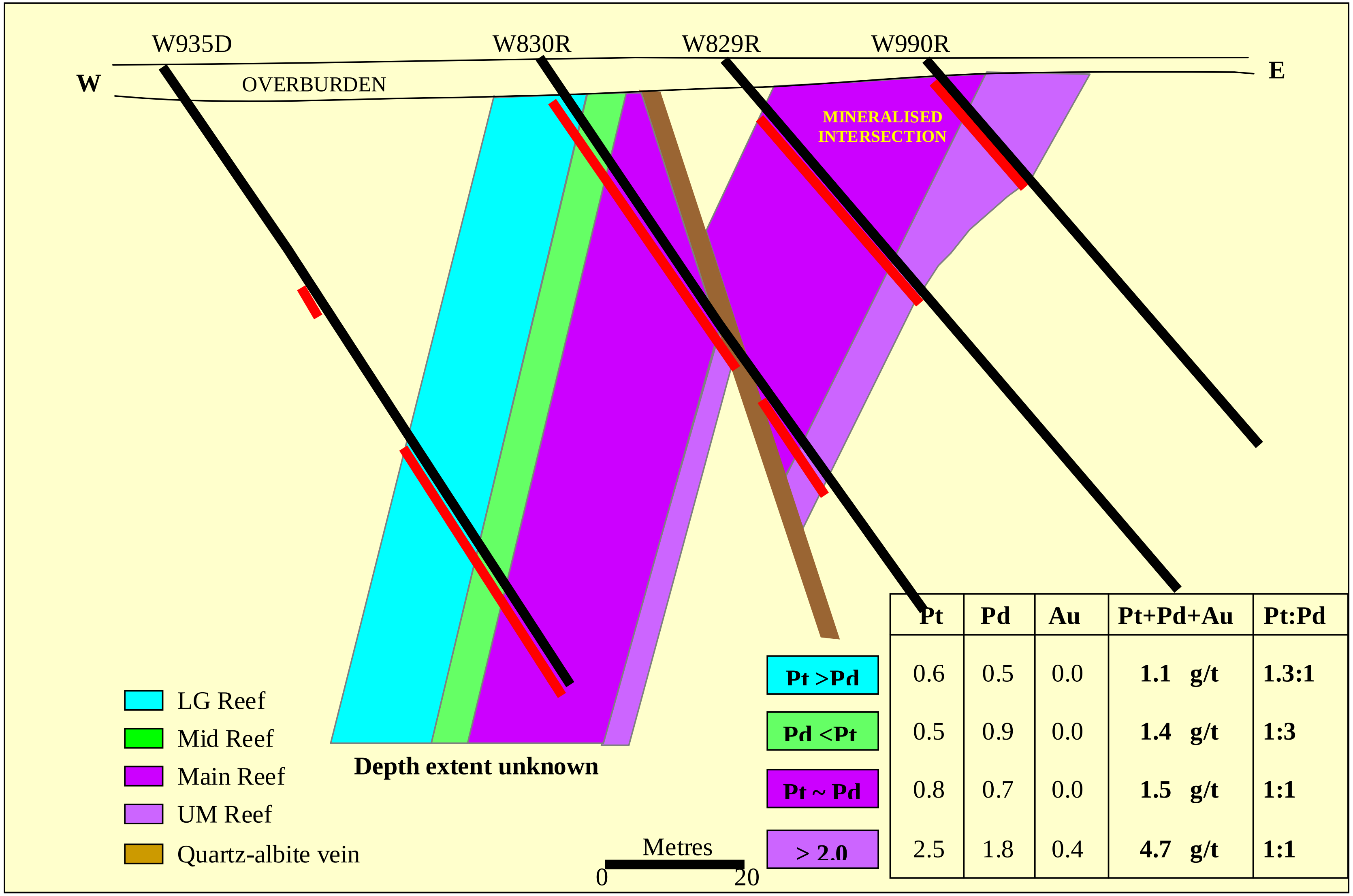


Figure 4.10: Crater Schematical Vertical Section

4.8 Applied Mineralogy

Mineralogical studies were done by Duarte et al. (2001) and also by Stefan (2001). These mineralogical studies have shown that Platinum Group Minerals occur as very fine grains, varying between 3 - 17μm and occurring on silicate, oxide and base metal sulfide (BMS) grain edges as well as being locked into these assemblages. Some grains also occur on grain boundaries of oxide and silicates or BMS and silicates. Most common occurrences (>65%) are on silicate grain edges and locked in silicates, which will cause difficult liberation of metal grains during grinding and milling.

Predominant PGM types are sulfur-deficient and include Pt-As, Pd-Sb(As), Pt-Bi, Pt- and Pd-Te, Pt-Sb, Pt- and Pd-sulfides, Pt, Pd, Au and Pt- and Pd-sulfarsenides. Volume distribution of the different phases is given in Table 4.1. The major mineral hosts are amphibole, epidote, biotite, chlorite and quartz.

XRD (X-ray diffraction) techniques were used to determine the gangue mineral phases. They consisted primarily of magnetite (some Ti-bearing), quartz, feldspar, chlorite and amphibole, with minor calcite. Approximately 25% by volume of the PGM's are locked in and 15 % attached to gangue particles. Average PGM diameter was determined to be 4μm. Approximately 75% per volume is smaller than 10μm.

The volume distribution of PGM's would suggest that the mineral Sperrylite ($PtAs_2$), constitutes the majority of the mineral grains. Samples used for mineralogical investigations were milled in a laboratory mill to around 80% -75μm. Liberated PGM grains constitute a major proportion (~60%) of the PGM-containing particles.

PGM Types	**Volume %**
Pt-As	68
Pd-Sb(As)	11
Pt-Bi	6
Pt and/or Pd-Te	6
Pt-Sb	5
Au	<1
Pt-Fe	1
Pt/Pd Sulfide	1
RhIr-sulfarsenide	1

Table 4.1: The PGM types and their Volume Distribution

4.9 Benchmarking

The characteristics of the Kalpats deposit are benchmarked in Table 4.2 against other known deposit characteristics, as previously described in Chapter 2.

It is evident that no deposit rivals the Bushveld's mineralized layers in size. The average grade of the Kalplats deposit is substantially lower than the grades of the other PGM deposits.

Pt:Pd ratios vary significantly across the deposits and the most favourable situation, given current market conditions is a larger Pt:Pd ratio. The economic viability does depend on the grade and size of the reserve though and generalizations are not to be made.

Interesting to note is the fact that the Kalplats and Stillwater deposits have similar ages, while the other deposits are in the order of 700 million years younger. Kalplats and Stillwater do not however display similar characteristics in terms of grade and Pt:Pd ratios.

Deposit	**Kalplats**	**Arctic Platinum**	**Stillwater**	**Uitkomst**	**Bushveld IC**
Resource (Moz)	3.9	12.2	NA	4.6	1,650
Reserve (Moz)	-	-	23.6	2.1	320
Status	feasibility	feasibility	production	production	production
Average Grade (g/t)	1.4	2.3	18	?	Merensky: 4.6 Platreef: 2.7 UG2: 4.4
Pt:Pd	1:1	1:4	4:1	1:2	Merensky:2.3:1 Platreef: 1:1 UG2: 1.2:1
Other elements	Au, Cu, Ti, V	Au, Rh, Cu, Ni	Fe, Cu, Ni, Au, Ag	Ni, Cu, Co	Rh, Ru, Ir,Os, Cr
Pt Production (koz/anum)	0	0	210	??	5,270
Age (Ga)	2.75	NA	2.7	2.0	2.05

Table 4.2: Benchmarking of PGE producers: This comparison is based on publically available information.

All the PGE deposits are associated with other minerals, including base metals, gold and silver.

Chapter 5 MINERAL RESOURCE

This chapter will describe the geostatistical distibution of mineralisation in the Kalplats PGM deposit and define a mineral resource for the deposit.

5.1 Resource definition

The Kalplats global *in situ* resource estimations are based on interpretations of reef intersections as obtained through drilling. The spacing of the drill lines were 25m at Crater and Orion and 50 – 200m on Sirius, Vela, Serpens North, Serpens South and Crux. For the purpose of this study all resources are classified as inferred. Section 5.4 deals with geostatistical definition of grades and confidence intervals in the different reefs. The deepest intersections on the Main Reef differs for the different ore bodies, with deepest vertical intersection (220m) on Crux. Deepest Main Reef intersections on Crater and Orion were 165m and 190m respectively.

Geological losses have not been accounted for in these estimates. Due to the narrow and irregular distribution of intrusive dykes, it will be impossible to exclude them from mining volumes. Provision will be made for the exclusion of these volumes as ore-bearing, in the financial model. This provision will imply the application of a 10% dilution factor to average grades.

'Geozones' were identified over the Crater and Orion ore bodies. These were defined based on density of drill hole spacing firstly, and bounded by the presence of major structures. Four geozones were identified on Crater, with the box cut area falling into geozone 2 (dense drill hole spacing) and zones 3 and 4 bounded by major shears and thrusts respectively. Three geozones were identified on Orion. The Crater geozones are illustrated in Figure 5.1.

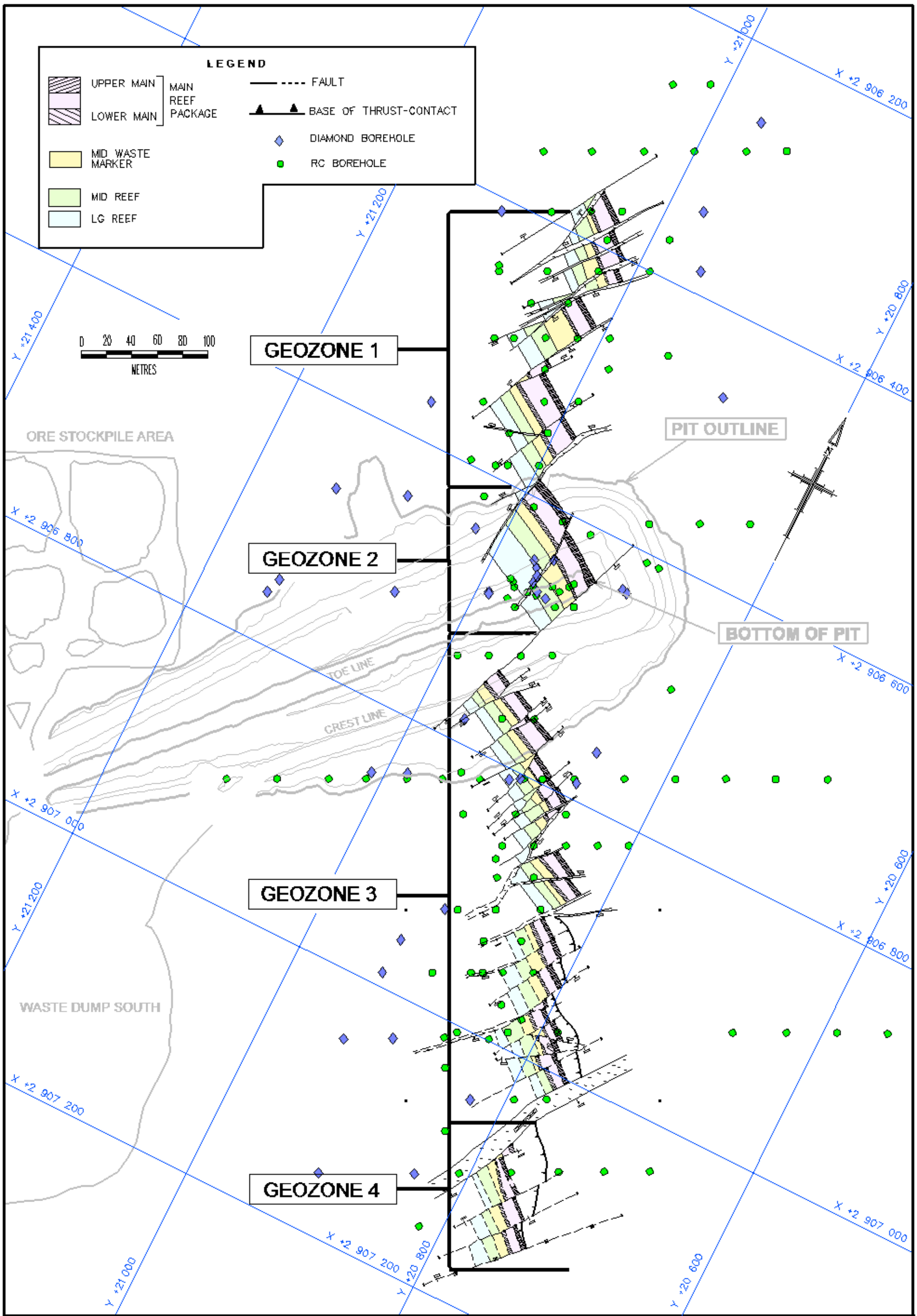


Figure 5.1 **CRATER GEOZONES**
As used for 3D Moddeling

Resource estimations for Crater and Orion were done through computations, using the 3-D ore body models as created in Datamine. Estimations on the other five ore bodies were done manually, using the transverse drill section, polygonal method. The polygonal method was also used for determining Crater and Orion resources, as a measure of control on confidence in contained resources of other ore bodies.

5.2 3-D Modeling Estimation (Crater and Orion)

In order to develop a geological and structural ore body model, the following steps were followed:

1. Each borehole section was interpreted manually and sections were linked up to create a geological and structural plan. A representative section of the Orion Ore body is illustrated in Figure 5.2.
2. Different level plans were also created, using the method of vertical projection of borehole data onto that level below surface. The 'surface' projection was made to the base of the overburden.
3. The sections and level plans were digitized in Microstation and exported to Datamine format.
4. Datamine processing comprised the creation of firstly a structural model, indicating faults and dykes, and thereafter linking of reefs at approximately 20m depth intervals.
5. The strings created for different reefs were wireframed in order to produce a 3-D wireframe model. Individual reef bands were extrapolated to depths of 500m in the case of Crater and to 270m for Orion, using a constant dip of 85° W or SW for projection to depth. This was done for each geozone identified over these two ore bodies. Certain geozones displayed folding and where present the wireframe models were not extended to depth. The depth extent of mineralisation below these sections needs to be tested by further drilling.
6. Volumes and tonnages were calculated for each reef band, with consideration for the weathered/fresh interface and the respective rock densities.
7. Assay data for all boreholes was recalculated to 1m intervals, as input to statistical analyses requires a uniform sample size. This method only impacted on a limited number of boreholes for which 2m intervals were sampled.
8. The cell size defined for statistical purposes was 4m x 4m x 10m for Crater and 8m x 8m x 10m for Orion. Unfortunately, as a result of serious time constraints, these blocks were not rotated parallel to the strike of the reefs, giving rise to blocks with boundaries outside of the main reef zones. The block sizes were also relatively large in relation to the narrow MR1 (2m) and LM (2m) reefs. This could result in reducing the average grades of these

reefs in the model. Discussion of grades and confidence intervals of these follows in Section 5.4.

9. Due to the structural complexity and sometimes wide borehole spacing, it was inferred that Kriging would not be the optimal geostatistical method to use for grade estimation. The geostatistical procedures are described in more detail under section 5.4.

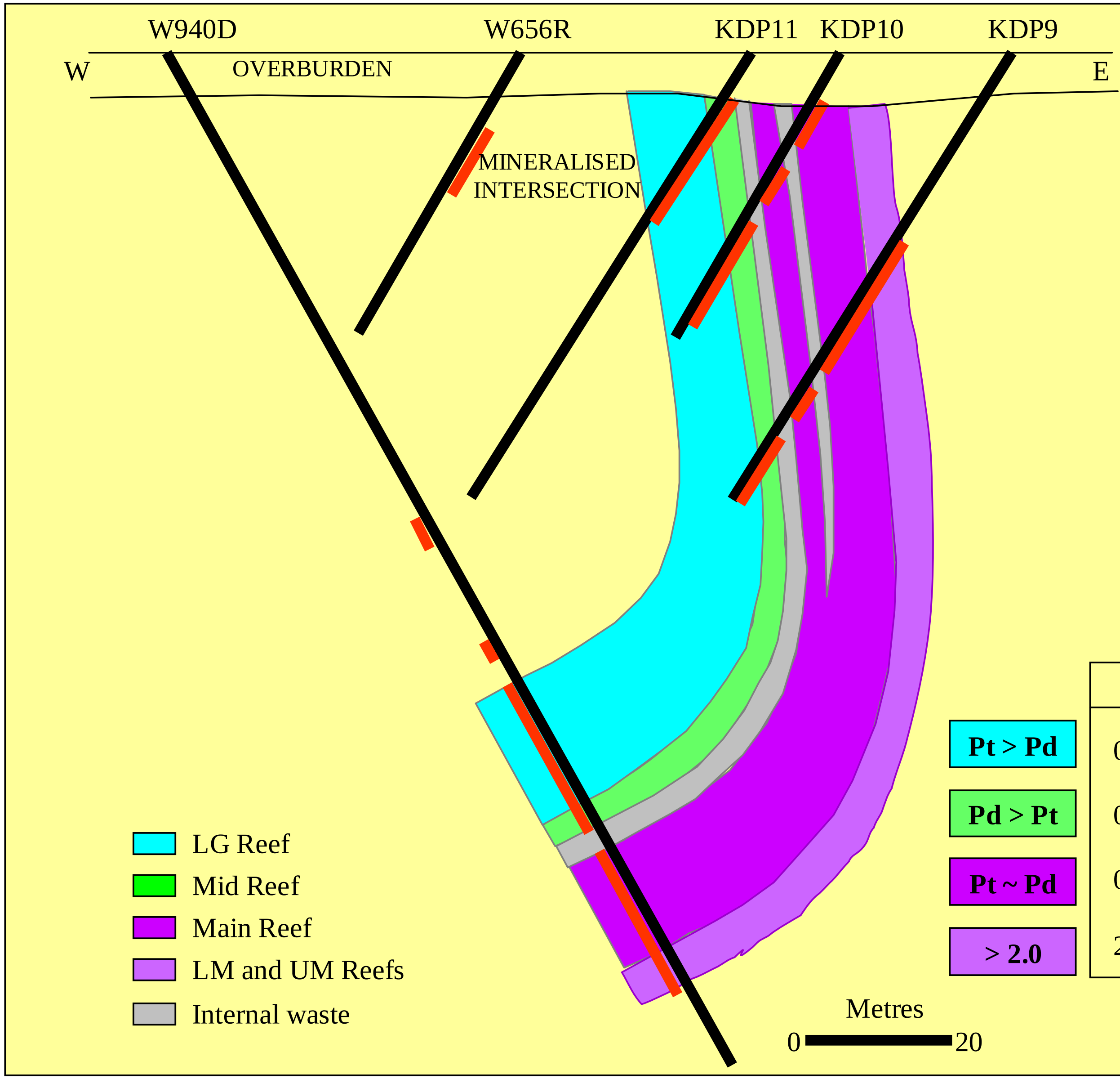
5.3 Polygon Estimation (Sirius, Vela, Serpens North, Serpens South, Crux)

For each drill section, polygons were constructed to define the different reefs. A distinction was made between weathered (oxide) and fresh (sulphide) material. The ore zone was defined as material above a Pt+Pd+Au cut-off grade of 0.5g/t.

The strike of the polygons was extended to midway between sections, except in cases where faults or dykes had an impact on the strike of the reefs, either cutting them off in depth or impacting on dips etc. Resources were calculated to 150m below surface, except in the case of Crux, where deep drill intersections justified extrapolation of these reefs to 230m.

Material volumes and tonnages were calculated for each polygon, taking cognisance of the differences in densities between oxide (3.1 g/cm^3) and sulphide material (3.25 g/cm^3). The densities used were obtained through measurement of dry and wet weight of different core pieces from 14 diamond drill boreholes.

From the drill intersections, weighted grades of Pt, Pd and Au were calculated for each polygon. Where two or more boreholes were present in the same polygon, the individual intersections were weighted to calculate the mean. Grades were projected vertically down in each polygon, instead of using adjacent blocks' values. Tonnages and grades for each polygon were used to calculate contained metals in that polygon and the sum of all polygons' metal contents were classified as inferred resource.



	Pt	Pd	Au	Total PGE
Pt > Pd	0.6	0.4	0.0	**1.0 g/t**
Pd > Pt	0.4	1.0	0.0	**1.4 g/t**
Pt ~ Pd	0.7	0.7	0.1	**1.5 g/t**
> 2.0	2.5	1.9	0.4	**4.8 g/t**

5.4 Geostatistics

Drilling sample assay results were used to determine average grades as distributed through the reefs in the Crater and Orion ore bodies, in order to calculate the resource contained. Frequency of the Pt and Pd grades indicate normal distributions for the occurrence of these metals. As illustration the distributions in the Crater Main Reef are indicated in Figures 5.3 and 5.4 and the frequency diagrams in Figures 5.5 and 5.6.

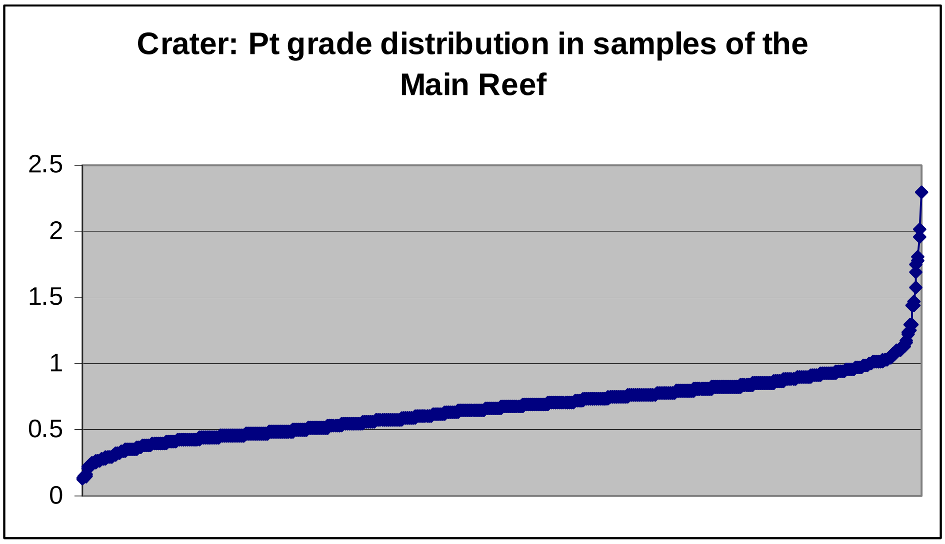


Figure 5.3: Crater Pt grades in the Main Reef: The diagram indicates the grades of 957 1metre borehole intersection samples. Grades plot on a straight line, except for some anomalous high grade samples

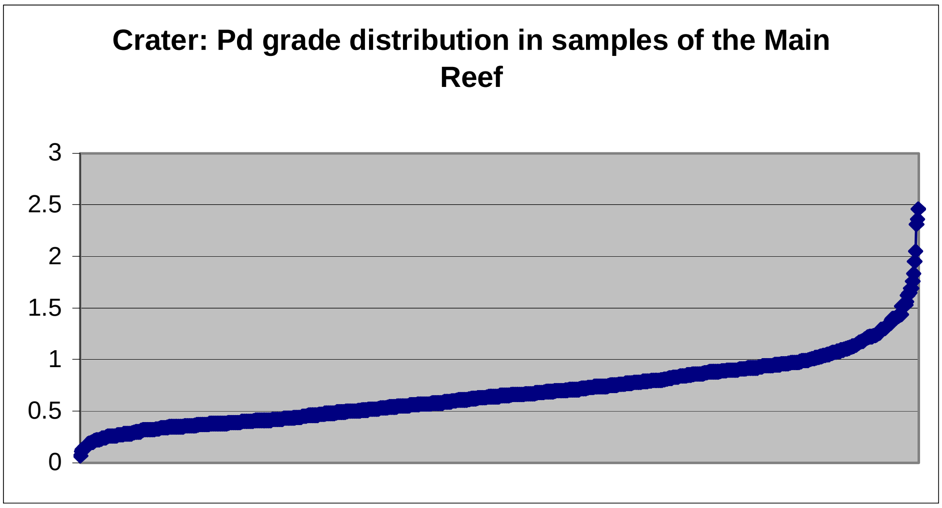


Figure 5.4: Crater Pd grades in the Main Reef: The diagram indicates the grades of 957 1metre borehole intersection samples. Grades once again plot on a straight line, except for some anomalous high grade samples

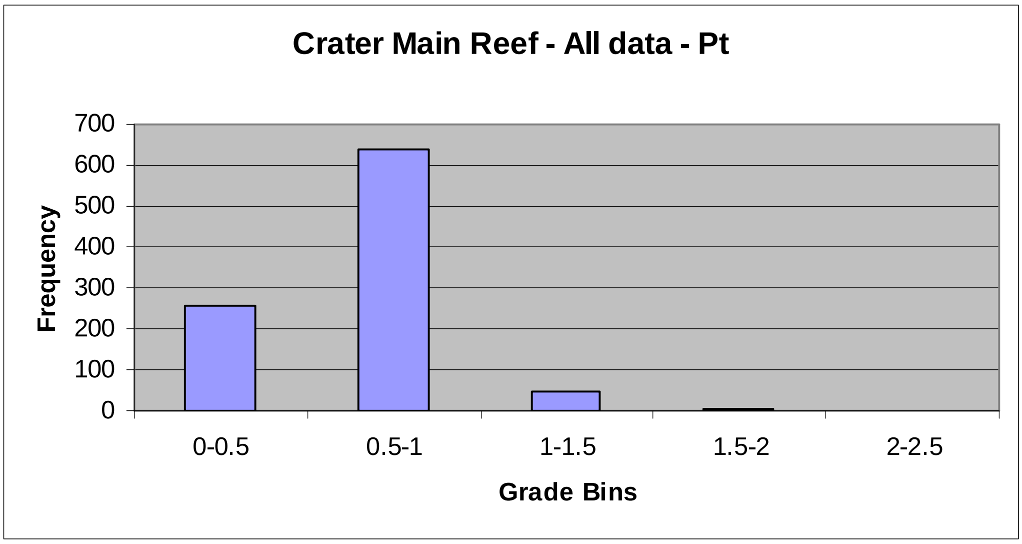


Figure 5.5: Pt frequency diagram Crater Main Reef.: illustrates a normal distribution of sample values

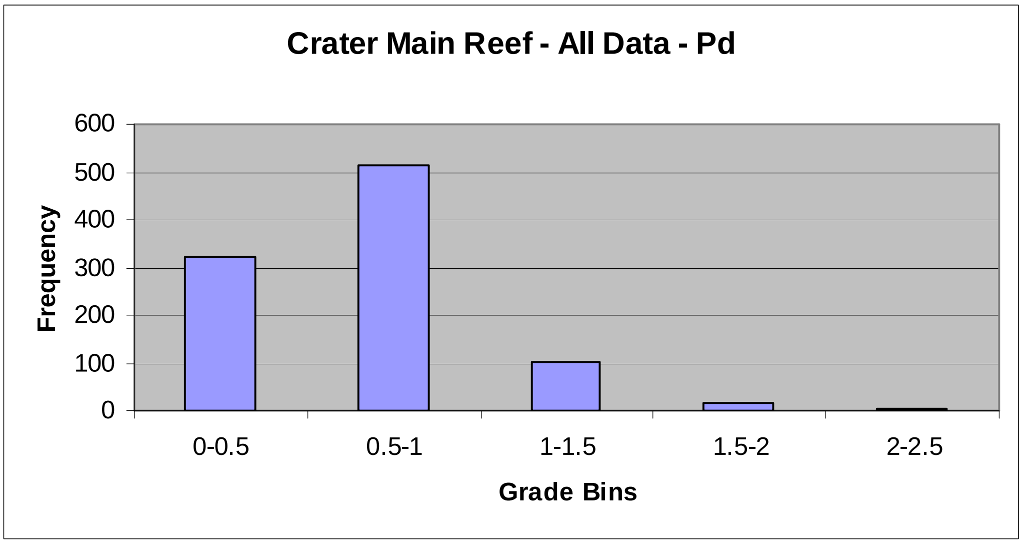


Figure 5.6: Pd frequency diagram Crater Main Reef.: illustrates a normal distribution of sample values

Low grade anomalies do occur in the sample sets, as the Pt+Pd+Au grade per sample were used to define reefs. Although this was also dependent on the ratio's between Pt and Pd, geological interpretation sometimes included samples on the boundaries of reefs, that may also have been assigned to other reefs. For this study, however, these samples are included in reefs as interpreted. Other reefs in the Crater ore body displayed similar grade distribution characteristics.

The following geostatistical characteristics were determined for the Crater ore body reefs:

1. LG Reef:

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	600	0	1.41	0.586	0.222	0.049	0.106	0.948	0.546	0.38
PD	600	0	1.25	0.440	0.177	0.031	0.633	1.627	0.411	0.40
AU	600	0	0.76	0.009	0.037	0.001	15.685	302.286	0.024	3.93

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.14	1.03
Pd	0.09	0.79
Au	0	0.08

2. Mid Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	547	0	1.48	0.307	0.155	0.024	1.741	7.398	0.288	0.51
PD	547	0	2.49	0.771	0.439	0.193	1.317	1.765	0.701	0.57
AU	547	0	0.16	0.013	0.018	0.0003	2.640	14.017	0.022	1.35

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0	0.62
Pd	0	1.65
Au	0	0.05

3. MR1

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	84	0.37	3.18	0.578	0.303	0.092	7.553	61.913	0.550	0.52
PD	84	1.21	2.81	2.005	0.398	0.159	0.021	-0.907	1.964	0.20
AU	84	0	0.78	0.043	0.085	0.007	7.859	65.681	0.039	1.99

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0	1.18
Pd	1.20	2.80
Au	0	0.21

4. Lower Main Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	214	0.58	3.58	1.705	0.590	0.348	0.412	-0.059	1.599	0.35
PD	214	0.51	3.96	1.888	0.726	0.527	0.101	-0.704	1.730	0.38
AU	214	0	0.34	0.021	0.029	0.0008	6.917	71.285	0.024	1.36

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.53	2.88
Pd	0.44	3.34
Au	0	0.08

5. Main Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	957	0.13	2.29	0.655	0.258	0.067	0.678	4.111	0.632	0.39
PD	957	0.01	2.46	0.663	0.334	0.112	0.929	2.416	0.611	0.50
AU	957	0	0.4	0.016	0.027	0.0007	6.448	68.351	0.025	1.64

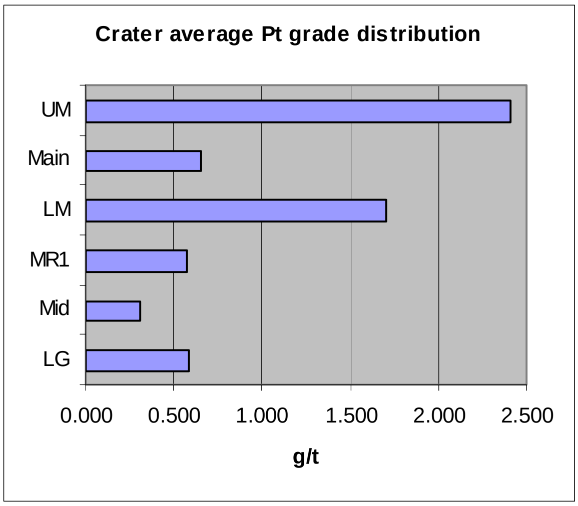
	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.14	1.17
Pd	0	1.33
Au	0	0.07

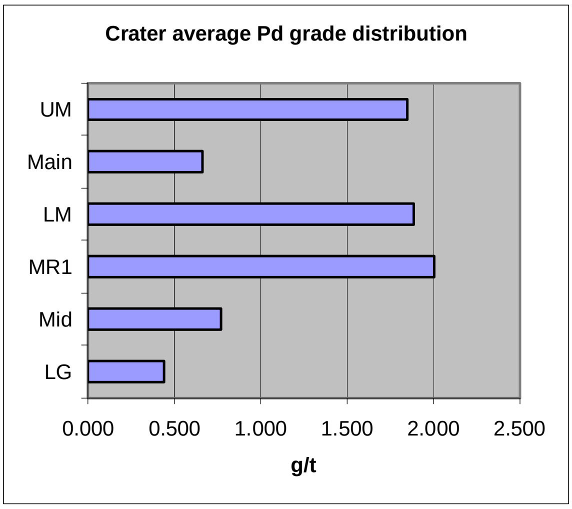
6. Upper Main Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	437	0.47	7.53	2.407	1.363	1.857	1.211	1.487	2.069	0.57
PD	437	0.03	6.88	1.848	1.348	1.816	1.195	1.680	1.295	0.73
AU	437	0	1.85	0.270	0.310	0.1	1.335	1.457	0.141	1.15

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0	5.13
Pd	0	4.54
Au	0	0.88

Following are graphical representations of the average grade distributions of Pt, Pd and total precious metals (Pt+Pd+Au) in the Crater reefs. The UM Reef contains highest grades of Pt, while the MR1 Reef contains highest Pd grades.





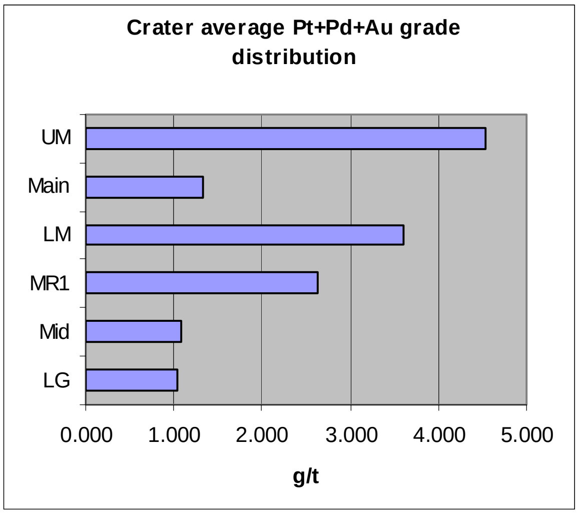


Figure 5.7: (a-c): Mean grades of metals in the Crater reefs

Following are the grade distribution and frequency graphs for 824 samples from the Orion Main Reef in figures 5.8 – 5.11.

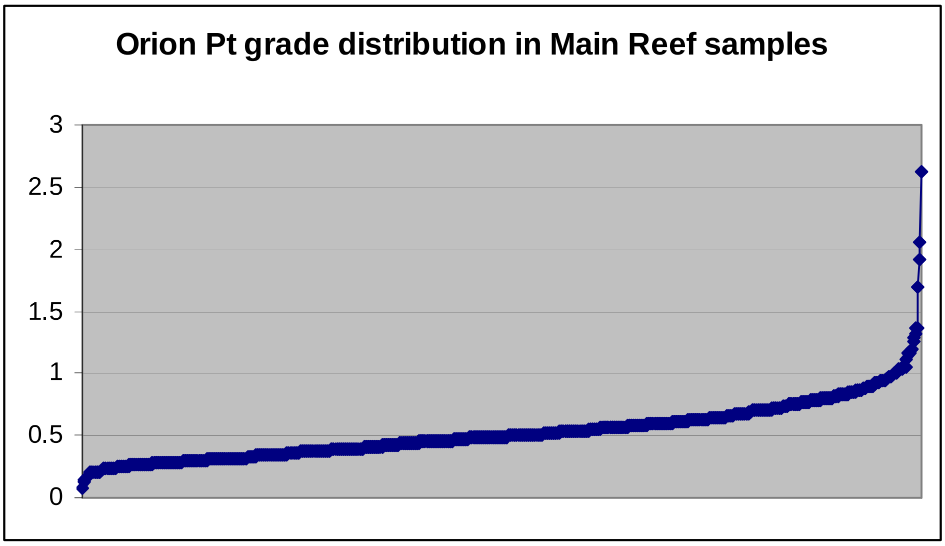


Figure 5.8: Orion Pt grades in the Main Reef: The diagram indicates the grades of 824 one metre borehole intersection samples. Grades plot on a straight line, except for some anomalous high grade samples

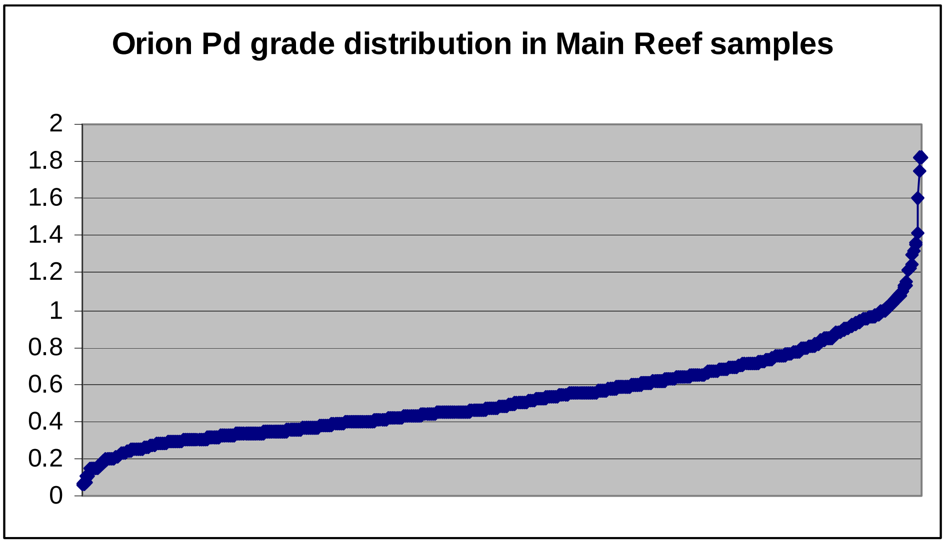


Figure 5.9: Orion Pd grades in the Main Reef: The diagram indicates the grades of 824 one metre borehole intersection samples. Grades plot approximately on a straight line, except for some anomalous high and low grade samples that occur in the series. This may be the result of some lower Pd grades, associated with high Pt grades in certain samples. As the boundaries of reefs were defined by Pt+Pd+Au grade in the samples, this occurrence of lower grade in certain samples, is possible

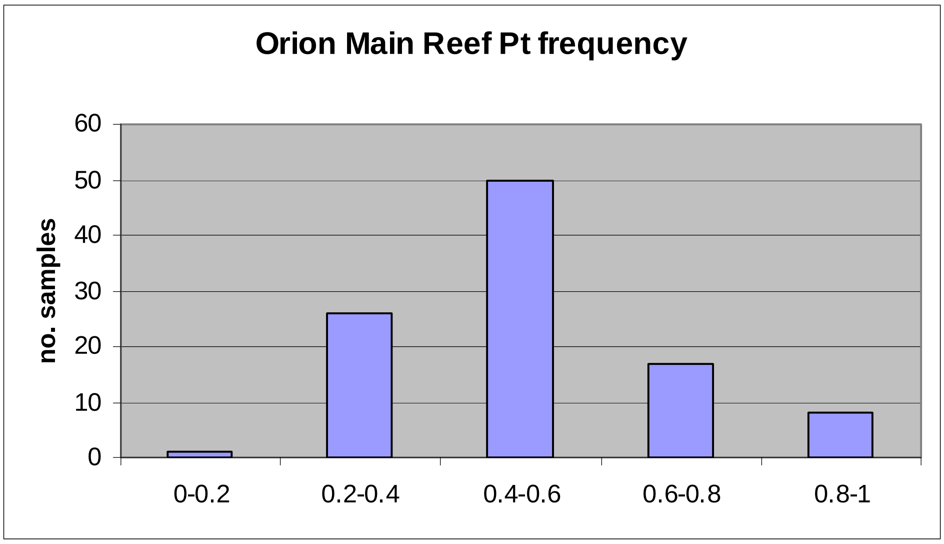


Figure 5.10: Pt frequency diagram Orion Main Reef: illustrates a normal distribution of sample values

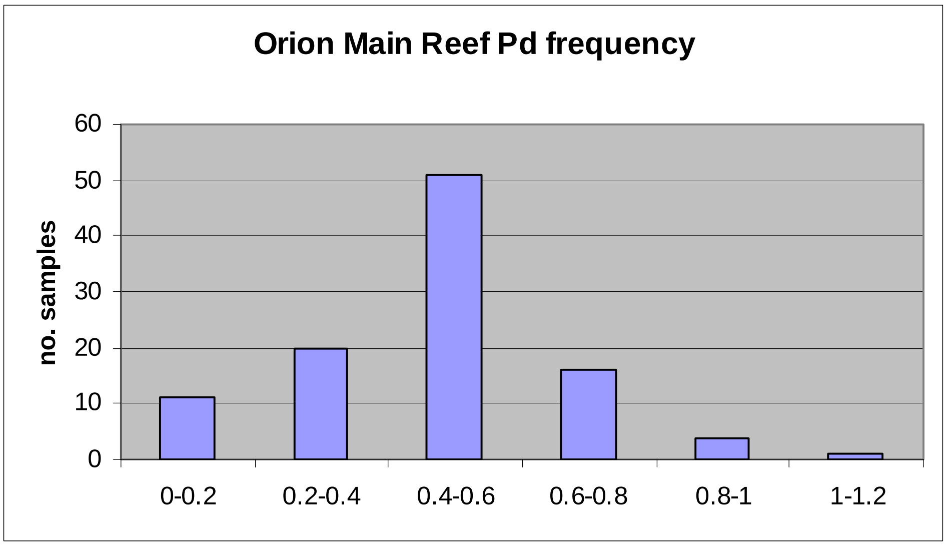


Figure 5.11: Pd frequency diagram Orion Main Reef: illustrates a normal distribution of sample values

Other reefs, in the Orion ore body, conform to the normal distribution of grades. As mentioned previously, inclusion of lower grade samples may be the result of geological interpretation of reef boundaries and not strictly according to metals ratios.

The following characteristics were determined for the Orion ore body reefs:

1. LG Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	33	0.09	1.5	0.541	0.234	0.055	0.969	1.277	0.522	0.43
PD	33	0.11	1.29	0.490	0.215	0.046	0.937	1.365	0.468	0.44
AU	33	0	0.4	0.013	0.039	0.0015	6.177	48.859	0.023	2.91

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.07	1.01
Pd	0.06	0.92
Au	0.00	0.09

2. Mid Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	283	0.07	1.26	0.327	0.214	0.046	2.015	4.451	0.286	0.65
PD	283	0.1	1.88	0.642	0.290	0.084	1.287	2.025	0.613	0.45
AU	283	0	0.40	0.017	0.044	0.0020	5.392	34.621	0.026	2.68

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.00	0.76
Pd	0.06	1.22
Au	0.00	0.11

3. MR1 Reef

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	59	0.21	1.35	0.677	0.222	0.049	1.087	1.927	0.671	0.33
PD	59	0.69	3.28	2.037	0.590	0.348	-0.024	-0.424	2.435	0.29
AU	59	0	0.18	0.023	0.040	0.0016	2.745	7.410	0.026	1.72

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.23	1.12
Pd	0.86	3.22
Au	0.00	0.10

4. <u>Lower Main Reef</u>

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	122	0.18	4.20	1.326	0.763	0.582	0.993	1.131	1.669	0.58
PD	122	0.143	6.45	1.438	0.985	0.971	1.925	6.323	2.113	0.69
AU	122	0	0.10	0.009	0.018	0.0003	2.394	6.383	0.021	1.87

	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.00	2.85
Pd	0.00	3.41
Au	0.00	0.05

5. <u>Main Reef</u>

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	824	0.08	2.63	0.530	0.238	0.057	2.072	10.823	0.513	0.45
PD	824	0	1.82	0.526	0.255	0.065	1.031	2.393	0.499	0.49
AU	824	0	0.98	0.035	0.119	0.0141	4.814	23.844	0.034	3.43

	95% CONFIDENCE LIMITS		
	LOWER	UPPER	
Pt	0.05	1.01	
Pd	0.02	1.04	
Au	0.00	0.27	

6. <u>Upper Main Reef</u>

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	SKEWNESS	KURTOSIS	GEOMEAN	CO-VAR
PT	219	0.20	7.50	2.109	1.235	1.525	1.089	1.494	4.560	0.59
PD	219	0.02	6.78	1.611	1.255	1.575	0.881	0.717	3.158	0.78
AU	219	0	1.26	0.342	0.340	0.1157	0.859	-0.343	0.232	0.99

Mean	95% CONFIDENCE LIMITS	
	LOWER	UPPER
Pt	0.00	4.58
Pd	0.00	4.12
Au	0.00	1.02

Following are graphical representations of the average grade distribution of Pt, Pd and total precious metals (Pt+Pd+Au) in the Orion reefs. The UM Reef has by far the highest Pt grades, while the MR1 Reef contains highest Pd grades. Overall, the UM contains highest grades of metals.

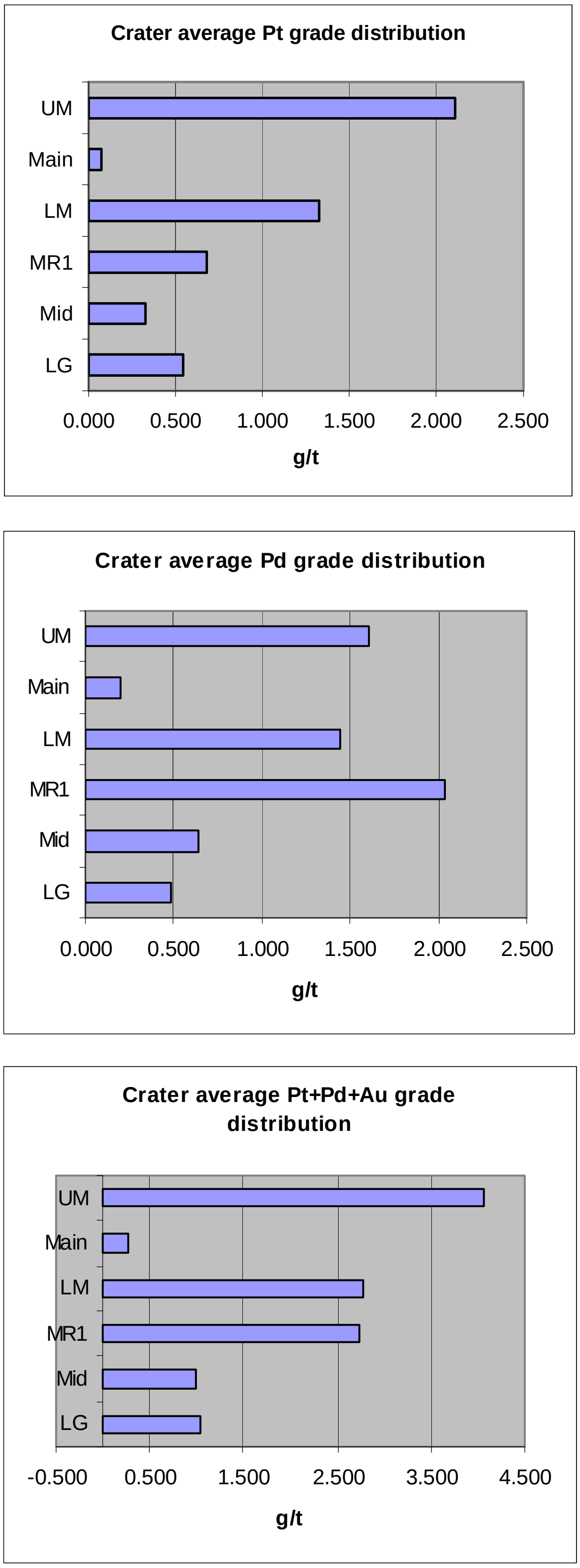


Figure 5.12 (a-c): Mean grades of metals in the Orion reefs

5.5 Resource Estimation

Table 5.3 summarises the Resource Estimations for the Kalplats Deposit. Tons are rounded off to nearest hundred thousand and ounces rounded off to nearest five thousand.

Fifty five percent (2.1Moz) of the total resource is located in the Main Reef Package, while the high grade reefs alone, host forty percent (1.5Moz) of the total resource.

The Crater resource as calculated using Datamine 3-D models and a cut-off of 0.5g/t Pt+Pd+Au is indicated below in the table.

Mains (UM, Main, LM)	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	464,546	525	1.13	475	1.02	22	0.05	2.20
Suphide Zone	6,902,419	7778	1.13	6970	1.01	313	0.05	2.18
Total	7,366,965	8303	1.13	7445	1.01	335	0.05	2.18

Mid Reefs (Mid, MR1)	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	281,856	93	0.33	263	0.93	4	0.02	1.28
Suphide Zone	4,090,794	1322	0.32	3967	0.97	58	0.01	1.31
Total	4,372,650	1415	0.32	4230	0.97	62	0.01	1.31

LG Reef	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	246,433	123	0.50	92	0.37	2	0.01	0.88
Suphide Zone	3,443,756	1671	0.49	1240	0.36	29	0.01	0.85
Total	3,690,189	1794	0.49	1332	0.36	31	0.01	0.86

Total	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	992,834	741	0.75	830	0.84	28	0.03	1.61
Suphide Zone	14,436,970	10,772	0.75	12,177	0.84	400	0.03	1.62
Total	15,429,804	11,512	0.75	13,007	0.84	428	0.03	1.62

Table 5.1: Crater resource as calculated from Datamine 3-D reef models

The polygonal estimates for the Crater resource, resulted in the following:
Total resource for all reefs: 15.6 Mt @ 1.7 g/t for 850 000 ounces Pt+Pd+Au contained and total for Main Reef package: 6.7Mt @ 2.3 g/t for Pt+Pd+Au content of 495 000 ounces.

Compared to the resource calculated from 3-D models of the reefs, the total resource is overstated by approximately 200,000 tons and 0.08g/t Pt+Pd+Au, while the Main Reef package resource was underestimated by approximately 600,000 tons, but grade overstated by 0.12g/t. Thus, total resource according to polygonal method is overstated by approximately 60,000 ounces, while Main Reef resource is approximately the same as calculated from the 3-D model method.

The Orion resource as calculated from Datamine 3-D models of reefs and a cut-off of 0.5g/t Pt+Pd+Au, is indicated below in Table 5.2:

Mains (UM, Main, LM)	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content Kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	911,110	837	0.92	709	0.78	68	0.07	1.77
Sulphide Zone	3,440,745	3,289	0.96	3,066	0.89	283	0.08	1.93
Total	4,351,855	4,126	0.95	3,775	0.87	351	0.08	1.90

Mid Reefs (Mid, MR1)	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content Kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	219,207	97	0.44	194	0.88	1	0.01	1.33
Suphide Zone	1,197,306	446	0.37	1,071	0.89	23	0.02	1.29
Total	1,416,513	543	0.38	1,265	0.89	24	0.02	1.29

LG Reef	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content Kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	1,293,211	1,020	0.79	972	0.75	71	0.05	1.59
Suphide Zone	5,536,663	4,221	0.76	4,588	0.83	314	0.06	1.65
Total	6,829,874	5,241	0.77	5,559	0.81	385	0.06	1.64

Total Reefs	**Tons T**	**Pt Content kg**	**Pt Grade g/t**	**Pd Content kg**	**Pd Grade g/t**	**Au Content Kg**	**AuGrade g/t**	**Pt+Pd+Au g/t**
Oxide Zone	2,423,528	1,954	0.81	1,875	0.77	140	0.06	1.64
Suphide Zone	10,174,714	7,956	0.78	8,725	0.86	620	0.06	1.70
Total	12,598,242	9,909	0.79	10,600	0.84	760	0.06	1.69

Table 5.2: Orion resource as calculated from Datamine 3-D reef models

The polygonal estimates for the Orion resource, resulted in the following:
Total resource for all reefs: 14.3 Mt @ 1.6 g/t for 735 000 ounces Pt+Pd+Au contained and total for Main Reef package: 8.1Mt @ 1.9 g/t for Pt+Pd+Au content of 495 000 ounces.

Compared to the resource calculated from 3-D models of the reefs, the total resource is overstated, by approximately 1.8 Mt, but grade underestimated by 0.09g/t Pt+Pd+Au, while the Main Reef package resource was overstated by approximately 3.8 Mt, but grade remained the same. Thus, total resource according to polygonal method is overstated by approximately 60,000 ounces, while Main Reef resource is overstated by approximately 230,000 ounces.

The large discrepancy in the Main Reef resource tons may be explained by different interpretations of reef thickness between boreholes, where no information is available. In order to be consistent with reporting of resources in the other ore bodies, the polygonal resource estimates for Crater and Orion were used in Table 5.3 below. Additional borehole information will be required to more accurately estimate the resource tons and grades in all ore bodies.

All resources estimated are in the inferred category.

	Total Resource			Main Reef Package			High Grade Reefs		
	Tons (000)	Grade (g/t)	Oz (000)	Tons (000)	Grade (g/t)	Oz (000)	Tons (000)	Grade (g/t)	Oz (000)
Crater	15 600	1.7	850	6 700	2.3	495	3 600	3.9	450
Orion	14 300	1.6	735	8 100	1.9	495	2 800	3.8	340
Crux	13 600	1.3	570	6 100	1.6	315	1 700	3.5	190
Sirius	11 100	1.3	465	2 600	2.0	170	1 500	3.2	155
Vela	10 500	1.3	440	2 400	2.0	155	1 100	3.2	115
Serpens N	8 900	1.4	400	3 600	1.7	195	1 500	3.4	165
Serpens S	10 800	1.3	450	5 900	1.7	320	900	5.1	150
TOTAL	**84 800**	**1.4**	**3,910**	**35 400**	**1.9**	**2,145**	**13 100**	**3.7**	**1,565**

Table 5.3: Kalplats Mineral Resource – Inferred Total Contained Precious Metals Pt+Pd+Au (rounded to nearest 5koz)

Chapter 6 MINING

Factors considered when selecting the appropriate mining method for the Kalplats ore bodies, include the depth and dimensions of the ore bodies, distance between the ore bodies, grade of ore bodies and costs related to specific mining methods.

6.1 General considerations

Mineralisation in this area is close to surface, being overlain by Kalahari sands and calcrete only. The weathered/fresh rock interface is at an average of 35 – 40m depth below surface and oxide ores from this weathered zone display lower recoveries and grades, because of leaching and high grade weathering in this zone.

Average maximum depth of the sand is 5m. Calcrete is not commonly developed in the Crater area. Quartz-albite veins and other structures (thrusts and normal or reversed faults) are common and a mining dilution factor of 10% was applied for internal waste. Displacement along shears in this area is limited.

3-D models of the ore bodies and relatively low grades as reported through geochemical analyses, dictates that the optimal mining route to follow here will be an open pit method. Depth extent of mineralisation varies over the different ore bodies, but mineralisation occurs from close to surface up to at least 150 meters depth over all of these.

Consideration has been given to the possibility of progress to underground mining from the base of the final open pit, focusing on the narrow high-grade reefs (UM and LM). Underground mining will however be difficult because of structural complications and poor visible distinction between ore and waste, as noted previously in Chapter 4. It was established that this method would have high capital requirements, longer leading times and eventual slower production rates than required to make the project financially attractive. This option may be considered in future in the event that there is a dramatic increase in market value of these metals.

6.2 Open Pit Mining

The mining at the nearby Kalgold open pit is done by DBE, a mining contractor, and as the operation has similar characteristics to the PGM project, mining costs from this operation were used in determining the operating costs of the project.

Different cost parameters were used for stripping of sand (dozing), for free dig to 35 meters (highly weathered zone) and for drilling and blasting below this depth, as costs increase incrementally with depth and mining of fresh ore. Loading and hauling of ore is more expensive than that of waste. An increment of 5% per 20m depths was used for the increase in mining costs. Maximum average open pit angles of 60° were determined to be possible in the fresh sulphide zone (through determining failure angles of this type of material), while these will be lower in the weathered zone, in the order of 30°. Mining costs are listed in Table 8.1.

The option of selectively mining out the higher-grade portions and main reef first, has been considered. However, in view of the ore body morphologies, the mining implications would still be the same – the volume of rock to be moved doesn't change significantly. Selective mining will have more of an impact on the mineral processing plant operations and recovered grade. This option can still be considered, as it will increase initial revenues expected from sales, thereby reducing the payback period.

Optimal production rate, as determined from reserve, is 153,000tpm, as discussed in Section 8, which equates to 1.8 million tons per annum. For scoping studies, the production rate of 1.8 million tons of ore per annum was considered. At an average strip ratio of 1:5, that amounted to a total of 10.8 million tons of rock mined per annum. Based on Datamine 3-D ore body models of the Crater and Orion deposits, pit designs were done and mine layouts designed. The pit depths were 200m for the Crater and Orion ore bodies. Where borehole reef intersection data was not available below 100m on certain section lines, reefs were projected downward, using interpolation of information available on nearby section lines. Simplified ore body models are illustrated in figures 6.1 and 6.2. Pit optimization exercises would be done should a full feasibility study be commenced.

Pit design and estimated maximum depths of mining were determined by using mine operating costs as anticipated and revenues expected from mining. As mentioned previously, these were not full optimisation exercises, but merely scoping studies reliant on limited information as available. Mining costs are discussed under section 6.4 and section 6.5 gives some indication as to estimated mineable reserve.

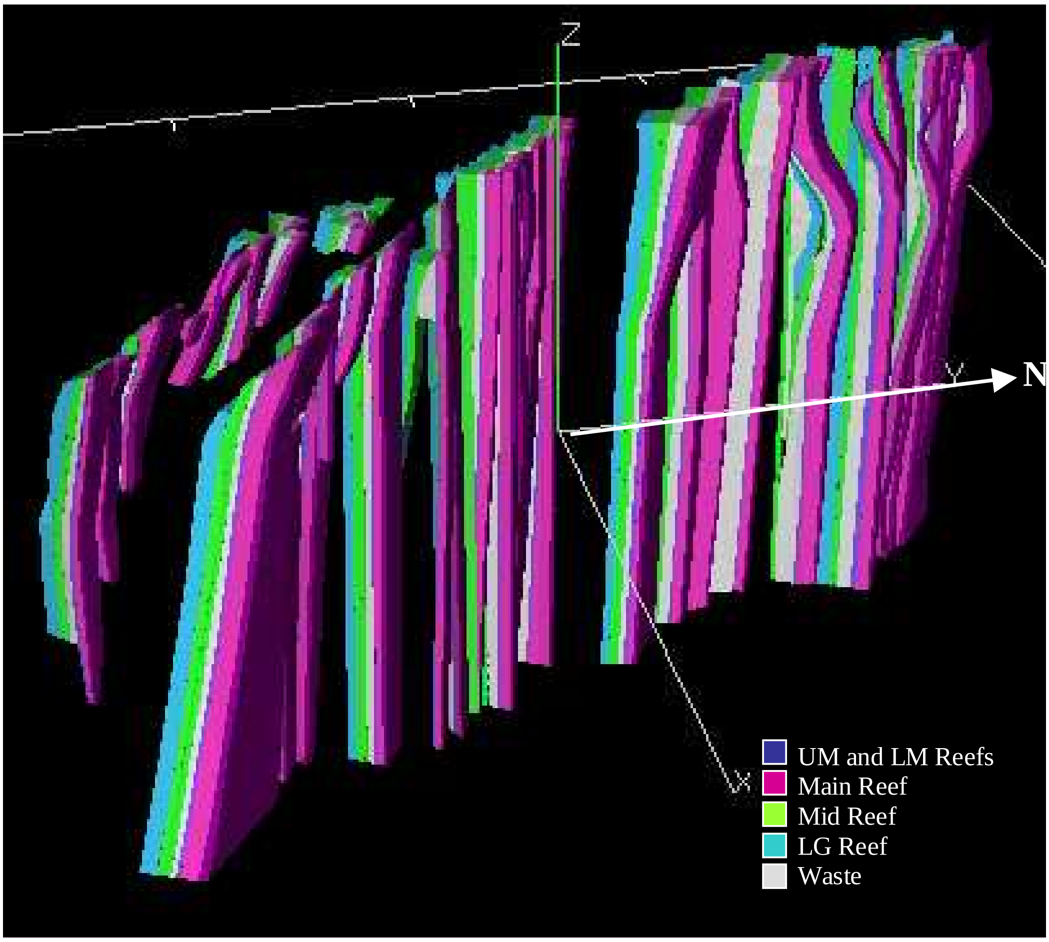


Figure 6.1: Crater 3-D Ore body Model: The model shows only the reefs and internal waste bands and not any dykes or other cross-cutting structures.

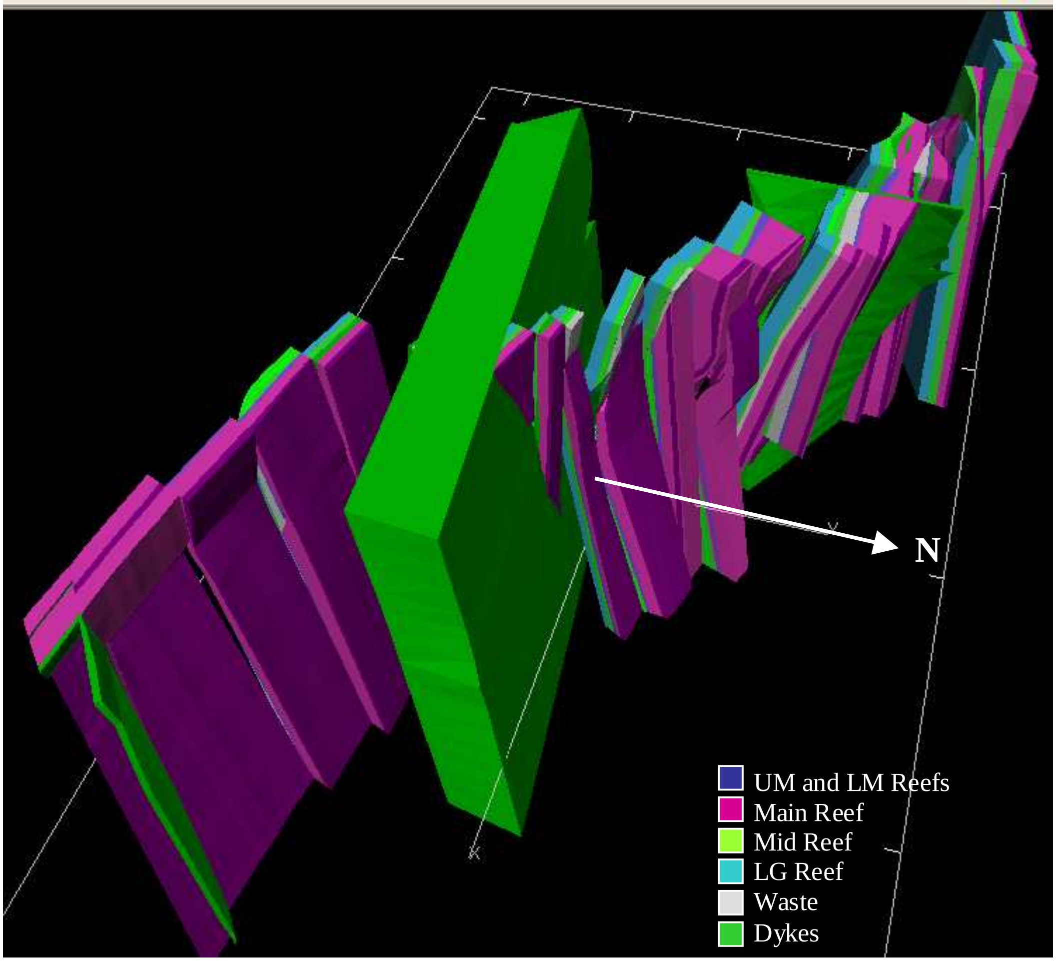


Figure 6.2: Orion 3-D Ore body Model: The model shows reef bands and internal waste, as well as significant dykes that cross cut the reefs.

6.2.1 Outstanding concerns

A number of outstanding concerns still need to be considered in the planning and mine layout, which will have a direct impact on profitability of the operation. These are shortly mentioned here.

a) Smelting:

The potential for erecting a smelter for the purpose of metal recovery from concentrates has been considered, but for economic reasons, the logic followed was that of the concentrate being transported for toll smelting, to a smelter in Rustenburg. If this process is to be executed, a smelting fee will be payable. A toll fee of 20% on the revenue from sales of metals was included in the discounted cash flow model.

b) Transport:

The mass pull in the concentration plant will be an important factor to consider, for the purpose of determining transport costs. It would be wise to maximise recoveries, but limit mass pull in order to recover lower volumes of concentrate. A balance needs to be achieved between recoveries and mass-pull. Mass-pull used for the purpose of the discounted cash flow model is 3%.

6.2.2 Infrastructure

With regards to infrastructure, the area is readily accessible by road and rail, with a 25km gravel road connection to the N18 national route between Mafikeng and Vryburg and a railway station at Setlagole, which is approximately 30km north on the N18. Electricity will be obtained from a 132kVa Eskom power grid, approximately 10km north of the envisaged plant site.

Water supply to the mine and processing plant may be problematic and a groundwater resource study was initiated to deal with this issue. Water requirements to the mine and plant will be in the order of 5 Ml per day. Approximately 30km of groundmagnetic and IP resistivity surveys were done to aid in the identification of suitable water bearing structures. The magnitude of resistivity and association with faults, dykes or lithological contacts were used to rate targets for water borehole drilling. A total of 36 water boreholes were drilled over an area of 7.5km x 2km, stretching from the Crater body to the Serpens North body.

The average borehole depth was 60m, except for an old farm borehole, which was deepened to 135m. Water was intersected in aquifers or structures in these boreholes between depths of 20m and 70m. Boreholes were tested using a v-notch, and boreholes yielding in the order of 10 000 l/hr (5 boreholes) were selected to undertake detailed step-test and recovery test work and determine constancy.

Results from the water borehole testing were forwarded to groundwater consultants for review and analysis. They estimated that the current annual abstraction from boreholes in the 184 km^2 catchment area (based on certain assumptions regarding farm water boreholes) is 1856 m^3/day. The potential for safe abstraction of groundwater over the Kalplats area (36 km^2) was estimated as follows:

Rainfall recharge:	+ 986 m^3/day
Existing abstraction:	- 370 m^3/day
Volume available:	**616 m^3/day = 616kl/day**

The yield from the four best boreholes drilled for the purpose of this study was estimated to be 306 m^3/day. It is then deduced that another 4 successful boreholes, targeted in different aquifers, would be sufficient to enable abstraction of 616 m^3/day. This would require in the order of 40 more water boreholes to be drilled to locate suitable boreholes (based on past success rate). This supply will not be sufficient for the requirements at Kalplats.

Water may also be piped to site from dams. The nearest sizeable dam is at Kgoro, 20km away. Costs related to piping water across this distance will be high and further feasibility studies should prioritise this issue. Similar problems were experienced at Kalgold, but the issue was solved through establishing a well-field and dam close to the mine. Kalgold's daily requirement of 6.5 Ml is met by the well-field. There is a possibility that a well-field and dam at Kalplats may also prove to deliver sufficient water, but re-circulation and treatment of water through the plant, will be necessary. No in-depth studies have been done in this regard and the author only treated costs related to establishment of well-field and dam in the discounted cash flow model.

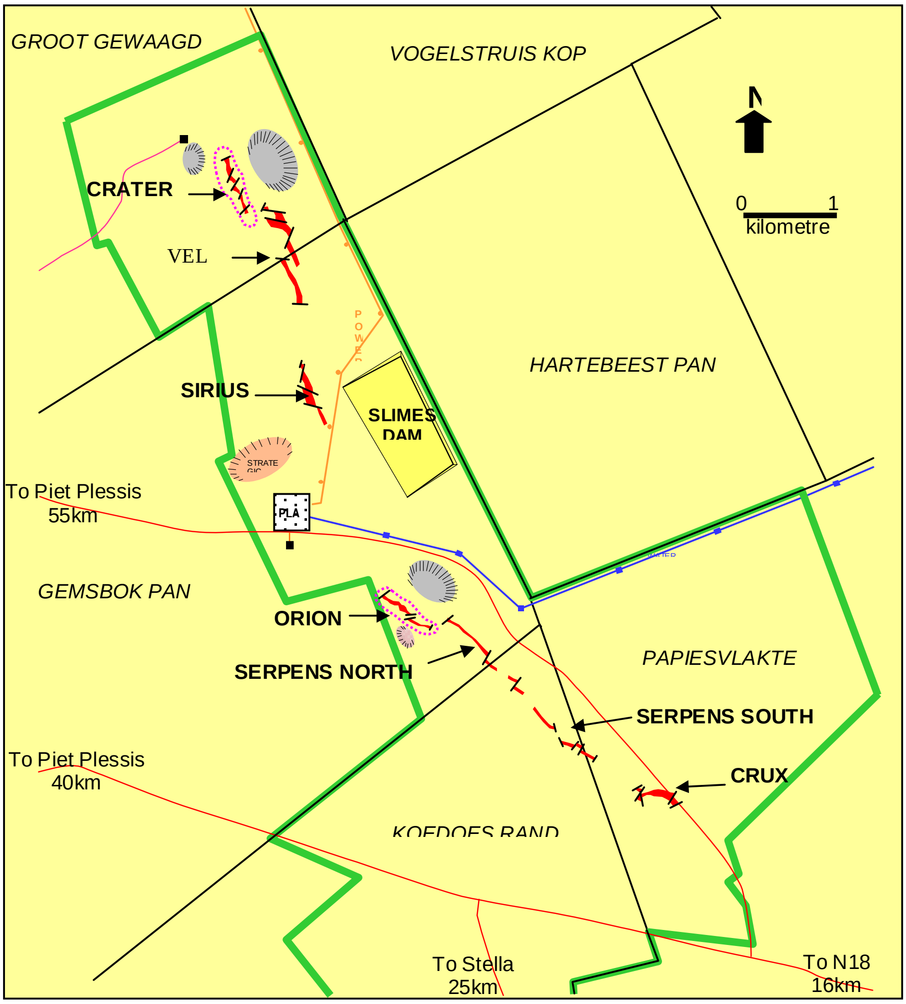


Figure 6.3: The Kalplats Mining Lease and Planned Mining Infrastructure
As no ore body is developed between Sirius and Orion, it was decided to build the plant infrastructure in this area, approximately equidistant between Crater and Orion.

6.3 Transport and Materials Handling Strategy

Three driving factors are of importance in developing a transport and materials handling strategy, namely technical performance, profitability and service effectiveness.

6.3.1 Materials Handling

Fleet size requirements will be determined by the mining contractor. The following section only mentions the equipment type requirements for the operation. Costs related to different items are listed in Section 6.4.

Initial excavation of overburden, consisting of sand and highly weathered rock to a depth of approximately 15 meters, will be done with excavators. The material is easily breakable and can be scooped up with little effort. The excavators will load material into haulage trucks (25 tons). The trucks will move this material to overburden dumps in an area outside the open pit perimeter and away from planned haulage ways.

While excavation continues over the perimeter surface area of the pit, bedrock, as ore and waste close to ore, will be blasted. Drill rigs mounted on trucks will be used to drill blasting holes. Explosives will be carried to the blast hole sites by smaller vehicles, typically 1-tonner trucks. After blasting, excavators will load the material into trucks, with ore carried to the processing plant and waste carried to the waste dumps next to the open pit. Care must be taken to minimize distances of hauling waste to dumps, resulting in the placement of dumps as close to the open pit perimeter as safely possible.

Cycle times should be designed to accommodate excavation rates, without any bottlenecks in loading, caused by trucks not being available to haul material. Channel design and continuous monitoring will eliminate the possibility of too many trucks available and no material or too few excavators to do loading.

Schedules should specify the rate of blasting required to produce sufficient volumes of material to the plant, with accompanying schedules to specify location of excavators and rates of loading for each, as well as truck requirements at these different locations. The volumes of waste to be moved will initially be substantially more than volumes of ore, in the order of a ratio of 10:1 waste to ore. A proportion of the haulage fleet should be dedicated to waste removal, while another proportion should be responsible for ore haulage to the plant.

Taking into consideration that the distance to the processing plant will be longer than the distance to the waste dumps, these cycle times for loading and vehicle requirements will differ dramatically. As haulage distances become longer with deepening of the pit, the number of trucks available for ore haulage should be increased. Similarly, as requirements for waste haulage decreases with deepening of the pit, the number of haulers should be reduced. Waste haulers can be re-deployed to haul ore to the plant. Schedules should also include provision for maintenance on haulers, which has to be carried out periodically.

Vehicle workshops will be located close to the open pit, where maintenance and servicing can be carried out. Suppliers of parts and consumables will deliver their products to the workshop on an order basis. The principles of just-in-time delivery can be applied to the order cycle of these products, with larger deliveries less frequently.

Electricity will be conveyed to the mining site and workshop with cables from a substation to be established close to the high-voltage power lines, approximately 1.5 km from the open pit location.

6.4 Mine Operating Cost

Mining costs were derived from actual Kalgold contract mining costs. These costs are a realistic guide to indicate expected costs at the Kalplats project, as mining characteristics and environment are similar. Table 6.1 summarises the mine operating costs and these are discussed further under Section 11.4.2. These costs include drilling, blasting and loading, as well as grade control. Provision is also made for rehabilitation of pits, dumps and slimes dams.

Mining Operating Costs	R/ton
Dozing	3.63
Freedig	5.97
Oxide waste mining	8.25
Sulphide waste mining	12.1
Oxide ore mining	11.55
Sulphide ore mining	14.3
Rehabilitation	0.5

Table 6.1: Mining Operating Costs: Actual costs related to mining at Kalgold open pit

6.5 Mineable Reserve

Based on the studies related to 3-D ore body modeling, mine layouts and planning, mine operating cost and metallurgical processing costs (Section 7.5), an attempt was made to estimate the mineable reserve in the Main Reef package. This does not constitute a proven or probable reserve, but simply an indication based on best estimates available. The cut-off grade used for determining the reserve was 1g/t Pt+Pd+Au and mining depths for Crater and Orion was from 20m to 200m, while for the other ore bodies it was from 20m to 150m.

Ore body	Tons (000)	Pt (g/t)	Pd (g/t)	Au (g/t)	Pt (000 oz)	Pd (000 oz)	Au (000 oz)	Pt+Pd+Au (000 oz)
Crater	6,125	1.16	1.04	0.08	228	204	15	447
Orion	7,985	0.93	0.83	0.09	238	213	23	474
Sirius	1,794	1.06	0.98	0.06	61	57	4	121
Vela	2,170	1.05	0.95	0.11	73	66	8	147
Serpens N	2,084	0.87	0.79	0.07	58	53	5	115
Serpens S	2,324	1.05	0.83	0.12	78	62	9	149
Crux	3,572	0.96	0.96	0.07	111	110	8	229
Total	26,054	1.01	0.91	0.09	847	765	72	1,682

Table 6.2: Estimated Pt, Pd and Au reserve ounces contained. Cut-off used was 1g/t Pt+Pd+Au and mining depths to 200m for Crater and Orion and to 150m for other ore bodies.

Chapter 7 METALLURGY

7.1 Floating characteristics

As previously mentioned in the Geology section under 'Applied Mineralogy' (Section 4.7), platinum group minerals occur as very fine grains (3-17µm) on silicate, oxide or base metal sulphide grain edges or locked into these assemblages. It is important to understand the behaviour of these grains during flotation.

Float ranks were calculated for the mineralogical samples. The float rank is defined as the ratio of the area of the floatable component (PGM and Base Metal Sulphides) to the area of the particle. Liberated grains will have a very high float rank of 1.0.

Average float ranks for these samples are shown in Table 7.1. The majority of liberated PGM's and Base Metal Sulphide host-minerals are fast floating.

Float rank/ Liberation Index	**PGM Volume %**	**Percentage exposed floatable surface**
<0.2	38	15
0.2 – 0.4	<1	<1
0.4 – 0.6	3	<1
0.6 – 0.8		-
0.8 – 1.0	59	59

<u>Table 7.1: Float Rank of PGM containing-particles</u>: 59% of grains could be liberated efficiently, while 38% were not liberated.

<u>7.1.1 Factors influencing beneficiation of minerals</u>

The mineralogical studies and float rank tests, identified a few factors which influence the beneficiation of valuable minerals.

1. PGM grain size: all the PGM's are smaller than 17µm, which when liberated should float readily under ideal conditions. There is a risk in losing some of the finer PGM particles during cleaning stages.

2. PGM mode of occurrence: the majority of PGM's in the samples are associated with silicates. PGM inclusions in amphibole generally occur as extremely small isolated grains (~2µm) in large amphibole particles (~50µm), implicating a small possibility of liberation.

PGM's associated with chlorite or epidote might be liberated with further milling, as the average diameter of these grains is ~30µm. Furthermore, chlorite's Moh's hardness of 3 should enable it to break up readily.

PGM's locked up in quartz frequently occur as clusters and milling should liberate some of these.

PGM's occurring along grain boundaries in silicate or silicate/oxide phases should be liberated readily by milling, as these particles will probably fracture along grain boundaries.

3. PGM-type: Sulfur-containing PGM's will float faster than their PGM-alloy counterparts.

7.2 Metallurgical Testing of Drill Samples

As a result of findings during the mineralogical studies, extensive metallurgical test work was conducted, in an attempt to better understand the behaviour of and impact on recoveries, of the small particle sizes of mineral grains and the grains locked up in silicates. Test work was conducted on drill core samples and on a bulk sample taken from the Crater deposit. Optimum conditions as obtained from the bench scale test work were applied during the subsequent pilot plant test work on the bulk sample.

<u>7.2.1 Drill samples</u>

Various composite drill core samples were metallurgically tested under different conditions with the aim of designing the optimum processing plant to most cost effectively beneficiate PGM's, as occurring in the ore bodies. Properties for both oxide and sulphide ores have been determined in these studies.

<u>7.2.2 Testing procedure</u>

Standardisation of processes was necessary for the metallurgical test program and the assumption was made that one would need to pull a high mass of material at the rougher stage in order to get a high overall PGM recovery. As stated earlier, a significant amount of PGM's are locked in silicates and intensive milling would be required to liberate these particles effectively. A PGM operation in the US (Stillwater), with similar constraints have had success in improving their PGM recovery significantly by using the high-mass pull, concentrate re-grind approach (Duarte et al., 2003).

a) Scoping mass pull tests

Scoping rate flotation tests were conducted on one sample under three conditions and mass pulls and recoveries calculated for each condition. Flotation time and froth removal method was kept constant, while reagent dosages and types varied. Details on these conditions are not available in this report. Results are displayed in Table 7.2.

Condition	Mass Pull (%)	PGM Recovery (%)
1	8.2	64.5
2	14.1	66.7
3	11.5	65.7

Table 7.2: Results of scoping mass pull tests: Highest PGM recovery obtained was 66.7%

The low overall recovery is related to the mineralogical aspects discussed earlier and nominal grind size distribution after primary milling.

Condition 2 resulted in the highest concentrate mass pull and PGM recovery, with no depressant being added. When comparing results it is noted that the other methods did not yield significantly lower recoveries, but condition 2 was applied in subsequent testing.

b) Rougher rate tests

These test were done in order to determine the maximum recoveries achievable for each of the ores using a primary mill-flotation circuit at a nominal grind of 80% -75microns.
Different ore types are summarized in Table 7.3.

Sample	Zone	Lithology	Depth (m)
H2	Sulphide	Magnetite gabbro	63 – 64
H3	Oxide-sulphide	Gabbro	39 – 40
H4	Sulphide	Gabbro/Magnetite gabbro	54 – 60
H5	Sulphide	Highly chloritised gabbro	58 – 68
H6	Oxide	Highly chloritised gabbro	11 – 16
H7	Sulphide	Chloritised gabbro	75 – 80

Table 7.3: Ore types per sample: Samples H3 and H6 were taken from the oxidized horizon, while the other samples were fresh rock from the sulphide horizon.

Ore Type	Mass Pull (%)	Maximum PGM Recovery (%)
H2	14.1	68
H3	22.9	42
H4	26.2	80
H5	26.5	58
H6	38.2	40
H7	20.9	60

Table 7.4: Maximum Recovery values for Different Ore Types: Recoveries from fresh rock were higher (58-80%) than recoveries from oxide samples (40-42%)

Laboratory rougher rate flotation tests were conducted on 6 samples using condition 2 of the scoping mass pull tests. Figure 7.1 shows plots of cumulative recovery vs. time.

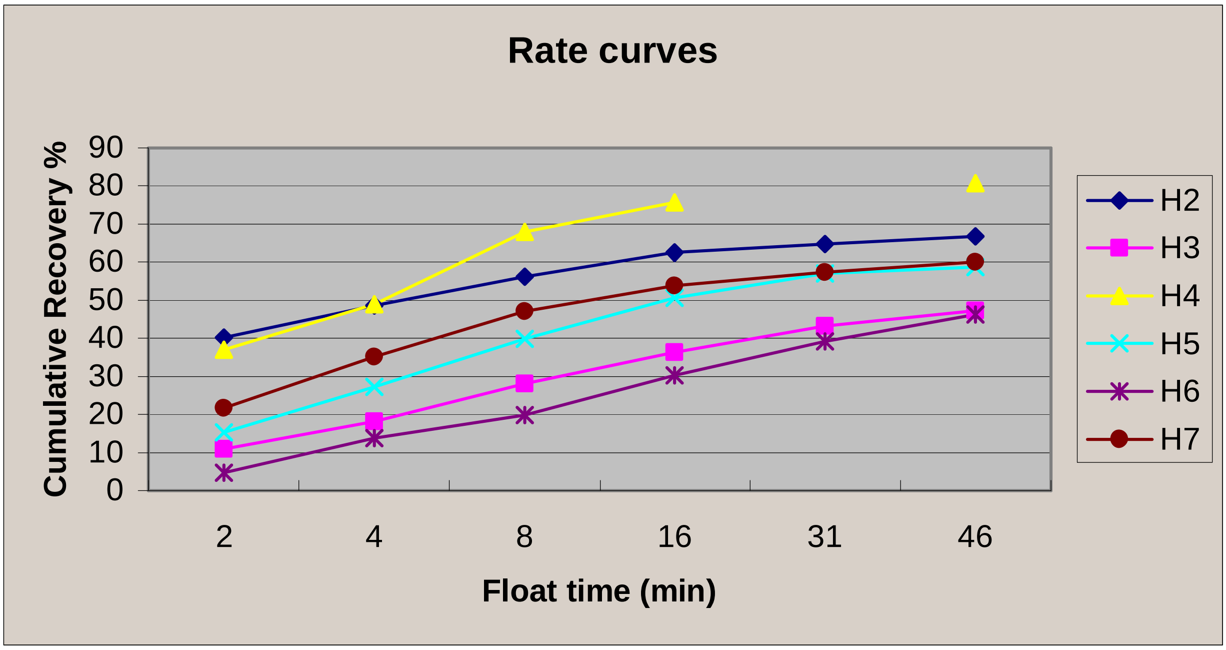


Figure 7.1: Rate curves: H2 – H7: Optimal recovery was obtained after floating for 46 minutes, but for most ore types tested, recoveries did not improve significantly after 31 minutes float time.

Recoveries varied between 45% and 80%. Samples H3 and H6 show low PGM recoveries and low grades, compared to the other samples. The viability of treating the oxide ores is not very high if a primary mill-flotation stage is the only option. H2, H4, H5 and H7 are all treatable using a primary flotation option, although recoveries are still low at between 60-80%.

c) Grinding characteristics of ores

A milling curve was set up on ore H2. The time taken to achieve the required 80% -75micron grind was used to mill the other ores. The grind achieved for each ore is indicated by the percentage -75micron in the rougher tailings in figure 7.2.

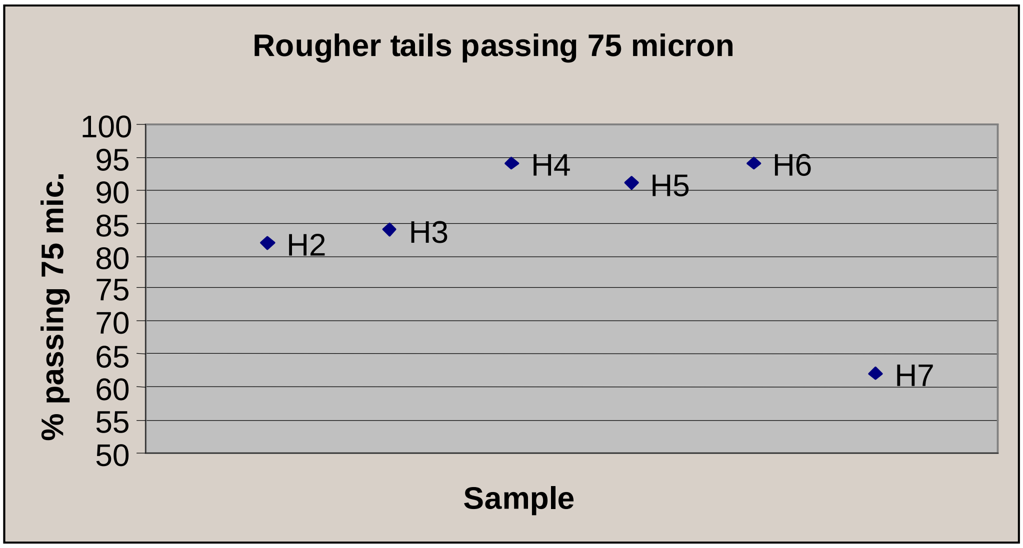


Figure 7.2: Rate Float Rougher Tailings Sizes H4, H5 and H6 were easier to grind, while H7 seemed to be much harder. H7 was the deepest sample taken.

d) Recovery by size curves

A recovery by size curve was compiled for H2 (Figure 7.3). A representative sample of the feed and rougher tailings from the rate tests was split into various sizes and their PGM contents assayed while the mass of each fraction was recorded. The recovery was calculated by determining the percentage of PGM's entering the system in the feed that reports in the concentrate for each size fraction.

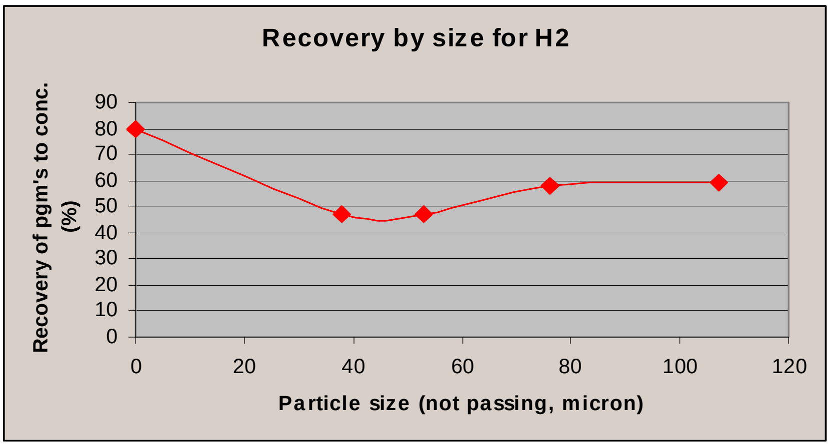


Figure 7.3: Recovery by size for H2 test: Best recoveries were obtained from the portions of H2 tailings sample coarser than 78µm and finer than 40µm. This is a possible indication that liberation of PGM's on grain edges is possible at >78µm grind and liberation from locked-in PGM's is probable at a fine grind, below 30µm.

e) Process selection cleaning tests

i) Three types of tests were conducted:

Test 1:

Rougher conditions set up in the scoping mass pull tests were applied in this test and the primary rougher concentrate was directed to a three stage open circuit cleaning circuit. Primary cleaner tails were combined with primary rougher tails and this was further milled to ~80% -30microns. The concentrate produced from this material was directed to an open circuit secondary cleaner circuit.

Test 2:

The object of the second testing method was to determine the value in moving some of the milling energy from the secondary mill to the primary rougher concentrate regrinder stage. Pulling the froth more vigorously in an attempt to increase the PGM recovery in the primary circuit increased the mass pull from the primary rougher. A stirred ball mill was used to treat the primary rougher concentrate; introducing 80kwh/t of energy and the secondary milling time was reduced by 50%. Final grind in this process was around 70% -38microns.

Test 3:

In the third cleaning test, only the primary circuit was used. The rougher mass pull was also increased for this test and further milling was done in a stirred mill to an energy input of 20kwh/t.

ii) Results of Cleaning Tests on Drill Samples

All process selection tests were done on samples H2 and H3, while only the first test was performed on the other samples. It would have been useful to treat all the samples with all the tests in order to have more representative results for scientific studies, but after treatment of H2 and H3 showed consistent total recovery behaviour, the following testing was limited to one method.

When comparing the three cleaning tests done on H2 and H3, it is evident that the highest total recoveries were obtained through using *test a*, with following *test c* and lowest recoveries through *test b*. This was determined through comparing calculated PGE grade with measured grades.

Based on this, it was decided to conduct the tests on the rest of the samples according to *test a*.

<u>H2 testing:</u>

Test 1: Most of the PGE's were recovered from the primary re-recleaner concentrate and the secondary rougher tailings. The primary circuit accounted for 53.1% of total PGE's recovered and the secondary circuit for 46.9%. Ninety two percent of the total mass of material ended up in the secondary rougher tailings and accounted for a PGE recovery of 31% of total. Total recovery grade calculated was 2.67g/t compared to measured head grade of 2.53g/t, the highest of all three tests conducted

Test 2: Similar situation to test 1. The primary circuit accounted for 54% of PGE recoveries and the secondary circuit for 46%. Ninety five percent of feed mass ended up in the secondary rougher tailings, accounting for 32% of PGE recovery.

Test 3: This process showed better total recoveries than test 2 (2.55 g/t vs. 2.49 g/t). 83% of the feed mass ended in the primary rougher tailings, accounting for 35.5% of the total PGE recovery. 46.5% of total recovery was in the primary concentrate, with the rest (18%) distributed in order of significance between the primary cleaner, primary re-recleaner and primary recleaner tailings.

<u>H3 Testing:</u>

Test 1: Once again, similar to testing of H2, this test showed the best grade recoveries, from sample, at 1.97 g/t. The primary circuit accounted for recovery percentage of 31% of total and the secondary circuit for 69%, which differs significantly from test 1 conducted on H2. 83.33% of total feed mass ended in the secondary rougher tails and contained 49.4% of total PGE's recovered. Relatively high recoveries were obtained from the secondary cleaner tailings in this test (11.88%).

Test 2: the results of this test correlates with test 1, as the bulk of PGE recoveries was made in the secondary circuit: 83.5%, with the rest accounted for in the primary circuit (16.5%). Relatively high recovery was obtained from the primary recleaner (6.9%) and in the primary concentrate (8.5%). Recovery from the primary re-recleaner was 1.1%.
The secondary circuit shows best recoveries from the secondary rougher tailings (53.4%) and the secondary cleaner (19%), with the rest distributed in rank from the secondary recleaner, secondary concentrate and secondary re-recleaner.

Test 3: this primary circuit supplied better total recovery grades than test 2 at 1.82 g/t. The largest mass percentage of feed was once again pulled to the primary rougher tailings: 79.7%, while the PGE recovery was 59.3% of the total recovery. Recoveries from the other components in the circuit varied between 20.5% in the primary cleaner tailings and 10.4% in the primary concentrate to 1.4% in the re-recleaner tailings and 8.3% in the recleaner tailings.

H4 Testing:

H4 was only tested using test 1. Most of the PGE's were recovered in the primary concentrate (34.5%), with the secondary rougher tailings showing lower recovery of 17.4% of total. The distribution of the PGE's through the circuit was not really concentrated and varied between 4.1% and 11.6% at different stages in the circuit. The primary circuit accounted for ~45% and the secondary circuit for ~55% of total PGE recoveries.

These results are more in accordance to results obtained from testing of H3.
The total PGE grade calculated was significantly higher (2.63g/t) than the measured grade (2.41g/t) for the sample.

H5 Testing:

For this sample the test results painted the following picture: PGE grade calculated as 1.85g/t was close to measured grade of 1.82g/t. The bulk of recovery was from the secondary circuit at 64.5%, while the other 35.6% was recovered from the primary open circuit. The primary recleaner showed a surprisingly high recovery of 17.1%, while 12.5% was recovered from the primary concentrate.

The 82% of feed mass routed to the secondary rougher tailings showed PGE recovery of 34.4% of total and the rest of the PGE's were recovered from the secondary cleaner (14.4%), secondary concentrate (7.4%), secondary recleaner (4.8%) and the secondary re-recleaner (3.5%).

H6 Testing:

H6 saw the removal of the re-recleaner stages from the circuit and reported a PGE grade calculated at 2.3g/t, which is significantly higher than the measured 2.01g/t.

The secondary rougher tailings contained a high proportion of PGE's at 68.1% of total, while secondary concentrate accounted for 10.4% of total recovery. Distribution of PGE's through the circuit was concentrated as follows: 6.4% in primary concentrate, 1.9% in primary recleaner tailings, 11.7% in secondary cleaner tailings, and 1.6% in secondary recleaner tailings.

H7 Testing

Results of testing on this sample showed 62.4% PGE recovery from the secondary circuit, with 31.1% concentrated in the secondary rougher tailings. The primary circuit resulted in 24.8% recovery from the primary concentrate, with a total of 37.6% from this circuit.

The PGE grade calculated was once again higher than the measured grade: 2.52 g/t vs. 2.4 g/t.

Sample	Primary concentrate grade (g/t)	Secondary concentrate grade (g/t)	Combined grade (g/t)	Recovery as % of total PGM's
H2	238.7	164.4	224.57	45.5
H3	207.0	126.6	195.51	25.2
H4	153.5	261.1	165.40	41.8
H5	215.9	95.4	146.81	19.9
H6	43.5	29.2	33.39	16.8
H7	150.0	319.2	179.33	35.9

Table 7.5: Summary of Concentrate Grades vs. Recoveries for Test 1 Samples

This table illustrates that recoveries in the concentrates are not very good, ranging between 17% and 46%. It would be preferable for more PGE's to report to the concentrates. Different samples show different distribution of grades, with H2, H3, H5 and H6 reporting better grades in the primary concentrates and H4 and H6 favouring partitioning into the secondary concentrates.

One cause for concern is the fact that a substantial percentage of the total PGE's are lost in the secondary rougher tailings. Table 7.5 shows a comparative situation of percentage PGE's reporting to secondary rougher tailings vs. percentage in concentrates.

Sample	% PGM's in secondary rougher tails	% PGM's in concentrates
H2	31	45.5
H3	49.4	25.2
H4	17.4	41.8
H5	34.4	19.9
H6	68.1	16.8
H7	31.1	35.9

Table 7.6: Distribution of PGM's: In concentrates and tailings

7.3 Metallurgical Testing of the Bulk Sample

7.3.1 Characteristics of bulk sample

A 500 ton bulk sample was taken over a portion of the Crater ore body as discussed earlier under the heading 'GEOLOGY'. The sample was taken from the Main Reef at a depth between 40 and 45m. The Main Reef grade is between 2 – 3 g/t total precious metals (Pt, Pd and Au) and contains the UM and LM reefs (2 – 6g/t). Sampling location was shown in figure 4.4. Figure 7.4 indicates a schematic of the sampling location in relation to the proposed pit outline as determined during the open pit planning phase.

7.3.2 Testing of bulk sample

The 500 ton bulk sample was delivered to Mintek for pilot plant test work. Initially, a 270t sample was allocated for the flotation test work. The circuit configuration was alternated between a 1-stage mill float (MF1) and 2-stage mill-float (MF2) setup, with magnetic separation circuit following the primary mill float, the latter which is indicated in Figure 7.5.

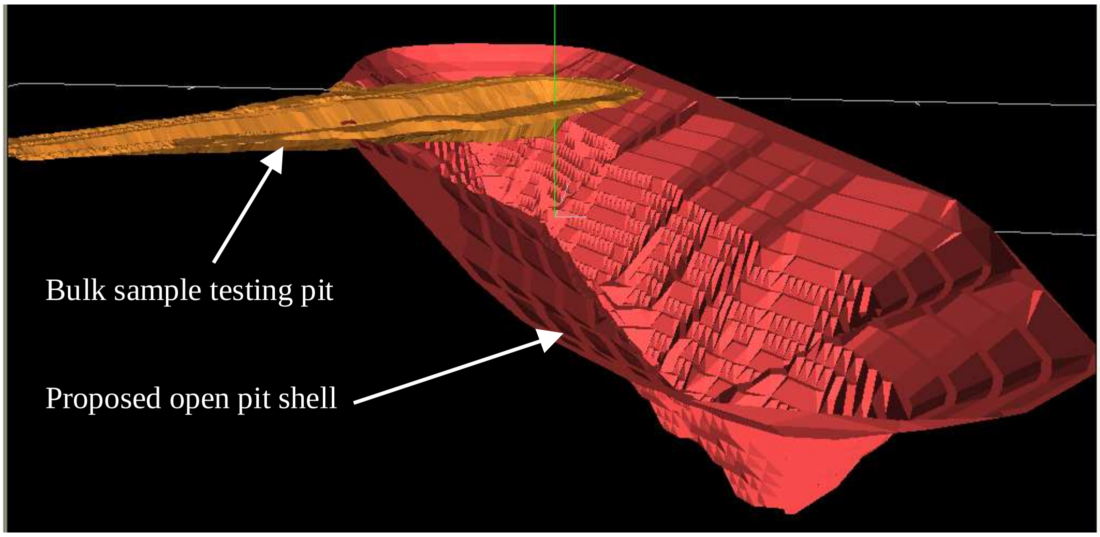


Figure 7.4: 3-D model of bulk sample location: indicates location of the bulk sample testing pit in relation to the planned open pit shell over the Crater ore body

The sample was crushed to –6mm and blended. The primary ball mill was fed at a rate of 400kg/h in a closed circuit to achieve the target grind of 80% -75 µm. The product from the primary mill was fed to a second primary ball mill and the product from here fed to a cyclone. The cyclone overflow was fed into the primary flotation circuit, while the underflow was fed back into the second primary mill.

The primary rougher tails were pumped to a magnetic separator before being fed to the secondary mill, followed by secondary flotation. The target of the secondary milling grind is 80% -38µm.

The pilot plant run was estimated at 4 days, in order to determine the normal flotation response of the ore. All streams sampled were analysed for 3PGE's, Cu, V_2O_5 and TiO_2.

7.3.3 Results of Bulk Sample Testing

The pilot plant design runs delivered fairly consistent results. Five external samples and one internal sample, from the circuit, were analysed in terms of recovery and grade. These results are displayed in Tables 7.7 and 7.8.

Samples	**3 PGE's Grade**	**3 PGE's Recovery**
External 1	83.3 g/t	73.1 %
External 2	104.4 g/t	73.1 %
External 3	88.9 g/t	73.1 g/t
External 4	89.2 g/t	73.8 g/t
External 5	108.4 g/t	72.9 g/t
Internal 1	60.3 g/t	73.0 g/t

Table 7.7: MF2 PGE Results – Final Concentrate: Better recoveries were obtained from the bulk sample testing, compared to the bench scale testing, as a result of finer grind of feed material.

The Pt:Pd ratio of the final concentrate was 1.2:1. Mass-pull of final concentrate was determined to be between 2.5 – 3%. For the purposes of the discounted cash flow model, a mass pull of 3% was used.

Copper recoveries were also tested and the results of these follow in Table 7.8

Samples	**Cu Grade**	**Cu Recovery**
External 1	1.31 %	74.0%
External 2	1.73 %	74.2%
External 3	1.22 %	69.5%
External 4	1.33 %	72.3%
External 5	1.74 %	72.4%
Internal 1	0.92 %	72.8%

Table 7.8: MF2 Cu Results – Final Concentrate

Recoveries from the MF1 and MF2 circuits showed very little difference. However, losses of fines from the MF1 did result in small recovery losses. Besides improved recoveries from the MF2 plant, the PGE concentrate grade also improved by 11% (from 59% to 70%). This was the result of the finer grind (80% -38μm vs. 70% -38μm for drill cores samples), which liberated more of the finer particles. For the purpose of the discounted cashflow model recoveries of 45% and 72% were used for oxide and sulphide material respectively.

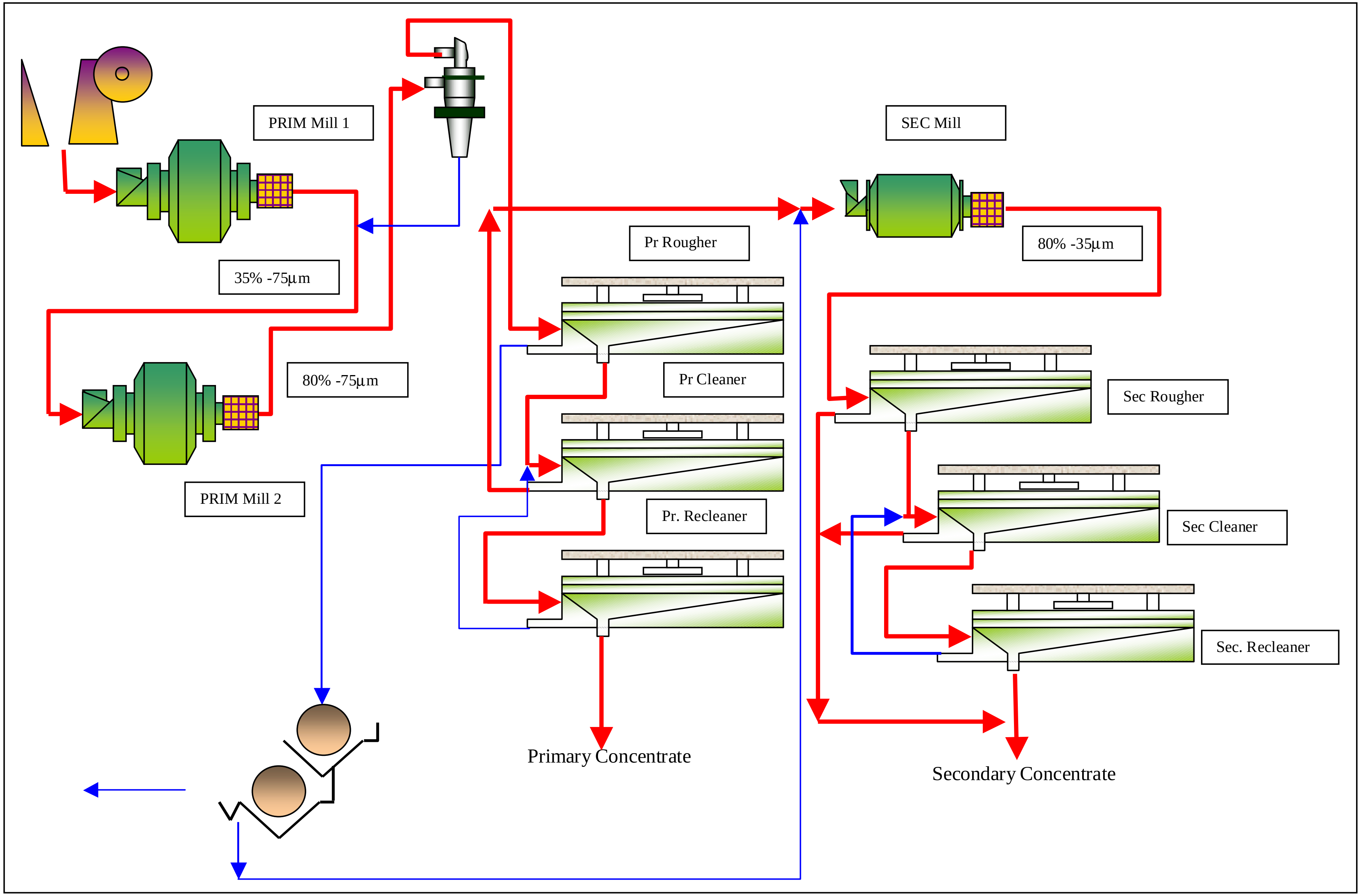


Figure 7.5: Process Flowsheet

7.3.4 Discussion of Bulk Sample Testing Results

The metallurgical test work done on the bulk sample, indicated that recoveries of approximately 70% were possible from a two stage mill-float circuit, when crushing to -38µm. In order to optimise liberation of mineral grains from rock, it is necessary to include the second stage in the circuit.

The distribution of Cu and PGE's between magnetic and non-magnetic fractions is even and can only be sufficiently separated at a grind of 80% -75 µm. Further findings indicated that fine-grained PGM mineralisation, associated with highly altered host rock, displayed lower recoveries of PGM's and a low concentrate grade. This may indicate lock-up of PGE's in the alteration assemblages. Further testing of these ore types will be necessary in order to optimize recoveries from altered mineral assemblages.

When using the average Main Reef grade of the Crater and Orion ore bodies (2.1 g/t Pt+Pd+Au), it must be kept in mind that the geological dilution (as a result of dykes and faults), mining dilution (10%) and mine call factor (95%), may reduce the feed grade to 1.5 g/t Pt+Pd+Au.

Figure 7.6: Mintek Testing Laboratory

7.4 Transport and materials handling

The material flow through the plant, was described in the flowsheet in Chapter 7 (Metallurgy). It constitutes conveyor and pipeline transport processes.

The heavy metal concentrate in the concentrator will be partially dried and then loaded into long-distance haulage carriers to be transported to a smelter in Rustenburg. The maximum metals concentration should be acquired in the concentrate in order to reduce transport cost per ounce of metal produced. Concentrate mass pull was determined at between 2.5 to 3% of feed. Also, the material should be as dry as can possibly be achieved given the time in the concentrator. Wet volumes and weights are more than the dry equivalents.

Services to be delivered to the plant will include that of water and electricity. Water will be carried in a pipeline from a dam approximately 25 km away from the site. Electricity will be carried in power lines from the substation.

The suppliers will transport all chemicals and spares necessary for operation of the plant, to the operation from Johannesburg. These supplies will be stored in the plant store/warehouse, which can be housed in the same building as the mining supplies warehouse. Mining supplies include safety equipment, as well as equipment as required for operations. Mining supplies will be transported from Rustenburg or Klerksdorp.

Bottlenecks could possibly develop in the crushing stage in the event that the delivery tempo of ore to plant is too high. A processing capacity of 150 000 tons per month, will relate to a capacity of 5000 tons per day, relating to ~200 tons per hour (24 hour processing). These volumes will dictate the mining tempo and number of haulage trucks required on the operation.

If 25-ton trucks are used in the operation, eight loads will have to be carried to the plant per hour. Depending on turnaround time from off-loading to loading to off-loading, the number of haulers can be determined. With a distance of 1 km between the plant and pit, a typical turnaround at start of operations can be expected to be approximately 20 minutes. The number of trucks required for ore transport will be three. This number will have to increase as the pit deepens later. Another three to four haulers will be sufficient to haul and dump waste next to the pit.

7.5 Plant Operating Costs

The metallurgical concentrating costs were obtained from applying the pilot plant operating costs to commercial plant specifications. The plant operating costs are summarized in Table 7.9 and are discussed further in Section 11.4.6

Total Plant operating cost R/ton	**35.70**
Labour	4.04
Reagents	5.50
Consumables and liners	5.02
Maintenance	5.77
Power	7.68
Assays	1.27
Transport	0.92
General and Admin	1.38
Management, grade control	4.13

Table 7.9: Plant Operating Costs

Chapter 8 FINANCIAL EVALUATION

8.1 Valuation method

The method adopted in valuation of this project is a fundamental valuation method. It is necessary to determine what the true current value of the project is. In order to do so, the nett present value (NPV) has to be determined. A discounted cash flow model (DCF) has been designed, which takes into account all variables related to the project.

This model projects expected future cash flows, which are then discounted to present value by applying a certain discount rate. The discount rate is dependent on risks associated with the project.

8.2 Criticism of the method

The DCF is a very useful visualisation tool, which highlights factors that influence the operation and resultant profitability and also attributes a bottom line current value to the project.

The greatest challenge to face when constructing such a model is the uncertainty of the future. Common factors, such as exchange rates, commodity market prices, inflation and tax regulations influence all mineral projects. Future projection of these factors is based on historical data, financial indicators at the time of design and on speculative predictions by financial analysts and industry.

It is often necessary to make uncertain assumptions regarding variables and this generally leads to these being overly conservative and subsequently penalizing the project profitability figure. A great number of variables have to be taken into account, which limits the confidence in the determination of present value somewhat. As confidence in information to be used increases (more drilling, new ore body models, etc.), so does confidence in the valuation.

As future values are discounted against a certain rate, the influence of the discount on later years is much more marked than on the early years' cash flows. It is therefore important to select a reasonable time period over which to valuate a project. Selection of the optimum period is a contentious issue, but one which is ultimately driven by the duration of the project. For mining projects this period will relate to the life of mine. After construction of the Kalplats discounted cash flow model over a life of mine of 10 years, it was noted that cash flows in years 9 and 10 were negative and the model was amended to consider only cash flows from years 1-8, in an attempt to improve the economics of the project.

Selection of a suitable discount ratio depends on risk factors influencing the project. These may include country risk, technology risk, economical and political factors, market risk and more. These are discussed further in Section 8.3.13.

8.3 Requirements for construction of a discounted cashflow model

It is important to note that the confidence in the results of a DCF is dictated by the confidence in the input parameters. Following are the input parameters used during design of a DCF for the Kalplats PGM Project.

8.3.1 Metals prices and product revenue

Market metals prices are dependent on the supply-demand chain. The metals market is volatile and prices fluctuate continuously.

The ruling metals price at time of sale directly determines the revenue generated from sales and it is therefore important to monitor these consistently. For the purpose of designing a DCF, certain projections need to be made regarding metals prices in future.

It is worthy to note the uses for the metals in order to gain understanding of how the demand for these metals may behave in future. These are considered here briefly:

i) Platinum

Main uses are in the jewellery industry and in catalytic converters for control of motor vehicle exhaust emissions. Other uses are in electronics (fibre optics, thermic couples, ceramics), chemicals (fertilizers, explosives, silicones, biodegradable detergents), glass (fiberglass, LCD's), petroleum (refining catalyst), medical (alloys, implants, anti-cancer drugs), spark plugs and fuel cells (catalysts).

The outlook for platinum prices into the future is very positive with the current supply deficit expected to rise over the next few years (Mining Journal, 21 May, 2004).

ii) Palladium

Palladium is mainly applied to the autocatalyst and electrical (multi-layer ceramic capacitors) industries. The demand for palladium is higher than that for platinum in autocatalysts. Other uses are in dentistry (alloys), jewellery (white gold when combined with gold) and chemical (fertilizers and acids).

The future outlook for Pd prices is less exciting with a serious oversupply anticipated by 2010 (2Moz). Reductions in palladium usage for electronics contribute to this problem. Most PGM projects with high Pd credits will struggle to be profitable.

Historical data for the prices of Pt, Pd and Au are displayed in Figures 8.1 – 8.6, as from the Kitco website (2005). These data have been used to predict metals prices to be used in the DCF.

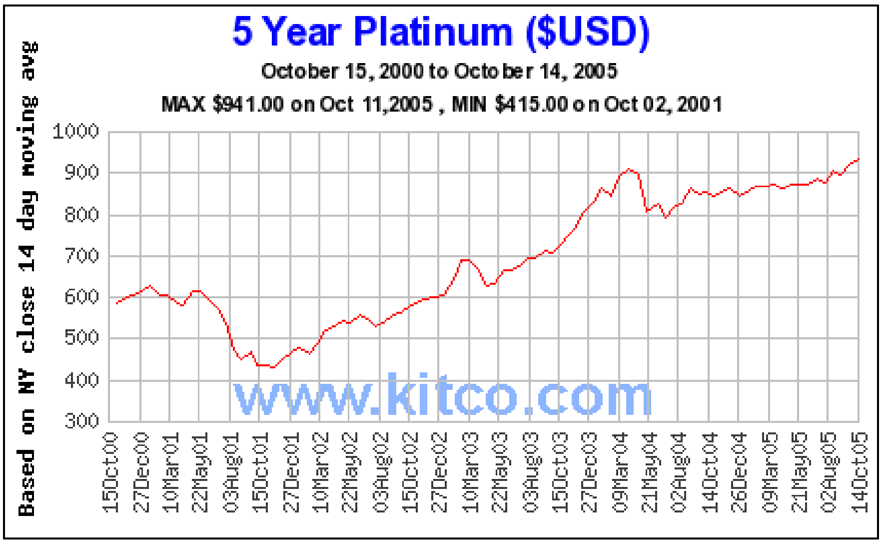


Figure 8.1: Pt PM Fix last 5 years

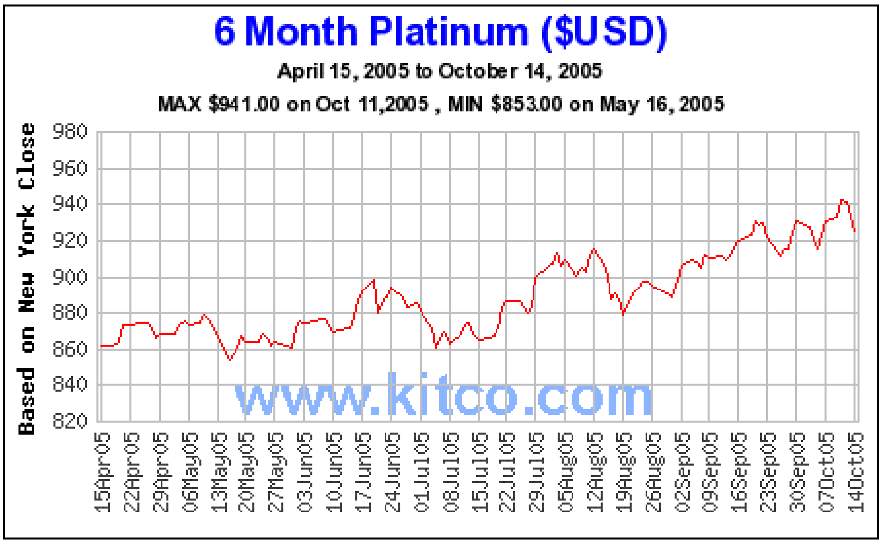


Figure 8.2: Pt London Fix last six months

For platinum a value of US$ 800/oz is used in construction of the DCF and sensitivity to platinum prices is tested at values of US$ 700/oz and US$ 1000/oz.

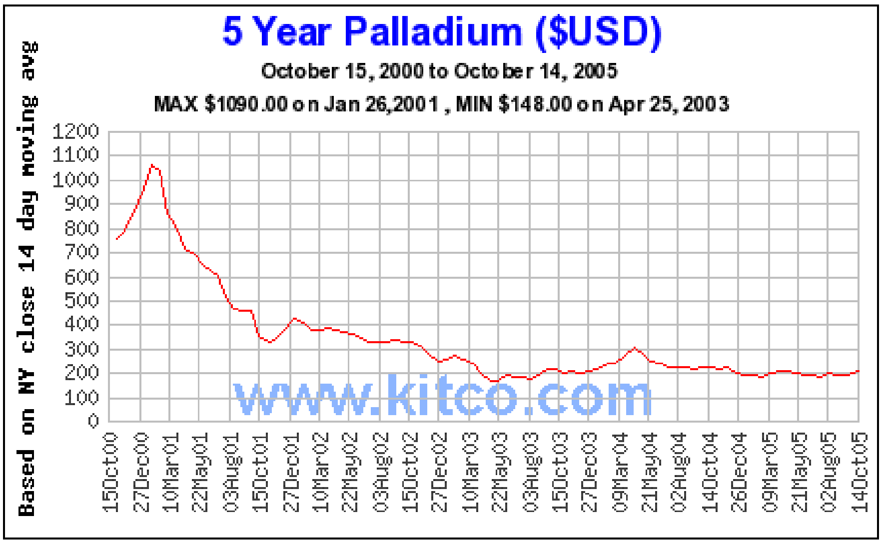


Figure 8.3: Pd PM Fix last five years

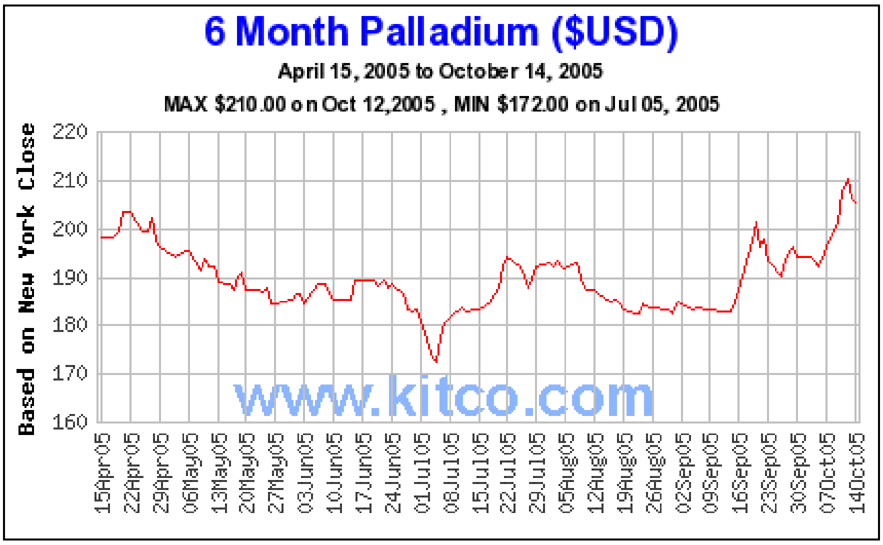


Figure 8.4: Pd London Fix last six months

For palladium, a value of US$ 200/oz is used in the DCF construction and sensitivities were tested at palladium prices of US$ 150/oz and US$ 250/oz.

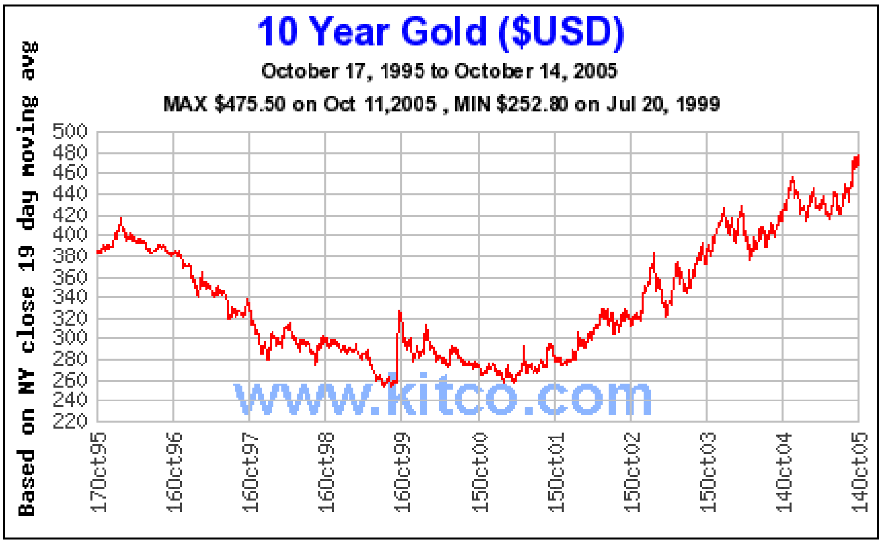


Figure 8.5: Au PM Fix last ten years

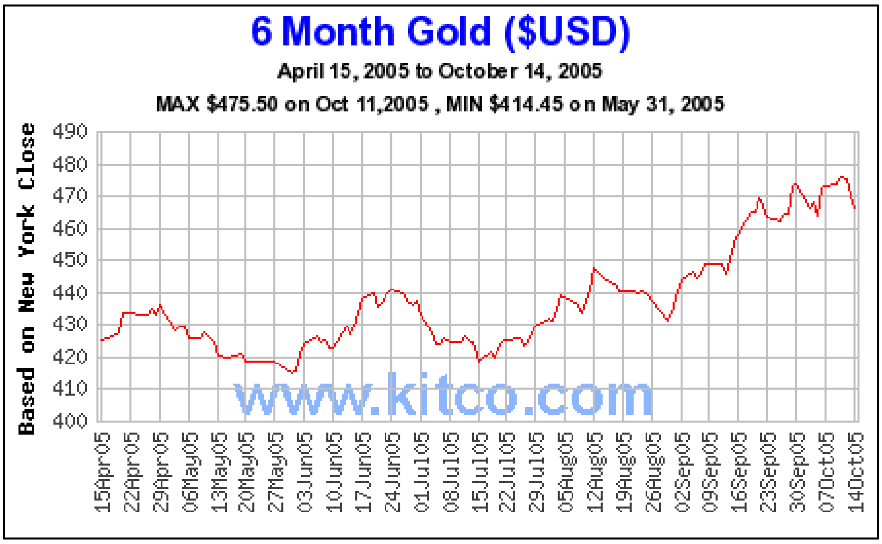


Figure 8.6: Au London Fix last six months

For gold, a price of US$400/oz has been used in the DCF construction, with sensitivities tested at US$ 380/oz and US$ 440/oz.

Platinum is mainly produced in South Africa and Russia, with minor production from the USA, Canada, Finland and Zimbabwe. The major producers of palladium are the Russian Federation and South Africa, with some production from Canada the US and Finland. Until very recently, Russia was responsible for two thirds of the world's annual production. The future of platinum and palladium prices is uncertain and high prices are threatened by new technologies that may promote the use of nanotechnology in especially autocatalytic converters (Bunkley, 2004).

8.3.2 Cost of mining

The costs related to mining have fixed and variable components. Fixed costs are those that are not related to production output, while variable costs are directly related to production activity.

Fixed costs will include administrative and management overheads and rentals if required. The fixed costs for the Kalplats project will consist mainly of salaries.

Variable costs will be derived from:

Pre-stripping
Drilling and blasting
Loading and hauling
Provision for closure and rehabilitation

Mining costs were discussed in Chapter 5 ‘Mining’. Provision was made for rehabilitation of the slimes, dumps and open pit at a rate of R0-50/milled ton.

A summary of the mining operating costs is listed in Table 8.1

Mine Operating Cost	R/ton
Dozing	3.63
Free dig	5.97
Oxide waste mining	8.25
Sulphide waste mining	12.1
Oxide ore mining	11.55
Sulphide ore mining	14.3
Rehabilitation	0.5

Table 8.1: Mining Costs:Ccosts were obtained from contractor mining cost at the Kalgold open pit gold mine

8.3.3 Cost of metallurgical concentration

Metallurgical plant costs will include all operational costs related to the metals concentration procedure. These are the costs of:

- Chemicals,
- Consumables,

- Maintenance,
- Materials handling,
- Water treatment,
- Electricity,
- Quality control.

Metallurgical costs are variable and depend on production rate as well as efficiency of operation in the plant. Maintenance is an important aspect to ensure that all processes function optimally.

The costs for metallurgical concentration, as discussed previously in Chapter 6 'Metallurgy', are listed below. Note that these costs only relate to the concentration of the metals and not to the final smelting procedure.

Plant operating cost R/ton	**35.70**
Labour	4.04
Reagents	5.50
Consumables and liners	5.02
Maintenance	5.77
Power	7.68
Assays	1.27
Transport	0.92
General and Admin	1.38
Management, grade control	4.13

Table 8.2: Plant Operating Costs: costs based on scoping study cost determinations

8.3.4 Metallurgical recoveries

The metallurgical recoveries from the concentration plant are paramount in determining revenue generated by ore processing. Different types of material have different recovery properties, e.g. oxide vs. sulphide mineralisation. Through metallurgical processes we attempt to maximize recovery of metals from the ore and the optimal route is chosen.

A balance between cost, time and quality has to be achieved and therefore it is necessary to optimize the procedure in order to obtain maximum profit at minimum cost.

The average recoveries, as discussed in Chapter 6 'Metallurgy', as used in the discounted cash flow model are as follows:

- Recovery from oxide material: 45%

- Recovery from sulphide material: 70%

8.3.5 Production rate

The selection of an optimum production rate is crucial during the planning stages of the project. This selection will influence not only the operating costs, but also the initial capital cost related to plant construction and equipment required.

There exists a number of rules of thumbs for selecting the optimum production rate, related not only to the physical characteristics of the deposit, but also to the economic characteristics of the entire project.

Examples are listed below:

a) Physical characteristics (Smith, 1997):

Taylor's Law: tons/day = 0.14 $(\text{Reserves})^{0.75}$

In the case of Kalplats this calculation would suggest a production rate of 5,100 t/d or ~ 1,8 Mt/annum.

b) Economic characteristics (Smith, 1997):

- The cash flow generated must be enough to repay capital expenditure twice.
- Annual production must exceed 100 000 oz (precious metals)
- Life of mine must be at least seven years
- Operating costs should fall within the lower quartile of the industry's cash cost curve. Table 8.3 presents a benchmark of industry costs.
- Life of mine should not be shorter than the cycle of the metal price

Company	**Annual Ounces PGE**	**Cash cost (US$/oz)**
North American Palladium	237,000	348
Aquarius Platinum	328,000	425
Norilsk Nickel	1,796,000	133
Implats	3,549,000	371
Angloplats	4,449,00	417

Table 8.3 Benchmark of industry costs: Average Cash Operating costs of PGM producers

The economic characteristics are presented in Section 8.5 'Discussion of results'.

When considering the mining of the Kalplats deposit, these economic characteristics are not satisfied (Section 8.5). Although operating costs are minimised over the life of mine, as a result of economies of scale, they still amount to $357/oz. Annual production does not exceed 100,000 ounces Pt+Pd+Au per annum and cash flow does not repay capital expenditure.

The mining of four ore bodies (Crater, Orion, Sirius and Vela) is considered in the DCF, which will contribute towards a life of mine of ten years.

8.3.6 Cost of smelting

The metal concentration product from the plant will be transported to a smelter, which is yet to be determined. It is therefore estimated that smelter costs will be payable at a rate of 20% of revenue generated from sale of the metals. Smelting costs impact severely on the profitability of the operation and it is suggested that negotiations with smelter operators are aimed at minimising smelting costs to 5% of revenues from sales or to pay a percentage fee of profit generated.

8.3.7 Cost of transport (on-site and to smelter)

The transport costs on site will consist of costs related to moving ore to the plant and waste to the dumps. This cost will be included in the contractor's fees and are variable. As mining deepens in the open pit, these costs increase accordingly.

Transport of the concentrate to the smelter is an additional cost. The aim of the concentrating plant is to minimize the mass pull to the concentrate, while maximizing metal content thereof. The mass pull used to calculate transport costs is 3% of mill feed.

Transport details	**R**
Transport/km/ton	0.7
Ave ore transport distance (km)	2.5
Transport distance to smelter (km)	300

Table 8.4: Transport Costs and distances

8.3.8 Cut-off grade

The cut-off grade for the purposes of this DCF is the grade above which, mined material, is classified as ore and will be processed. The cut-off grade can be determined as the average cost break-even grade, or at a level higher than break-even grade to maximise nett profit.

Break-even grade is the grade at which the revenue generated from sales of the product equals the costs associated with mining and processing the ore. The break-even grade for Kalplats can be calculated as follows:

The following is based on an assumption that average grade mined is 1g/t:

- Average working costs of 1 ton sulphide ore ~ R 110.
- Assuming a constant ratio of Pt:Pd of 1:1 and metals prices of US$800/oz Pt and US$200/oz Pd the average revenue per ounce of metal is US$500:

Revenue generated from 1 ton of ore at an average grade of 1g/t PGM (Pt + Pd):

500 * 1/32 = US$ 15.63

Converted to ZAR (R6-50 = US$ 1) = R101

- Estimated smelting cost (20% of revenues) per ton ore = R20
- Royalties estimated at 1% revenue = R1/t
- Estimated transport cost (3% concentrate) per ton ore = <R0.10

Thus total revenue after smelter fees and royalty = R101- R20- R1 – R0.1 ~R80

In order to calculate the breakeven grade, it is necessary to balance these figures:

At a grade of 1 g/t: Total Revenue = R80

At a grade of x g/t: Revenue = R 110

It follows that x (breakeven grade) = 1.38 g/t

Grade tonnage curves are useful in determining the ore tonnage available at or above the cut-off grade. A grade tonnage curve was not constructed for Kalplats, but this will be done should a full feasibility study commence. The cut-off grade used to determine reserves was 1g/t, which is lower than the break-even grade determined here. This is sure to influence the discounted cash flows negatively.

Future determinations of reserve volumes and grades should consider an increase in the cut-off grade to 1.5g/t. It must be noted that the cut-off grade of 1g/t was determined during early 2002, when the palladium price was $400/oz and Pt $500/oz, with a view that both of these would see an increase over time as a result of higher demands related to catalytic converter technology. This has, however, not proved to be the case, with only the Pt price increasing.

8.3.4 Capital expenditure

The capital expenditure consists of capital requirements for mining operations, including equipment, vehicles, roads, safety equipment and offices and capital for the metallurgical processing facility, which includes plant construction, equipment, spares, chemicals and consumables, well field establishment and electricity supply.

The capital requirements are dependent first and foremost on the production rate selected. The pilot plant study plays a vital role in the design and determination of capital requirements for plant construction. Capital expenditure for mining is significantly reduced when electing a contract miner to do mining. In this event, no purchasing of vehicles and equipment is necessary. The mining contractor's rates will include vehicle and equipment hire at a premium and these will be accounted for under operational expenditures.

The capital requirements for Kalplats are listed below:

MILL FEED REQUIRED (000 tons/yr)	**1,800**
Process Plant Capex	Rm
Direct Capital Costs	**200.00**
Civils	19.00
Construction	17.00
Structurals	22.00
Transport	5.00
Mechanicals	100.00
Piping	5.00
Electrical	17.00
Management and Design	14.00
Vendor Commissioning	1.00
Working Capital	**10.00**
Spares	6.00
Strategic spares	3.00
Consumables	1.00
Plant Infrastructure	**17.00**
Assay laboratory, buildings etc.	5.00
Tailings Dam	12.00
Powerline & Substation	**3.00**
Wellfield & pipeline	**10.00**
Mineral Surface Rights	**15.00**
Mining infrastructure (roads, tips)	**5.00**
Pre-strip Crater	**8.00**
Total	**268.00**

Table 8.5: Capital Requirements: consists mainly of plant and infrastructure related capital costs

8.3.10 Exchange rate

The exchange rate of the Rand vs. US$ is important as revenues are generated in dollars, while costs are payable in Rands. Once again it is difficult to make accurate predictions regarding the exchange and as we have noted over the past three years, these rates are highly variable and can react with some volatility. It is impossible to predict accurately when serious losses are to be expected.

The exchange rate has been projected constantly into the future at R6.50/US$1. This is in line with currently accepted indications for rand depreciation.

Another important aspect of the operations is that of fuel costs. With higher crude oil prices, transport costs will increase and negatively impact on the operations. the study is based however, on the earlier assumptions made for transport costs.

8.3.11 Royalties payable

Mining royalties are payable to the government on revenues received from sales of metals. The new minerals bill has not been approved and implemented yet and there is much debate regarding proposed royalties on mining operations. The royalty at present is 1% on sales of Au and Pt. This rate may increase to a proposed 2% in the near future and this possibility needs to be considered during DCF calculations. Sensitivity analyses were done on 2% and 3% royalties on revenues.

8.3.12 Tax payable

As this project is not a gold project, it will not be taxed according to the gold formula, as is normally the case with Harmony operations. Standard mining company tax is 29% and this will apply to the Kalplats project.

Capital expenditure is deductible for mining operations, with the result being that no tax will be payable until after the capital has been paid back in full.

8.3.13 Discount rate

Selection of discount rate depends on a number of risk factors. The higher the risks related to a project, the higher the discount rate used to discount future cashflows.

Two terms that will be used during valuation of the project are the NPV (nett present value) and the IRR (internal rate of return).

The IRR = i, when $\Sigma^{0 \text{ to } n}$ $CF_n/ (1+I)^n = 0$

Where C = capital

F = free cash flow

n = years (life of mine)

The NPV = $\Sigma^{0 \text{ to } n}$ $CF_n/ (1+I)^n$

Current South African bond rate is 7%. The project risk related to Kalplats is high and a project risk rate of 6% was selected. Thus, total discount rate as applied to the project is 13%. The nett present value was also calculated, applying discount rates of 5%, 10%, 12%, 14% and 16%.

8.3.14 Mine schedules (NPV Scheduler+ v. 3.1)

The ore body models were used for optimal open pit design, done with Mine Scheduler version 3.1. The software also calculates the optimum sequence of mining, utilizing the grade distribution and physical characteristics of the reefs.

Mining schedules determine the average feed grade to the plant and in turn the recovered grades, which has direct bearing on the revenues from sales of the products.

An important factor in open pit mining is that of the strip ratio. By using the mine scheduler, the volumes of waste mined, relative to the volume of ore mined can also be determined. This has direct bearing on mining costs per unit of time (a year in this case).

Mining schedules were determined based on a production rate of 150,000 tpm.

8.3.15 Local grade variations and recoveries

The ore deposits do not display homogeneously distributed grades. Grades vary not only in the different reefs, but also within the same reefs. A summary of average Pt+Pd+Au grade distribution in the different reefs was illustrated in Figure 4.6.

Recoveries differ for different material in the deposit, the distinction being most evident between the oxide and sulphide material. Typical recoveries for oxide material is in the order of 45%, while for sulphide material the recoveries vary between 68 to 74%.

For the purpose of the DCF, average recoveries of 45% and 72% were used for oxides and sulphides respectively. These estimates may be conservative, but conforms to the predetermined course of action during design of the DCF. Further work should focus on improvement of metals recoveries.

8.3.16 Pre-stripping costs capitalised or working costs

A large volume of material will have to be pre-stripped before mining can commence. The material to be stripped will consist mainly of overburden (soil and sand) and highly weathered bedrock (recoveries < 25%). Estimated depth of pre-strip will be to 15 m.

The costs of the pre-strip for the Crater ore body was included in capital expenditure, while for the other ore bodies, they were accounted for in operating costs.

With increase in depth of the open pit, the mining costs will increase as a result of longer hauling distances to waste piles and to the plant. An incremental increase of 5% per 20m was estimated.

8.4 Construction and results of the DCF model

The focus here will be on the final model as created for the purpose of discussion. Costs used are average costs as determined for mining or stripping of specific portions of the ore body, as pre-determined through operational scheduling, using the mining costs as indicated earlier.

The DCF is attached in Table 8.6

8.5 Discussion of results

It is evident from this model, when looking at the bottom line, that furthering the project is not an exciting prospect for the future.

At a 13% discount on expected future cashflows the nett present value of the project will be negative (-R206 million), meaning that mining of the deposit, according to current plans (grades and production rates), will not be profitable. The IRR of -14% indicates a loss on capital spent.

The factors which impact negatively on the NPV have been identified as:

- Cut-off grade: cut-off grade for ore tonnage calculations should be increased to at least 1.5g/t (higher than calculated break-even grade).
- Smelter costs: 20% of all revenues on sales are estimated to go towards payment. At smelter fees of 3%, the operation will break even.
- Government royalties: 92% of nett profits will go towards payment.
- Cash operating costs equates to 82% of revenues from sales.
- Rand-dollar exchange: R/ton from sales only R140, while total costs are in the order of R150/ton. Depreciation of the rand to R8=US$1, will make the operation marginally profitable.
- Metals prices: although the projected value of platinum is quite high, the price projections for palladium are low, resulting in an average US$/oz price of $500 (Pt:Pd = 1:1). An increase in palladium price to US$430/oz, will make the operation profitable.

These and other factors will be addressed, when calculating NPV sensitivities to changes in these factors. It is evident that these factors can act as triggers to indicate when mining of the Kalplats deposit becomes profitable.

A simple summary of the inputs to, and results of the DCF is provided here:

Capital Development		**± R 270m**
Metal Prices:	Platinum	US $800/oz
	Palladium	US $200/oz
	Gold	US $400/oz
R / US$ Exchange		6.50
Smelter Royalty		20 %
LOM:		8 years
Payback Period		not over LOM
IRR		-14%
NPV at 13 % Discount		- R 206m
Revenues	(LOM)	US $499/oz
Cash Cost	(LOM)	US $357/oz
Total Cost	(LOM)	US $476/oz

KALPLATS PGM PROJECT										
CASH FLOW FORECAST										
The model is based on the mining of the entire Main Reef package (including UM and LM), over a life of mine of 8 years										
	TOTALS	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Tons Milled (000)	**14,400**	-	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800
Average Head Grade Pt+Pd+Au	**2.09**	-	2.28	2.28	2.28	2.08	1.94	1.94	1.94	1.97
Total Waste Tons (000)	**67,548**	-	6,600	14,400	5,000	17,003	13,192	5,653	-	5,700
Strip Ratio	**5**	-	4	8	3	9	7	3	-	3
Recovered grade Pt+Pd+Au	**1.37**	-	1.39	1.56	1.56	1.18	1.33	1.33	1.33	1.26
Ounces Platinum Produced	**307,184**	-	40,964	45,917	45,917	33,395	35,626	35,626	35,626	34,114
Ounces Palladium Produced	**299,640**	-	36,727	41,167	41,167	32,308	37,605	37,605	37,605	35,456
Ounces Gold Produced	**25,840**	-	2,825	3,167	3,167	2,765	3,563	3,563	3,563	3,228
Ounces Pt+Pd+Au	**632,664**		80,516	90,252	90,252	68,468	76,793	76,793	76,793	72,798
Inputs:		Base price								
Pt Price		**800.00**	**800.00**	**800.00**	**800.00**	**800.00**	**800.00**	**800.00**	**800.00**	**800.00**
Pd Price		**200.00**	**200.00**	**200.00**	**200.00**	**200.00**	**200.00**	**200.00**	**200.00**	**200.00**
Au price		**400.00**	**400.00**	**400.00**	**400.00**	**400.00**	**400.00**	**400.00**	**400.00**	**400.00**
Capex	**268.00**									
Exchange Rate	**6.50**									
Smelter Royalty	**20%**									
Financial Evaluation (R millions)	**TOTALS (Rm)**	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Revenue	**2,054**	-	268	301	301	223	243	243	243	232
Smelter royalty	**411**	-	54	60	60	45	49	49	49	46
Project Revenue	**1,643**		**214**	**240**	**240**	**178**	**195**	**195**	**195**	**186**
Transport of concentrate to smelter	**65**	**-**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**
Working Costs	**1,469**	**-**	**169**	**265**	**151**	**245**	**251**	**159**	**91**	**137**
Working Profit/(loss)	**110**	**-**	37	-33	81	-75	-64	27	96	40
Total State Royalties	**13**	**-**	**1.78**	**1.99**	**1.99**	**1.46**	**1.57**	**1.57**	**1.57**	**1.50**
Profit(loss) after Royalties	**96**	**-**	**35**	**-35**	**79**	**-76**	**-65**	**26**	**94**	**39**
Tax Calculation										
Tot.Cap.Amount (Rm)		**-268**	**-**	**-**	**-5.3**	**-**	**-**	**-1.6**	**-5.7**	**-2.5**
Capital Tax Shield (Rm)		-268	-233	-268	-194	-270	-336	-311	-223	-187
Taxable Cashflow		-	-	-	-	-	-	-	-	-
Tax Rate %			0.29	0.29	0.29	0.29	0.29	0.29	0.29	0.29
Tax Payable	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Cashflow	**- 187**	**-268**	**35**	**-35**	**74**	**-76**	**-65**	**24**	**88**	**36**
Cumulative cashflow		**-268**	-233	-268	-194	-270	-336	-311	-223	-187
Discount rate applied		**0.00%**	**5.00%**	**8.00%**	**10.00%**	**13.00%**	**14.00%**	**16.00%**	**18.00%**	
NPV for LOM of 8 years cash flows		**R -186.81**	**-201**	**-205**	**-206**	**-206**	**-205**	**-205**	**-203**	
Internal rate of return (IRR) -8 years		**-14.07%**								
Payback (years)		**never**								

Table 8.6: Discounted Cash Flow Model of the Kalplats PGE Deposit

8.6 Sensitivities

A number of sensitivities were tested for the project, including metals prices, exchange rates, grades, recoveries, capital expenditure and smelter costs. All factors were adjusted with 10% and 20% positively and negatively, in order to construct a sensitivity chart as shown in figure 8.7.

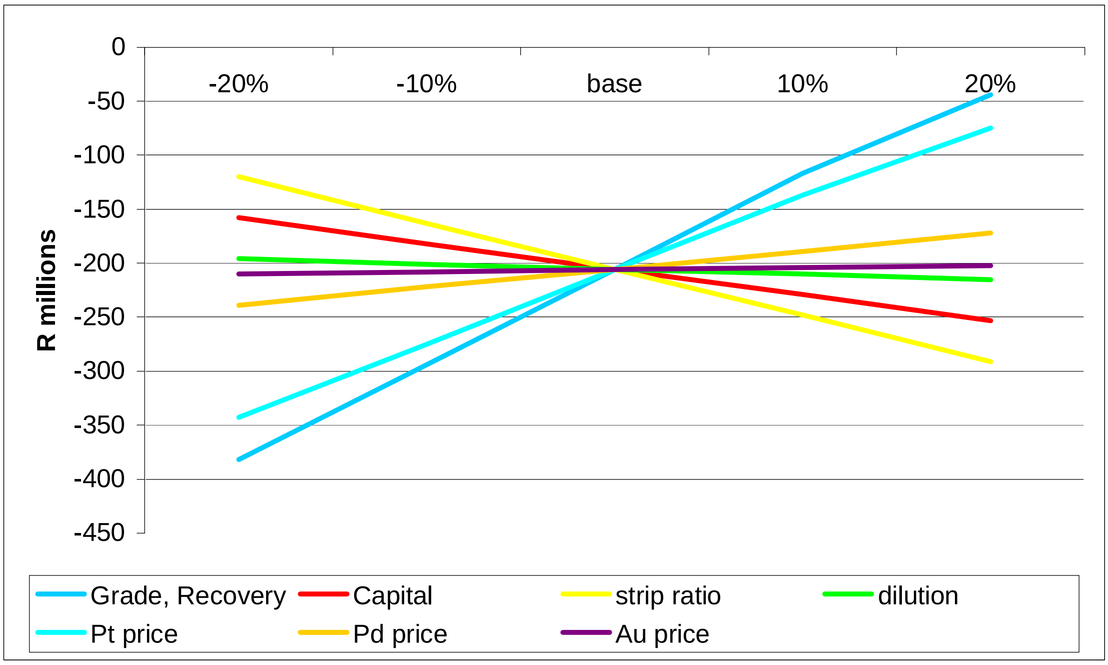


Figure 8.7 The influence of sensitivity factors on the Kalplats project NPV

The chart indicates clearly that the greatest sensitivities to the project economics are grade, recovery and Pt price. The sensitivity to the Pd price is also high, but this is not visible in this graph, as current Pd price is low and percentage variations do not markedly influence the nett present value.

Variations in working costs have less of an impact on the bottom-line figure, but are still a major influence. An increase in strip ratio can be detrimental to the project and care should be taken during planning to minimize the waste volumes to be removed.

In the event that payment of government royalties on platinum revenues can be excluded, the impact of this will be noteworthy, but not sufficient to ensure profitability of the operation.

As mentioned previously, it will be useful to be aware of trigger points that will indicate potential feasibility of the project. As such, a list of these factors and trigger points are indicated in Table

8.6, indicating the current value of the factor and the trigger value above or below which the project becomes feasible.

Factor	Current Value	Trigger value
Average Head Grade Pt+Pd+Au (g/t), higher than	2.09	2.71
Pd Price (US$/oz), higher than	200	480
Pt Price (US$/oz), higher than	800	1,070
Smelter Royalty on revenue (%), lower than	20	0
R:US$, higher than	6.5	8.21
Recoveries – sulphide (%), higher than	72	91
Capex (Rm), lower than	268	22

Table 8.7: Trigger values that may indicate feasibility of the project

The trigger factors should be monitored and future evaluations of the project should consider certain influencing factors, such as head grade, recoveries, smelter fees and capital expenditure.

In addition, as indicated earlier, specific sensitivities were tested to Pt, Pd and Au grades, as well as to royalties. The results are tabulated.

Factor	Value	NPV at 13% discount
Pt price	$700/oz	-R292 m
	$1000/oz	-R48 m
Pd Price	$150/oz	-R247 m
	$250/oz	-R164 m
Au price	$380/oz	-R207 m
	$440/oz	-R203 m
Royalties on all metals	2%	-R221 m
	3%	-R232 m

Table 8.8: Influence of change in selected factors on nett present value

8.7 Risk

Any new mining development is associated with a host of risks, from the external and internal environments. External risks typically involve commodity and exchange rate risks, as well as other macro-economic factors, while internal risks are directly project related. Here follows a summary of the typical risks to take into account when evaluating this project's viability.

Internal:

8.7.1 Grade

A major risk is the sustainability of high grade ore feed to the plant. In the event of an unexpected decrease in mining grade, the impact on revenues from sales will be large, as this will be a marginal operation. When considering the variability of contained metal grades in borehole samples over the Crater and Orion ore bodies, as indicated in Section 5.4, it is evident that grade is not homogeneously distributed through the reefs. Confidence limits for the estimated means are wide, indicating a large risk related to grade variations.

If we assume that only one standard deviation (38% confidence) is possible for the mean grades, which is an optimistic estimate, then considering grades in the Main Reef (Section 5.4) indicates that grade variations in Crater may be in the order of 39% for Pt and 50% for Pd grades and for Orion in the order of 45% for Pt and 48% for Pd. Risk related to grade variations is thus estimated at an average of 45%. This risk translates directly to a risk in project revenues.

8.7.2 Metallurgical recoveries

The metallurgical recoveries will have a similar effect on revenues as a reduction in feed grade. The recovery will rely on the efficiency of processes in the concentration plant. Efforts should be made to maximize recoveries, but the cost of these efforts should not impact negatively on the operational costs. When considering recoveries from the bulk sample, which vary between 72.9% and 73.8%, the risk of reduced recoveries is relatively low. These recoveries were, however, obtained from a limited number of samples and further work should focus on testing more samples and improving recoveries. The risk related to lower recoveries is thus estimated at 5%.

8.7.3 Ore body model

The accuracy of volume and ore resource calculations depends on the accuracy of the ore body model. As mentioned in Section 5.2, 3-D models or the ore bodies were constructed in such a way that non-reef intersections were sometimes included in reef packages. Besides the fact that this impacts negatively on the average grades of the reefs, it also alerts us to the fact that waste may have been included in reef tons. With an increase in information, the model can be refined and presented more accurately. An incorrect ore body model will influence all aspects of the mining operation, thereby influencing all subsequent processes, from mine design to eventual profitability estimates.

In an attempt to quantify the risk related to incorrect ore body modeling, we consider the following, where blocks as used in the model were not orientated along strike and block sizes were larger than the width of the MR1 and Mid Reefs, with average thickness of 2m. Thus, for

Crater, where a 4m x 4m x 10m block size was used, there could effectively be 50% dilution of grades and a 100% overstatement of tons. These should theoretically cancel out. However, this dilution of grade could lead to re-classification of an ore block as waste and could reduce the resource tons. As the two thin reefs are both high grade reefs, they would not have been re-classified as waste, but rather included into the Main or Mid Reef packages. The risk related to dilution of grade as a result of incorrect ore body modelling is estimated at 25% for the MR1 and LM reefs. The risk associated with the incorrect modeling of the other reefs is lower, as they have average widths of 7m and more. As we are considering only targeting the Main, UM and LM Reefs for mining, the overall risk is minimal.

8.7.4 Geological and Mining dilution

Dilution of grades as a result of the presence of narrow dykes or structures that may terminate mineralised reefs at depth is a risk at Kalplats. Drilling and the bulk sample have illustrated that these structures do exist and that the frequency and distribution of these vary considerably in different parts of the ore body.

The presence of more small-scale faults than have been identified thus far, is expected, as some only become apparent during excavations. Major faults and dykes cross cutting the reefs have been identified and mapped. Smaller faults with displacements of between 20cm and 5m are more difficult to identify. Were we to take into consideration that only small faults and dykes of this scale will not be identified prior to mining in the area where the fault occurs, the risk related to dilution by these structures is very low. As open pit mining methods will be used, grade control drilling will identify the presence of these structures timeously and classification of material will be done into ore and waste components.

Mining dilution is mainly a result of over-breaking and geotechnical problems. In areas where a high frequency of joints, or joint systems occur, material may collapse, then having to be excavated. Over-breaking may occur as a result of incorrect drilling and blasting patterns or incorrect ore body delineation. Mining dilution has been estimated at 5%, while the probability of occurrence is low. Safe mining practice should prevent falls of ground and slope failures.

8.7.5 Life Of Mine

The Life of Mine depends on production rates selected for mining and the size of the reserves. It is important to realize that a longer life of mine will negatively impact on the sustainable capital requirements and overhead costs. Personnel and administration costs will be higher over all, as will other fixed costs. The probability of a longer life of mine risk is very low and estimated to impact by 5% on overall capital required. Additional fixed and variable costs should be off-set by additional revenues generated.

It is also important to mention the effect of scheduling of mining. The preferred route would be to start mining and processing high-grade ore during the first months/years of the operation in order to maximise return on investment, capital payback and cash flows. This will however be a complicated process at Kalplats, because of the structurally complex nature of the ore bodies and also because of difficult visual distinction between mineralised reefs. Magnetic susceptibility tests on diamond drill core have indicated that there is not much difference between the magnetic properties of the UM, LM and Main Reefs and as such, the possibility for identification on that basis will be highly unlikely. The probability of this risk occurring is very high and as a result it is foreseen that the Main Reef package will be mined as a single unit. The impact of this risk on profitability is thus negligible, as only the risk related to grade variations through the Main Reef package will apply.

8.7.6 Capital expenditure

Capital requirements for the construction and initiation of plant and operations need to be determined accurately. This requirement will depend on the selections of mining methods, metallurgical concentration method, production rates and infrastructure requirements. All factors have to be taken into account and a contingency built into the figure. Confidence in the capital estimates as derived through scoping studies is more than 80%, which is sufficient for this study. Risk related to capital estimations is thus negligible.

Sustainable capital has to be addressed here too, which will entail replacement of equipment, costs to maintenance of the plant (balls and liners to mills etc.) and potential future expansion. As costs will increase over the life of mine, it is suggested that a 10% escalation per year is considered. The probability of this occurring is very high and as such a 10% impact on sustainable capital is estimated. Sustainable capital is included in operating cost and estimated at 5% of operating cost. Overall impact of this risk on cost will then be in the order of 0.05% and regarded as negligible.

Capital requirements for this project were based on recommendations and quotes from specialists in the respective fields. Influencing factors, such as the R/$ exchange rate should be taken into account, as the capital value may differ dramatically with fluctuations.

8.7.7 Technology

Improvements in technology may have a positive influence on capital required and on operating costs, as certain processes may be improved and costs lowered accordingly. However, the flipside is that improvements may be more costly.

The use of new technology, which hasn't been sufficiently proven to be successful, carries a large risk. Confidence should exist in the method and it should have been proven over long time periods to be the most cost- or time efficient procedure to follow. The selection of methods to be employed has to be substantiated by test work and comparative studies.

New, related technology may well be developed during the lifespan of an operation, but the costs of conversion and confidence in new equipment or procedures may not justify a conversion. At present the proven technology to be used in the operation poses a negligible risk threat.

External:

8.7.8 Rand-dollar Exchange Rate

The exchange rate has a direct influence on the profitability of the operation, as revenues from sales will be received as a dollar-price. Costs, however, are Rand-based. It should be noted here that with an increase in Rand-strength, the actual revenues received in rands are lower, while the costs still remain at the same levels, if not even increasing. This makes the profit margin of these operations smaller. Economist opinions on the future of the R:US$ exchange rate, received from RMB, Citibank, RBC, Nedbank, BMONB and Investec (all through personal communications), are that the Rand may strengthen somehat agains the dollar in the short term, but that long-term projections are of R:US$ of 6.5 to 7.

The impact of this risk on revenues received when considering the difference in future exchange rate projections, is less than 1%. As can be seen from Figure 8.8, the exchange rate has stabilized around the R6-50 level since 2003 and the trend is expected to continue.

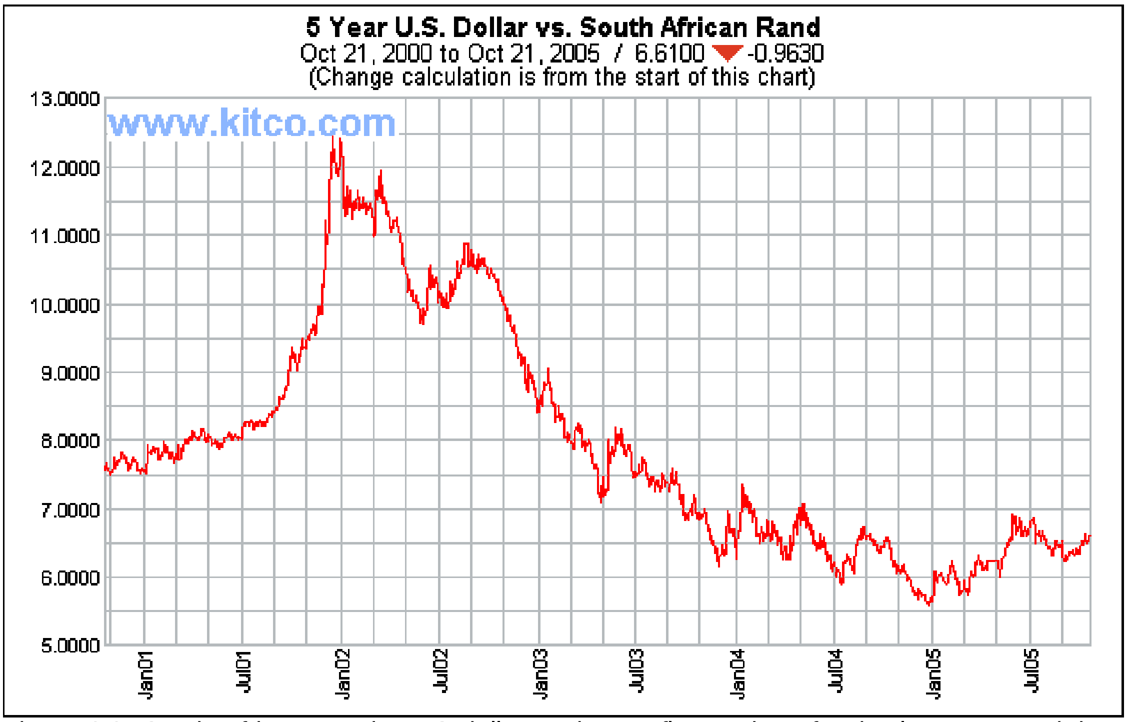


Figure 8.8: South African Rand – US dollar exchange fluctutations for the last 5 years (Kitco, 2005)

8.7.9 Metals market

Metals price fluctuations are not controllable and do not depend on operational variables. The market prices are driven by demand and supply. Revenues obtained from sales of metals, ultimately rely on market price of metals on day of sale. When considering the historical five year graphs indicated in Section 9.3.1, it becomes evident that fluctuations can be extreme over longer periods.

The indication is that the palladium price has stabilized around the US$200/oz mark, since 2002 and no indication is given that this will change dramatically in the near future. We did see record palladium prices of over US$1000/oz in 2000, but the trend has been downward and stable since then. Platinum, on the other hand, has seen a dramatic increase of 50% in value over the past five years. We may well be experiencing the apex of value and the risk exists that Pt value may reduce to prices obtained during 2000. Similarly, gold has reached a 17-year high and value is seen to be stable and possibly increased for another year, before beginning the down-turn.

The overall probability of this risk is high and impact on profitability mostly related to fluctuations in the Pt price, as the largest percentage of revenues (80%) will be generated through sale of Pt. The impact on profitability is thus estimated at 40% (50% reduction of 80% revenues).

8.7.10 Social Environment

To date, the community has been supportive of the exploration efforts at Kalplats, as it has brought promise of new developments to the region and together with that, promise of employment and education. There are always risks associated with safety and security at mining operations and the management of social plans and initiatives play a vital role in preventing and eliminating crimes. Good relations with local communities are pivotal in the successful management of the operation. The probability of this risk occurring is low, but should community activity hamper operations for even 1 month, the impact on revenue generated will be 8%. This situation is, however, not anticipated and the impact of this risk is negligible.

Should a unionized workforce be active at the operation, the risks associated to union demands and rights and strikes that may impact on operations also come into play.

8.7.11 Legislative Environment

Legal compliancy with respect to environmental laws, the Mining Charter, employment equity acts, black economic empowerment initiatives and government royalties on sales of products may pose certain risks. The company complies with these laws or aims to be compliant where that is not yet the case. The risk here relates to not being issued with mining permits and operations being penalized for non-compliance to regulations. At present this risk is negligible. Should royalties related to mining of the Kalplats deposit be subjected to stipulations in the Mining Act, there will be royalties payable of 3% on revenues from sale of metals. At present royalties of 1% on gold and 0.8% on Pt are considered. An increase in these, plus additional royalties on Pd, will impact total revenue by in the order of 2.5% (0.22% additional Pt sales royalty on 80% revenues plus 3% Pd sales royalty on approximately 20% of revenues).

8.7.12 Summary of Risk

The risks highlighted above need to be monitored and early warning signs identified, which can indicate the appearance of every risk or increase in risk factor. The following two graphs represent the impact and probability of risks and a risk profile related to the project. Probability was assigned on a scale of 0 – 10, with 0 being the lowest probability and 10 the highest.

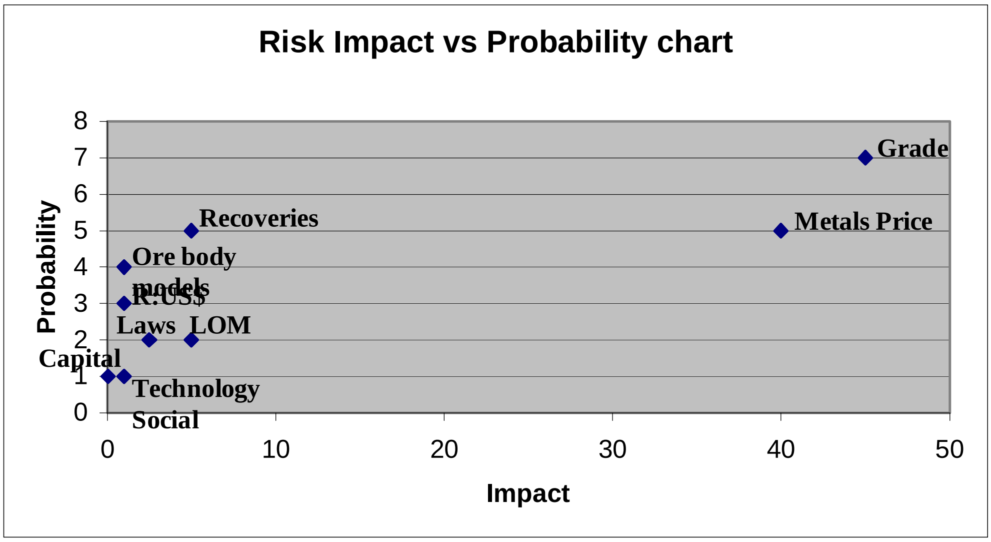


Figure 8.9: Risk vs Probability of occurrence: Probability was assigned on a scale of 0 – 10, with 0 being the lowest probability and 10 the highest.

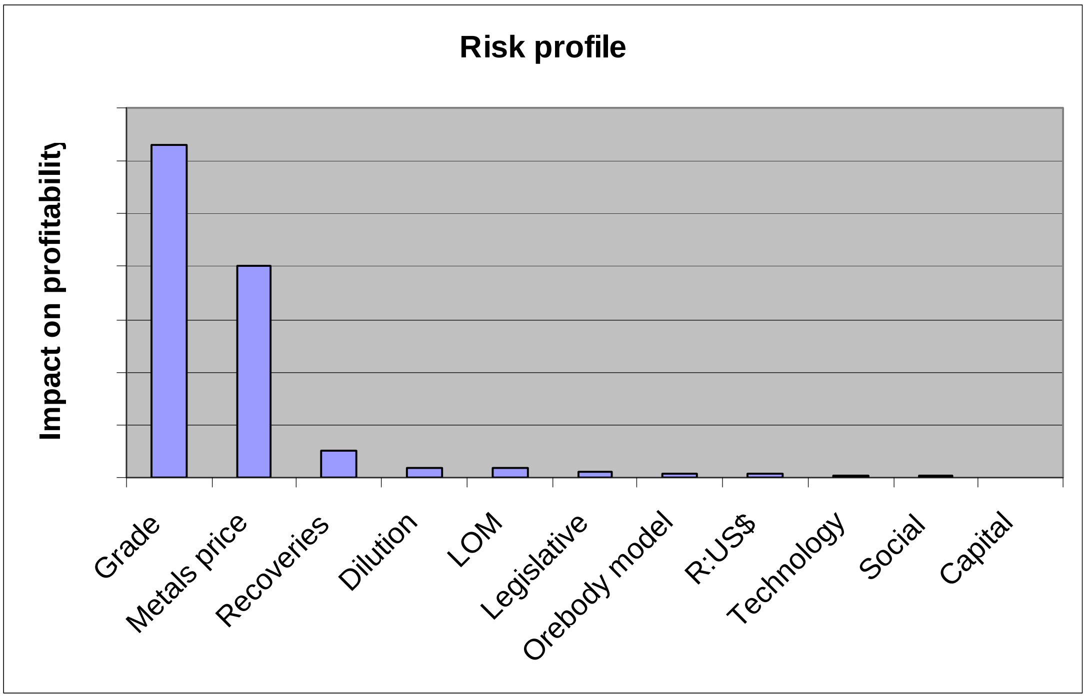


Figure 8.10: Risk Profile: Impact of risk multiplied by the probability of occurrence to indicate highest risks related to the project

This figure can be compared to the sensitivity graph as indicated in Fig 8.7. It is noticeable that a number of correlations exist between the two graphs, especially with regards to the importance of grade, metals prices and recoveries.

Chapter 9 RESULTS AND SUMMARY

The study has shown that mining of the Kalplats deposit is not viable in the current economic environment. This chapter serves as a summary of the dissertation.

9.1 Important aspects related to mining of the Kalplats PGE deposit

A number of factors were highlighted that may act as trigger points to indicate the potential of the project becoming viable to operate. The study also highlighted a number of concerns, which need to be addressed in future studies of the deposit. These include:

a) Cut-off grade determination: at current cut-off grades, the deposit is not mineable. The reserve should be estimated at higher cut-off grades
b) Re-engineering of production rate to optimally exploit the ore body: should cut-off grades increase, the associated ore tonnages will decrease and production rates will have to be reviewed in order to establish best rate to maximize profitability of the operation.
c) Improvement of metallurgical recoveries from oxide and sulphide ores: an increase in sulphide recoveries from 72% to 91% will make the project viable. It is suggested that more effort is invested in metallurgical design, in order to maximize recoveries.
d) Smelter fees payable: smelter fees should be negotiated to a minimum and an attempt should be made to base the fee on profits obtained from the operation.
e) Concentrate mass pull: will influence transport costs and should thus be minimised
f) Government royalties on production of metals: an attempt should be made to ensure the stability agreement regarding royalties remains intact, instead of being subjected to new royalties on metals sales of 3%, from 2009 onwards.
g) Supply of water: all options related to water supply should be investigated, including establishment of dams and pipelines from nearby dams.
h) Capital reduction options: capital expenditure requirements should be analysed and reductions attempted.

Factors, which cannot be controlled, include:

- Metals prices
- Rand-US dollar exchange rate

Precious metals markets are volatile and unpredictable and indications exist that the demand for these products will more likely reduce than increase. The sensitivity to grade and metallurgical recovery still places pressure on the project, even if metals prices increase dramatically and a marginal drop-off in either of these will render the project unviable.

9.2 Summary of information

9.2.1 Geology

The Kalplats ore bodies are hosted in a layered mafic intrusive package. Reefs are discernable, based on metals ratios. Visual differentiation is not possible in all cases. Reefs show broadly homogeneous metals grades. Displacement along fault planes is common and major faults, generally striking NE-SW, divides ore bodies into geozones. Quartz-albite veins have intruded the layered intrusion in various directions.

9.2.2 Mineral Resources

Metals contents (platinum, palladium and gold) vary through different reefs. Reefs have been divided into the LG, Mid, MR, UM, Main and LM reefs. The highest grades of metals occur in the UM and LM reefs, which bound the Main Reef. Mineral resources were calculated using a combination of manual and computer-aided methods. The total resource as calculated is 3,920,000 ounces of total precious metals (Pt+Pd+Au), contained in 84.8 Mt, while the Main Reef package hosts 2,150,000 of these ounces, contained in 35.4 Mt.

9.2.3 Mining

The open-pit mining method is most suited to extraction of ore, while there is a possibility of later underground development, when the open pit reaches its optimal cost effective depth, from the bottom of the pit. 3-D ore body models were constructed from drilling data and these models were used in mine design and optimization and scheduling. Infrastructure as required for mining purposes was determined and sources of services investigated. Mineable reserve is estimated at 1.7 Moz Pt+Pd+Au, contained in 26 Mt ore.

9.2.4 Metallurgy

Mineralogical studies of ore types determined the major ore-bearing minerals and distribution, while metallurgical testing of drill samples and the bulk sample were used to design the optimal process flowsheet to be used for designing the metallurgical plant. Testing indicated that a two-stage mill-float circuit, with magnetic separator (vanadium and titanium separation) is most effective at liberating maximum metals into a concentrate. Sulphide recoveries of 72% PGE's and Cu are possible.

9.2.5 Financial Evaluation

The method used for financial valuation of the project was that of the discounted cashflow model, calculating nett present value (NPV) and internal rates of return (IRR). Costs of capital and operations were derived from specialist recommendations and similar current operations. For an operation with a production rate of 150 000 tons per month, the current project economics (June 2004), indicate a NPV of –R206 million and IRR of -14%, using a discount rate of 13%.

Risks related to the operation have been identified as grade, metallurgical recoveries, metals prices, ore body models, geological and mining dilution, life-of-mine, capital expenditure, technology, rand-dollar exchange, social environment and legislative environment. The highest impact risks are grade and metals prices.

9.3 Limitations

Limitations related to the study include those of not being able to accurately predict the future trends of the metals markets and rand-dollar exchange rates. Furthermore, assumptions are made that two other ore bodies, except for Crater and Orion will be mine-able, based on limited data as support.

As no operational results are available, metallurgical recoveries and process are based on test work as done. Further refinement of these may be necessary to fully evaluate this type of operation. Other limitations were listed above in Section 9.1.

9.4 Future Research

Future research will be conducted by ARM Platinum and it is suggested that such work concentrate on the factors as outlined in Section 9.1.

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