AN EVALUATION OF SMALL SCALE OPEN CAST MINING OF UG2 IN THE BUSHVELD COMPLEX

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"At present it appears rather doubtful whether they will ever be worked for their platinum content"

P A Wagner, on the chromitite deposits of the Bushveld Complex, 1929.

Abstract

The current weak state of the platinum market, as well as the large inventory of platinum group metals held by Russia, necessitates that the South African platinum mining industry must carefully evaluate the benefits and disadvantage of small-scale mining of shallow, open castable ore resources.

Until the late 1980's, these resources were ignored due to the metallurgical complexities of treating oxidized ore, as well as the mindset that existed within the South African mining industry which militated against open cast mining in the Bushveld. During the latter part of the 1980's and early 1990's, advances in the metallurgical treatment of oxidized ore, specifically the UG2, as well as operational problems, created the impetus to begin the exploitation of these resources.

Small-scale open cast mining has become viable due to the development of suitable mining methods that facilitate mining practices acceptable to the Department of Mineral and Energy Affairs, in terms of environmental legislation.

Metallurgical advances and growing experience, especially with respect to the UG2, enables reasonable platinum group metals recovery from oxidized ore. The problems experienced in doing this can, and are being overcome. With growing public awareness of environmental issues, particularly related to the mining industry, the requirement to ensure that the small-scale open cast mine site is well managed is paramount.

A methodology for the evaluation is presented along with a case study of a small-scale UG2 open cast pit. Evidence is presented that shows that these small-scale open cast mining operations are extremely profitable and require minimal capital expenditure. However, caution is advised when evaluating Merensky Reef open cast operations because of the problems that they present.

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GLOSSARY

Amplats	Anglo-American Platinum Corporation
BC	Bushveld Complex
BHP	Broken Hill Properties
DWAF	Department of Water Affairs and Forestry
EMPR	Environmental Management Programme Report
EPL	Eastern Platinum Limited
g/t	grams per ton (parts per million)
Impala	Impala Platinum Limited
LGS	Lebowa Granite Suite
LPD	Lonrho Platinum Division
MLCC	Multi-Layered Ceramic Capacitor
Moz	Million ounces
NNC	Noril'sk Nickel Combine
oz	ounces
Mt	Million tons
PGM's	Platinum Group Metals
PPRust	Potgietersrus Platinum Limited
Matte Valuation	Net realization (Rand) per kilogram of metal in converter matte
RC	Reverse Circulation (drilling)
RGS	Rashoop Granophyre Suite
RLS	Rustenburg Layered Suite
ROM	Run-of-mine
RPM	Rustenburg Platinum Mines
RC	Rougher concentrate
RT	Rougher Tailings
TDS	Total dissolved solids
UG2	Upper Group 2 Chromitite Layer
t	Tons
WPL	Western Platinum Limited

Chapter 1

Chapter 1

Platinum Group Metals & the Bushveld Complex

1.0 Introduction:

The aim of this dissertation is to present small-scale open cast mining of the UG2 as a viable option, both to existing mines and to new mines in the western Bushveld Complex. The first chapter examines the state of the platinum group metals market and the changes that have taken place between 1993 and 1996 in supply and demand. The future of the market is examined briefly to set the scene for the role that small-scale open cast mining of the UG2 can play. The geology of the western Bushveld Complex is reviewed, with special reference to the UG2.

In chapter two the history of this kind of small-scale mining is outlined and the mining methods and economics of open cast mining of the UG2 is examined. Before considering a case study of a specific small UG2 open cast operation in chapter five, a number of pertinent issues, of great importance to any open cast project are discussed in chapters three and four.

Chapter three looks at the specific metallurgical problems that need to be addressed in treating shallow, oxidized UG2 ore, and the implications that they have on recovery of PGM's and the performance of the concentrator and smelting operations. In chapter four the environmental considerations of open cast mining are discussed, with reference to the legal requirements and the Environmental Management Programme Report.

Chapter five presents a detailed, financial and operational case study of the evaluation of a small-scale UG2 open cast project. It details the methodology of the project from a pre-feasibility desktop study, through to production and examines the actual performance of the project against that predicted by the feasibility study. Conclusions are presented in chapter six, along with a discussion of the future potential for this type of small-scale open cast mining, for both the UG2 and the Merensky Reef, in the western Bushveld Complex.

1.1 Platinum Group Metals

The Platinum Group Metals (PGM's) are a unique group of related elements comprised of platinum (Pt), palladium (Pd), rhodium (Rh), osmium (Os), iridium (Ir) and ruthenium (Ru). They share several properties that make them valuable both as industrial metals and as precious metals. Pt, Pd and Rh all possess important catalytic properties, as well as electrical properties that create an important industrial demand in addition to demand as precious metals. The closely related Os, Ir and Ru have similar properties, and are used as hardeners for Pt, Pd and Rh, although only Ru acts as a catalyst. The standard metre is composed of a Pt-Ir alloy in the ratio of 90% Pt to 10% Ir. Os is the heaviest known element in the periodic table.

The PGM's are produced both as primary products, mainly from layered intrusions and placer deposits, and as secondary products, mainly as a by-product of copper-nickel and gold deposits. Vermaak (1995) estimates that the world resource of PGM's is approximately 112 735 tons of contained metal. However, this estimate only considers resources down to a maximum depth of 1 200m below surface. As several of the major South African producers are already beginning to produce from depths below 1 000m this estimate is considered to be somewhat conservative. A slightly older report (Minerals Bureau, 1992) estimates world resources of PGM's to be 67 041 t, down to a depth of 2 700. This estimate also includes gold in addition to the PGM's. Most of the South African production comes from one of the Bushveld Complex's three platiniferous "reefs"; the Merensky Reef, the UG2 Chromitite and the Platreef. Vermaak (*op. cit.*) estimates that the Bushveld Complex contains some 53% of world's PGM resources. The Minerals Bureau (1992) estimates that the Bushveld Complex contains 87.9% of world resources. Whichever estimate is accepted, it is clear the Bushveld Complex of South Africa is certainly the largest known repository of PGM's in the world.

South Africa is the largest producer of PGM's in the world, followed by Russia and then North America. The three largest South African producers are, in order of size, Amplats, Impala and Lonrho. Some minor production also comes from Northam in the Bushveld Complex, the gold mines of the Witwatersrand, and from Palabora Copper Mine. The Russian production comes primarily from the Noril'sk Nickel Combine (NNC) with minor primary production from placer deposits in the Urals. The North American production comes from both primary sources, such as Stillwater in the USA and the North American Palladium Mine in Canada, as well as from secondary producers such as the Sudbury and Raglan nickel mines. Other minor production comes from Columbia (placer deposits), Australia (nickel deposits), Finland (nickel deposits), Baltic States (copper deposits) and, increasingly important, Zimbabwe (layered intrusion).

1.2 PGM Supply

1.2.1 Relative Production of Pt, Pd & Rh

Of the six PGM's, Pt, Pd and Rh are economically the most important and this section will concentrate on the factors which affect the supply and demand of these three elements. In this section the supply and demand of PGM's over a four year period, 1993 to 1996, will be examined and related in a later section to price. It must be borne in mind that it is virtually impossible to vary the production rates of the PGM's individually. The primary PGM deposits are polymetallic. An increase in the Pt production will, *de facto*, cause an increase in the Pd and Rh production. The ratio of the various PGM's, in the Merensky Reef of the western Bushveld Complex for example, for Pt:Pd:Rh is 16.42:8.45:1 and for the UG2 the ratio is 6.57:3.0:1 (Vermaak, 1995). The Merensky revenue/kg of metal produced is very sensitive to the base metal prices (Cu & Ni). This, combined with the fact that the larger producers in South Africa (Amplats and Impala) are more and more exploiting their relatively untouched UG2 reserves following a move taken by the Lonrho Group in the early 1980's, as well as for strategic reasons, directly affects the relative proportions of the various PGM's produced. Stillwater, in the USA, is a primary Pd producer, the ratio of Pt:Pd:Rh being 16.16:58.25:1 (Vermaak, 1995). Placer deposits, on the other hand tend to consist almost entirely of resistate Pt and a little Rh, as Pd is more mobile in the secondary environment. The tenor of PGM's in Ni deposits tend to be highly variable and, as the PGM's are produced as by-products, their production tends to follow the vagaries of base metal prices and production.

1.3 Platinum Supply and Demand

The information contained in this section is a précis of data gathered from several sources. Unless otherwise stated, the sources are Johnson Matthey Annual and Interim Reviews, Minerals Bureau of South Africa Annual Reports and various stockbroking reports and reviews. All sources are listed in the Bibliography and References at the end of this work.

During the period under review, Pt supply has risen steadily, from 4.39 Moz in 1993 to 4.53 Moz in 1994 and up again to 4.98 Moz in 1995. In 1996 Pt supply is expected to drop off slightly to 4.77 Moz. Of the 4.39 Moz produced in 1993, 3.36 Moz were produced by the three main South African producers, Amplats, Impala and Lonrho. Russia produced 680 000 oz; North America 200 000 oz and a further 120 000 oz were produced by various other producers (Table 1.1).

Pt demand is governed by demand in eight main sectors: autocatalysts, chemical, electrical, glass, investment, jewellery, petroleum and others. A ninth, and slightly artificial sector, is termed "Western sales to China". If there is an excess in supply over demand, the excess goes into "stocks" (in Western countries, not Russia). Similarly, if there is a shortfall, it is met from stocks (presuming that stocks are sufficiently large.).

In 1993, autocatalysts accounted for 1.685 Moz of Pt, of which 275 000 oz was sourced from recycling and recovery of Pt from scrap. Use of Pt in the automobile industry is governed by the fortunes of mainly western economies, as well as clean air legislation in those countries.

Pt demand in the jewellery sector, especially in Japan, continued to grow, as it has since the early 1980's and accounted for 1.615 Moz in 1993. Investment, again mainly in Japan, accounted for 305 000 oz in 1993, up from 255 000 oz in 1992. Most other sectors either remained static or showed a small change in requirements for this period (Table 1.1). Overall demand in 1993 was 4.025 Moz, with a further 20 000 oz going to China. Against a supply figure of 4.39 Moz, this meant that 345 000 oz went into stockpiles.

In 1994 there was a dramatic change in the supply figures with South African production dropping to 3.16 Moz, Russia's sales increasing to 1.01 Moz, North America's production remained static at 220 000 oz and production from other producers increasing to 140 000 oz. The enormous increase of Russian sales reflects a large draw-down on strategic stockpiles rather than an increase in production.

The bulk of Russian production comes from Noril'sk as a by-product of nickel and copper mining with only some 150 000 oz coming from placer deposits, which are believed to be near to exhaustion. The increase of 49% in platinum delivered by Russia is believed to reflect a need for hard currency at the time. During the period 1990-1993, the nickel output of the Noril'sk Nickel Combine fell by an estimated 45%, due to serious financial difficulties, resulting in a lack of finances to maintain production. Severe weather caused major problems at their smelters early in 1994 and an explosion at a power station in November 1994 further curtailed production.

Political "upheaval" in South Africa during 1994 is blamed for the drop in production compared to the 1993 figures. The period prior to the April election was dominated by strikes, protests and a general lack of direction and insensitivity on the part of management to the imminent new dispensation. The almost immediate introduction after the elections of several new public holidays and the inability of management structures to arrive at mutually acceptable agreement to cope with these, highlighted this. RPM (part of the Amplats Group) experienced a severe bottleneck at their smelter complex, caused by a modernization programme then underway. Impala closed No.11 Shaft, recognizing its lack of profitability and problems were also encountered at their concentrators. All these factors contributed to a drop in South African production during 1994.

Total demand in 1994 was up to 4.56 Moz against the supply of 4.53 Moz, creating a shortfall of 30 000 oz, which came from stocks (Table 1.1). Virtually every sector, including sales to China, showed increased demand. The exception being the petroleum sector, which took only 90 000 oz compared to 105 000 oz the previous year. At the time, it was felt that much of the demand was driven by fears over the South African elections and the effect the new dispensation would have on future short to medium term supply. The Japanese market especially, seemed to purchase more than it required. In retrospect, it can be seen that the demand was driven by a general recovery of western economies and the various sectors have generally increased their off-take in subsequent years. The possible exception to this is the investment sector, which is speculative by nature, and took 395 000 oz in 1994. The two largest sectors, autocatalysts and jewellery both increased their demand to 1.685 Moz (of which 310 000 oz came from recycling) and 1.74 Moz respectively (Table 1.1).

TABLE 1.1

Platinum Supply and Demand 1993-1996 ('000 oz)

SUPPLY		1993	1994	1995	1996
South Africa		3,360	3,160	3,370	3,420
Russia		680	1,010	1,280	1,100
North America		220	220	240	225
Others		130	140	100	105
Total supply		**4,390**	**4,530**	**4,990**	**4,850**
DEMAND					
Autocatalysts	**gross:**	1,685	1,870	1,850	1,820
	recovery:	-275	-310	-335	-370
Chemical		180	190	215	220
Electrical		165	185	215	235
Glass		80	160	225	225
Investment		305	395	345	235
Jewellery		1,615	1,740	1,810	1,840
Petroleum		105	90	120	125
Other		165	190	225	260
Sales to China		20	50	130	180
Total Demand		**4,045**	**4,560**	**4,800**	**4,590**
Movement in Stocks		**345**	**-30**	**190**	**80**

(After Johnson Matthey, 1996)

In 1995, South Africa produced 3.37 Moz of Pt, recovering somewhat from the setbacks the previous year. Russian sales increased to 1.28 Moz and North America increased production to 240 000oz. Production from other sources declined to 100 000 oz, giving a grand total for production of 4.98 Moz. The Russian sales were 188% of the 1994 sales. It is estimated that the Russian primary Pt production amounted to about 700 000 oz in 1995, therefore some 580 000 oz were drawn-down from stocks in 1995. The increase in South African production is attributed to several factors: RPM's modernization was completed in late 1994 providing better metallurgical recoveries as well as utilizing the concentrate backlog created during the modernization process; increased metallurgical recoveries reported from the Lonrho Group and slightly increased production from Northam. On the down side, Impala's main Merensky furnace was put out of action for six weeks from August 1995 due to a matte runaway and a fire which that caused. It was not until late November that Impala once again reached full production at the smelters. Some 56 000t of concentrate had to be "put on the floor" locking up 120 000oz of Pt. This, compounded by lower production from underground than anticipated meant that Impala's overall Pt production dropped from 1.01 Moz in 1994 to 900 000 oz in 1995, forcing them to buy metal on the spot market to fulfil contractual arrangements. Impala reopened No.11 shaft and also increased production from No. 14 shaft in an attempt to alleviate some of these problems. Despite the fact that production at Northam was up slightly, year on year, it was still battling to make a profit, and by June had spent R397 million of the R504 million it had raised through a rights issue early in 1995.

As mentioned previously, 1995 showed further demand increases, with total demand rising to 4.80 Moz. All sectors except investment and autocatalysts showed increases. Increases in the jewellery sector in Japan were due to the strength of the Yen pushing platinum prices to its lowest levels since the 1970's, prompting increased buying at what were perceived to be low prices. The decline in demand in the autocatalyst sector was primarily due to increasing application of Pd-rich autocatalysts at the expense of the more costly Pt autocatalysts. China boosted its off-take to 130 000 oz, mainly for use in their jewellery and industrial sectors.

The recently published Johnson Matthey Interim Results (1996), details the performance of the suppliers and industrial demand, based on the first nine months of the calendar year, and provides forecasts for the whole year. This indicates that total production is expected to fall to 4.85 Moz. South African production has risen by a meagre 1% to 3.42 Moz, due mainly to Impala processing the concentrate they were forced to put on the ground after the accident at their smelter during 1995. Russia's sales are down from 1,28 Moz in 1995 to 1.1 Moz in 1996, due to lower demand from the Japanese markets. North America's production will probably fall slightly to 225 000oz despite a 15% increase in output from Stillwater. The drop in output is due mainly to the variable PGM grade of the Ni deposits in Canada which provide a sizeable proportion of the North American production, as well as a reported drop of head grade at the North American Palladium Mine. Production from other producers will increase by ± 5 000 oz to 105 000 oz, (due mainly to the Hartley Mine coming on-line and production from Mimosa) giving a grand total of 4.85 Moz, as shown in Table 1.1.

The 1996 production year in South Africa got off to a very poor start. All three producers reported low productivity and a failure to meet production targets. Although this will have given Impala a welcomed opportunity to process the stockpiled concentrate, created when it lost its main furnace in August 1995, their annual production will probably not greatly exceed

900 000oz, despite several years of attempting to produce 1.0 Moz of PGM. The strikes early in 1996 at Amplats have created a shortfall of 150 000 oz. However, Amplats may have made up some of the lost production through its new UG2 open cast operation at Amandelbult and its Merensky open cast at Bleskop. As most of the production of the South African producers is sold forward, in the form of contracts, this means that Amplats will have to purchase 150 000 oz on the spot market to fulfil its contractual obligations. Effectively this means that the market will be short of 300 000 oz. Although production in South Africa improved in the latter part of 1996, the effects of this will not be seen until the 1997 reporting year.

1996 sees the first year of production of PGM's from Zimbabwe's Hartley Mine. This is estimated to be approximately 10 000 oz of Pt and is included in "others" in Table 1.1. Although most of the South African producers tend to dismiss Hartley's production, it is felt that this will be dangerous in the long term. BHP, Hartley's owner, is aggressively expanding in the PGM exploration/mining field and will be a force to be reckoned with in time. Strategic considerations will, it is believed, become more and more important once Russia depletes its stockpiles, making Hartley more attractive to western purchasers. Total production is likely to be down to about 4.85 Moz with Russian sales having dropped to 1.1 Moz.

Demand in 1996 is expected to be in the region of 4.77 Moz, a drop of 30 000 oz over the 1995 figure. The increasing substitution of Pd autocatalysts will to some extent be offset by increasing demand for Pt autocatalysts for diesel vehicles, especially in Europe. The other great unknown is, of course, the Chinese market, which is expected to absorb 180 000 oz in 1996. The jewellery demand is expected to increase slightly to 1.84 Moz in 1996. Any growth in the jewellery sector will largely be met from Pt in investment bullion taken up in 1994. The Japanese jewellery sector is coming under increasing pressure due to competition from gold, and a slow down in the Japanese economy. To some extent this is offset by increasing use of alloys with a higher Pt content.

Industrial demand will rise by 65 000 oz, due to small growth in most sectors. An interesting new growth area is that of fuel cells. The US Government has instituted a subsidy programme to encourage the commercialization of non-polluting power sources. Funds have also been made available to the Department of Defence to install fuel cells at all military installations across the US. Perhaps this marks the beginning of the long awaited expansion in the application of fuel cells. During 1996, Mercedes Benz launched its first fuel cell powered Zero Emission Vehicle. This application is expected to increase over the next few years so that the automobile manufacturers can comply with current legislation, in respect of requirements that a certain proportion of vehicles sold are Zero Emission Vehicles, especially in North America.

1.4 Palladium Supply and Demand

The Pd supply over the same period, 1993-1996, shows a slightly different picture. In 1993, the total Pd supply was 4.28 Moz, of which 1.395 Moz were produced by South Africa, 2.4 Moz by Russia, 415 000 oz by North America and 70 000 oz by other producers (Table 1.2).

Like Pt, palladium's demand is made up of several sectors: autocatalyst, chemical, dental, electrical, jewellery and "others" (Table 1.2). Of these, electrical (electronic) applications were by far the most important until 1996, consuming ± 50% of production. The dental sector was the next most important sector followed by autocatalysts. This situation changed in 1996.

In 1993, total demand was 4.265 Moz, of which electrical applications utilized 2.015 Moz, dentistry 1.21 Moz, and autocatalysts, 705 000 oz (of which 100 000 oz came from recycling). Demand and production were fairly well matched in 1993, with only 15 000 oz going into stocks.

1994 saw a dramatic increase in supply, from 4.28 Moz in 1993 up to 5.28 Moz in 1994, an increase of over 23%. This increase was due to increased production in South Africa to 1.5 Moz and a substantial increase in Russian sales to 3.3 Moz, up from 2.4 Moz in 1993. This took place, as mentioned previously, against a backdrop of very low production levels at Noril'sk and it is felt that the bulk of this was drawn down from stockpiles. It is difficult to estimate exactly how much of the 3.3 Moz was primary production and how much was drawn from stockpiles, as Russia does not publish any data on this. However, if the 1989 supply figures are taken to represent only primary supply, because Russia had not begun to sell PGM's from its stockpiles in any quantity at that time, and allowing for the fact that that production had dropped by 45% between 1990-1993, some 2.0 Moz must have been sourced from stockpiles during 1994. South African production of Pd rose slightly despite the problems encountered in production, mentioned above, mainly due to the increased proportion of UG2 ore being processed by all three of the major South African producers.

In 1994, supply exceeded demand by some 410 000 oz, with demand of 4.87 Moz against a supply of 5.28 Moz (Table 1.2). Demand in all sectors increased, except in the chemical sector. Autocatalyst demand grew by 270 000 oz while recovery from recycling increased by only 5 000 oz. Electrical use increased by 215 000 oz and dentistry by 55 000 oz. Demand in other unspecified sectors grew by 80 000 oz.

TABLE 1.2

Palladium Supply and Demand 1993-1996 ('000 oz)

SUPPLY		1993	1994	1995	1996
South Africa		1,395	1,500	1,600	1,670
Russia		2,400	3,300	4,200	3,800
North America		415	220	470	435
Others		70	140	70	75
Total supply		**4,280**	**5,280**	**6,340**	**5,980**
DEMAND					
Electrical		2,015	2,230	2,650	2,120
Dental		1,210	1,265	1,300	1,310
Autocatalysts	**gross:**	705	975	1,730	2,270
	recovery:	-100	-105	-115	-135
Chemical		190	185	210	220
Jewellery		210	205	205	200
Other		35	115	120	125
Total Demand		**4,265**	**4,870**	**6,100**	**6,110**
Movement in Stocks		**15**	**410**	**240**	**-130**

(After Johnson Matthey, 1996)

1995 saw yet another astounding increase in Pd supply, up to 6.34 Moz from 5.28 Moz the previous year (Table 1.2). Once again this reflected a massive increase of Russian sales, up 900 000 oz from 3.3 Moz in 1994 to 4.20 Moz in 1995. South African production increased to 1.6 Moz in 1995, mainly due to technical factors: RPM (Amplats) processed the concentrate that it had stockpiled while its smelters were being modernized; Lonrho increased production due to an increase in head-grades and recoveries, and production at Northam increased slightly. It will be remembered that Impala had lost their main furnace due to an accident late in 1995, but the effects of this were negligible in terms of Pd supply. The supply from North America increased to 470 000 oz mainly due to the North American Palladium Mine in Canada coming on line as well as increased production at Sudbury, due to the high Ni price. Stillwater's production dropped slightly due to expansion that generated large amounts of "low-grade" tonnage.

1995 saw a dramatic change in the use by sector (Table 1.2). Autocatalysts suddenly almost doubled demand to 1.73 Moz with only 115 000 oz being sourced from recycling. Dental applications grew (due to changes in the Japanese state health care system) as did electrical applications. The huge increase in demand by the autocatalyst sector was in response to new "clean-air" legislation both in western Europe and in North America. Although there was a concomitant decrease in Pt demand for petrol powered vehicles, this was offset by an increase in demand for Pt catalysts for diesel powered vehicles, created by the same legislation. However, the Pd loading (amount of Pd used in a catalytic converter) is greater than that of Pt catalysts. The growth in demand in the electrical sector was driven by increasing manufacture of MLCC's (Multi Layered Ceramic Capacitors), which are used in cellular telephones and computers, and continues to do so. Some of the increasing demand in these sectors may have been supplied from bullion held in investment bars taken up in 1994, but the scale of this is impossible to estimate. Total demand was up to 6.1 Moz, 25% up on 1994. Despite this, 240 000 oz went into stock, as supplies had risen to 6.34 Moz (Table 1.2).

As discussed previously, the early part of the 1996 production year was plagued by low productivity in South Africa as well as strikes at Amplats which affected the Pt production. These problems similarly affected the Pd production. It is expected that Pd supply will drop slightly to 5.98 Moz, the bulk of which will, once again, be supplied by Russia. Russian sales are expected to be in the region of 3.8 Moz in 1996. South African supply will increase to around 1.67 Moz, and North American supply could decrease to 435 000 oz. This decrease is due to lower grade production from the North American Palladium Mine and from other secondary producers in Canada. New supply is expected from Falconbridge's Raglan Mine in Northern Quebec during the 1997-1998 year. Zimbabwe's Hartley Mine will also contribute to Pd supply in the 1996-1997 year, but this will be a negligible amount. In all, the supply of Pd for 1996 is estimated to be approximately 5.98 Moz, up 80% on the 1989 supply figure of 3.235 Moz.

It is expected that Pd demand in 1996 will increase to around 6.11 Moz. Autocatalyst demand will continue to increase as will electrical demand, making up the bulk of this increase in demand. Russian sales will once again exceed actual production, but due to the low Pd spot price, it is felt that Russian sales will simply take up the slack in the market, probably selling ± 3.8 Moz. It appears that the threat of substitution worries the Russian producers. This was especially evident in 1995 when they increased sales to 4.2 Moz in the face of threats of substitution on the part of the Japanese Kyocera Corp. when Pd reached $174/oz (Mining Journal, March 1995). This fear probably reflects the fact that Russia still holds vast stockpiles of Pd and would like to see Pd trade at prices that will not confer any benefit in substituting base metals for Pd (Mining Journal, 1995).

1.5 Rhodium Supply and Demand

By far the largest producer of Rh is South Africa. In 1993 total Rh supply was 376 000 oz, of which South Africa provided 278 000 oz (or 74%), Russia 80 000 oz, North America 17 000 oz and 1 000 oz were produced by other producers (Table 1.3).

Rh production has increased due mainly to two factors: firstly, increased overall production of Pt and Pd and secondly, an increasing proportion of that production being sourced from the UG2 in which the Rh makes up ± 9% of contained metal as opposed to ± 3% in the Merensky reef. This is an ongoing trend, as will be seen in the data for 1994-1996 (Table 1.3), and goes hand-in-hand with increasing Pd supply from South Africa, despite the Pt supply reducing in 1994. As with Pt and Pd, Rh has several applications, notably autocatalysts (the largest sector by far). It is also used in the chemical, electrical and glass fabrication sectors.

In 1993, Rh demand in the autocatalyst sector was 356 000 oz, including 26 000 oz from recycling (Table 1.3). Total demand was 365 000 oz against a supply of 376 000 oz, requiring 11 000 oz to be taken up by stockpiles.

1994 saw production rise to 426 000 oz, of which 330 000 oz came from South Africa, 80 000 oz from Russia, 15 000 oz from North America and 1 000 oz from other producers (Table 1.3). In 1994, autocatalyst demand increased to 379 000 oz, of which 36 000 oz was sourced from recycling. The glass fabrication industry increased its demand to 14 000 oz, up from only 3 000 oz the previous year. The other sectors each increased their demand slightly. Total demand was 386 000 oz against a dramatic increase in supply, mainly by South Africa, of 426 000 oz. Some 40 000 oz were taken up by stockpiles (Table 1.3).

Production and supply from South Africa further increased during 1995. 342 000 oz were supplied by South Africa and 80 000 oz by Russia. North American supply dropped slightly to 13 000 oz, probably reflecting the expansion problem encountered at Stillwater. Other production remained static at 1 000 oz.

TABLE 1.3

Rhodium Supply and Demand 1993-1996 ('000 oz)

SUPPLY		1993	1994	1995	1996
South Africa		278	330	342	355
Russia		80	80	80	90
North America		17	15	13	16
Others		1	1	1	2
Total supply		**376**	**426**	**436**	**463**
DEMAND					
Autocatalysts	**gross:**	356	379	450	405
	recovery:	-26	-36	-39	-47
Chemical		11	10	13	17
Electrical		9	8	8	7
Glass		3	14	17	28
Other		12	11	10	9
Total Demand		**365**	**386**	**459**	**419**
Movement in Stocks		**11**	**40**	**-23**	**44**

(After Johnson Matthey, 1996)

The South African figure of 342 000 oz for 1995 reflects sales rather than production. With an average market price at a 22 year low, the South African producers, it is believed, sold only what contractual agreements required and stockpiled the remainder as they are prone to do (Vermaak, 1995). It is estimated that any production in excess of contractual sales may have been stockpiled, awaiting price improvements.

A major reversal of this situation occurred in 1995, with demand exceeding supply by 23 000 oz (Table 1.3). This probably reflects the stockpiling of Rh by South African producers mentioned previously. Autocatalyst off-take increased further to 450 000 oz with only 39 000 oz of this being sourced from recycling. The glass sector continued to grow, taking 17 000 oz and 13 000 oz being taken up by the chemical sector.

The increase in demand by the autocatalyst sector was in response to tighter emission requirements in North America as well as increased manufacturing levels in the automobile industry in SE Asia. The fact that Rh was at a 22-year low probably boosted sales. The other industrial demand increases were due more to thrifting than a real increase in PGM demand. In the early 1990's many industrial Rh applications were substituted by Pt due to its lower price and similar physical characteristics. However, Pt is not as hard wearing as Rh and as these Pt parts needed to be replaced, Rh was once again used in preference to Pt due to its then current lower price. Obviously this occurs at the expense of Pt.

1996 will see primary Rh production (sales) rise to ± 463 000 oz, with the South African producers possibly selling just enough to fulfil contractual obligations, and stockpiling the remainder. Russian output is expected to increase slightly to 90 000 oz, while North American supply will increase slightly to ± 16 000 oz as the expansion at Stillwater nears completion. Other output will increase to 1 000-2 000 oz with production from Hartley being included here.

During the period 1993-1996, Russia has dramatically increased its sales of both Pt and Pd, but not Rh. This can be interpreted in one of two ways: either their increased sales of Rh during the 1988-1991 period, when Rh was highly priced, exhausted their Rh stockpile or, they feel that current spot market prices are too low to justify such continued sales. Indeed, were more Rh to be put on the market it would further depress the price. In light of this, it is difficult to explain Russia's decision to increase Rh sales at this point in time, unless the additional sales were sourced from recycled material, which is discussed below.

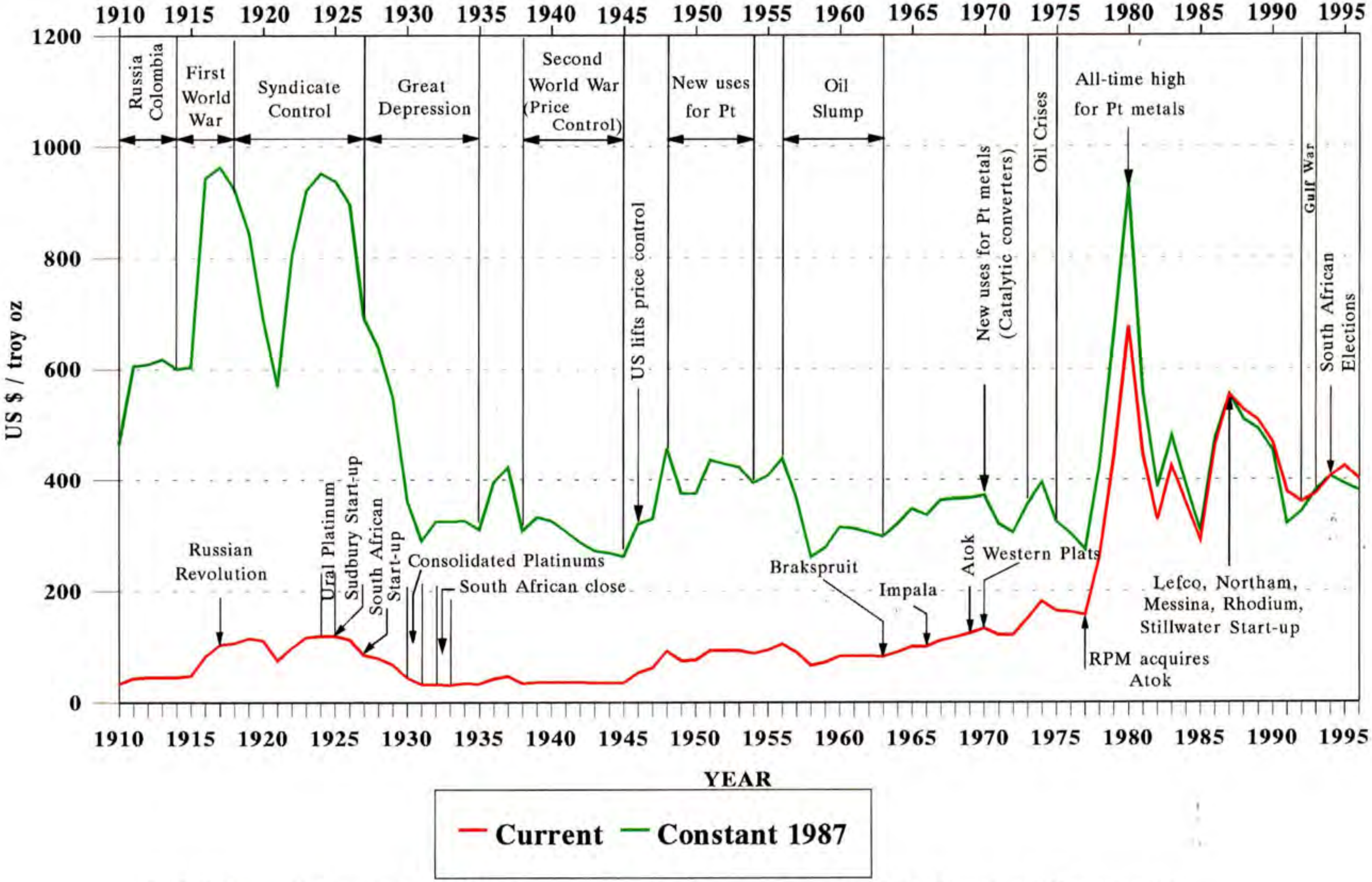
Demand for Rh will continue to grow in 1996, driven by the current low price, and tighter emission control regulations, especially on heavy diesel vehicles. Since 1988, ±140 000 oz of Rh have gone into stockpiles (excluding the stock currently held by South African producers) and only 47 000 oz have been drawn down from those stockpiles, giving a net current stock of 93 000 oz. However, some of this may have been used by Japanese end-users over the last few years. Despite this there could be 50 000-60 000 oz of Rh available to the market, over and above supply. The fall in Rh demand, down from 459 000 oz in 1995 to 419 000 oz in 1996, with 44 000 oz going into stocks will no doubt further depress the Rh price. The only real growth in this sector will be the autocatalyst sector that requires a minimum amount of Rh in the catalysts to deal with diesel emissions.

1.6 PGM Prices

An analysis of the Pt price through time (Figure 1.1) demonstrates a number of interesting points. The data is plotted in constant 1987 (US $) money terms. The evident price cycle period for Pt is in excess of 25 years and at present the Pt price is in the middle of a period of low price, having peaked in 1980 at $677/oz. The previous peaks were in 1920 and in 1956. The next peak will occur around the year 2005.

In present money terms, Pt has traded below its average price for 53 of the 85 years for which data is available, and in fact, it has declined for 39 of those years. Although Pt traded for in excess of $100/oz in 1917, Pt has only reached triple figures (US $) for 39 of the 85 years. In 1910, the Pt price was $31/oz, the same as it was in 1933. In 1945 the Pt price was $35/oz. Russia (Soviet Union) has dumped Pt on the world market periodically since 1924.

For no apparent reason there exists a 97% positive correlation between the spot market price of gold and platinum (Figure 1.2) especially since 1975 (Figure 1.2.1). "No apparent reason" needs some explanation: Pt production is one-twentieth that of gold; there are a small number of Pt producers (93.8% of Pt was produced by South Africa and Russia between 1982 and 1994); Pt is more difficult to mine and refine than gold, leading to much longer pipelines than for gold; PGM have more unique applications than gold, both as precious and industrial metals (Vermaak, 1995). However, Vermaak (1995) points out that there is indeed a very good reason why this correlation should exist: the same institutions, and indeed the same individuals, who set the gold price in London twice a day also set the platinum and palladium price. These institutions are known as the "Club of Five" and are made up of Rothschild & Sons, Mocatta & Goldschmid, Sharps Pixley, Samual Montague, and Mase-Westpac. Mase-Westpac replaced Johnson Matthey Bankers Ltd in 1984.



NOTE: — Adjusted to 1987 constant dollars using the US Dept. of Commerce GNP implicit price deflator

Figure 1.1 : Average Annual Price of Platinum Metal, 1910 to 1996 (After Vermaak, 1995)

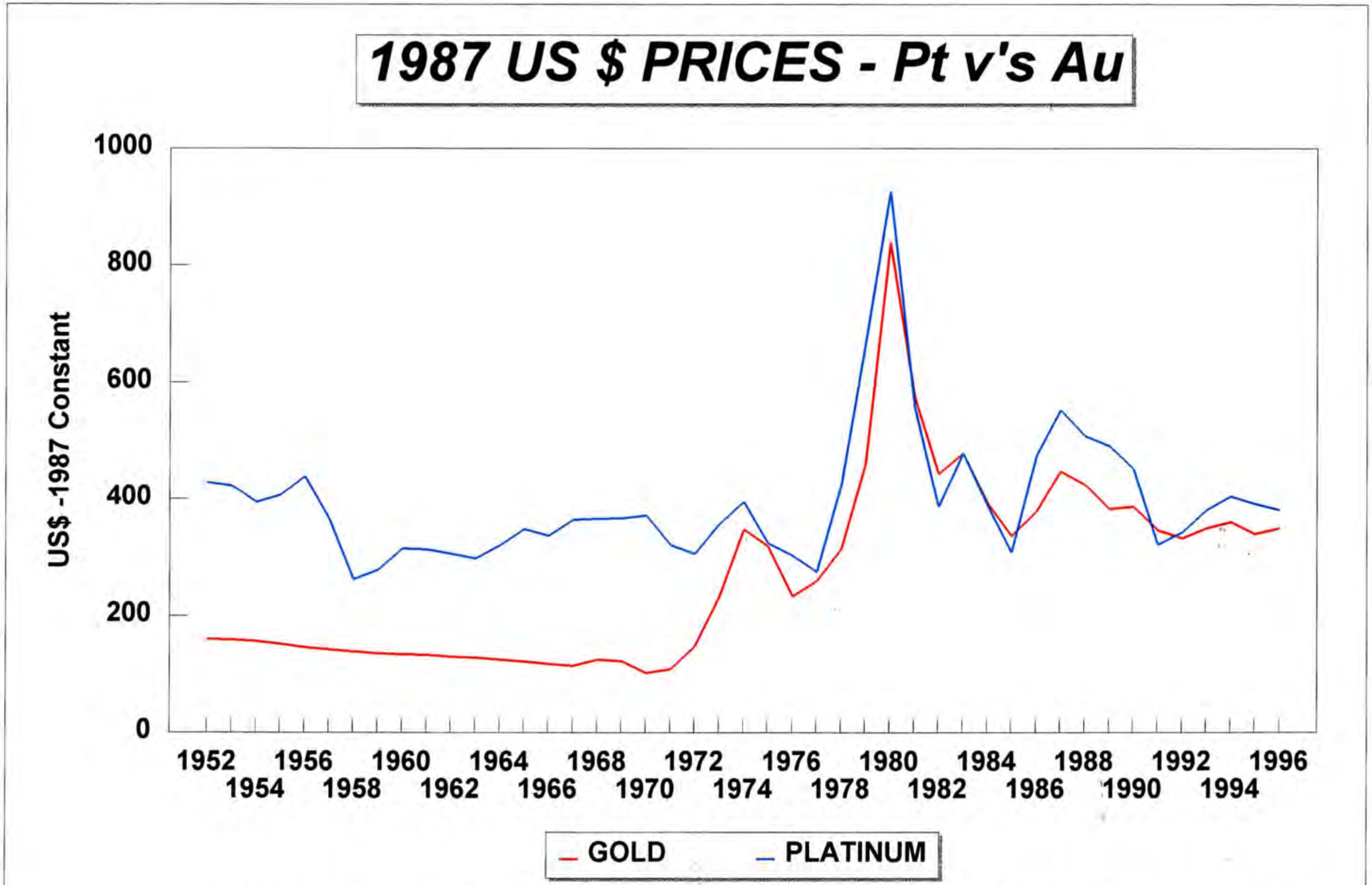


Figure 1.2 Correlation of Pt & Au Prices 1952 - 1996

Although the Pt and Pd prices are supposed to be determined according to balance of supply and demand, it is easy to imagine that the gold price can influence this in the minds of the people who actually set the prices, in disregard of the fundamentals applicable to Pt and Pd at that moment in time. Indeed, that appears to have been the case in 1993, when speculators, George Soros and Sir James Goldsmith, drove the gold price up by purchasing massive amounts of gold bullion, and Pt and Pd rose in sympathy with it despite the market fundamentals being completely different at the time. Vermaak (1995) notes that there is a good inverse relationship between the gold (and therefore platinum) price and rhodium price.

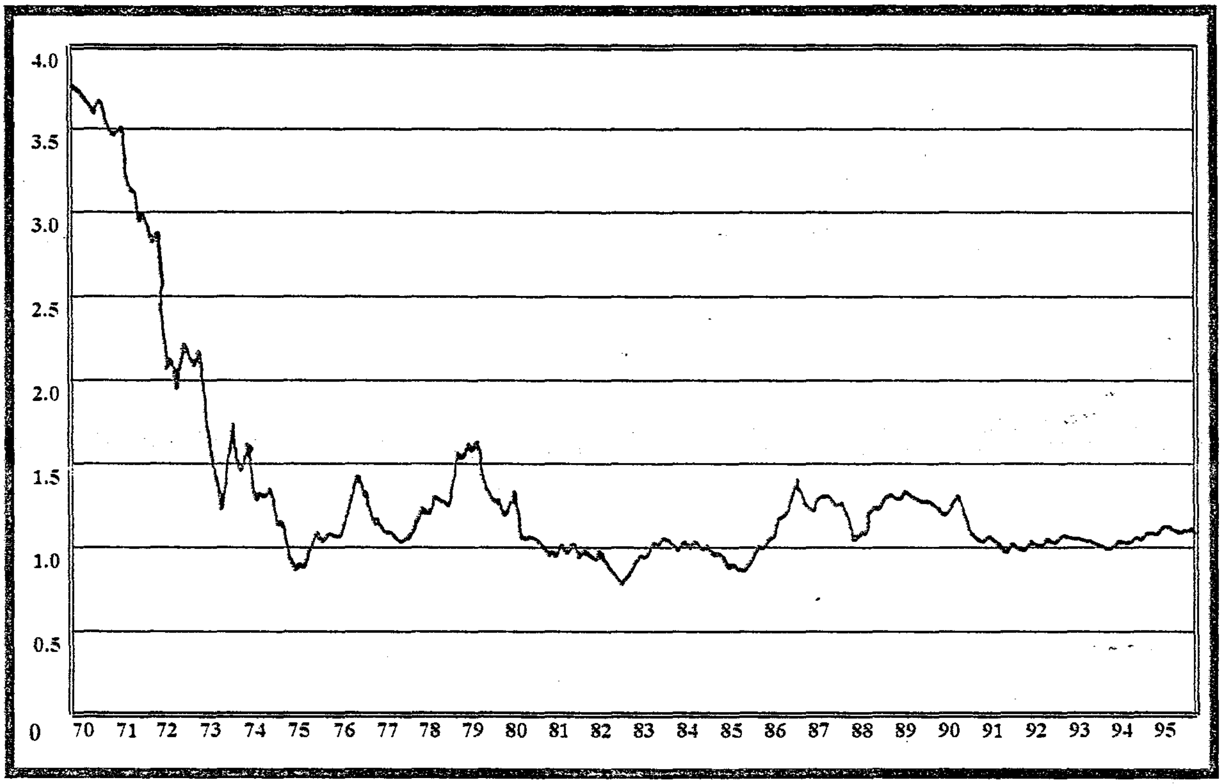


FIGURE 1.2.1

Platinum to Gold Price Ratio: 1970-1995

(From Forrest, 1996)

This is felt to reflect true market fundamentals of Rh. If the Pt/Pd prices are high, Pt and Pd production will be increased, which in turn will *de facto* cause an increase in the Rh supply. As the applications (market) for Rh are much more limited than for either Pt or Pd, the Rh market price is much more sensitive to oversupply (and shortages) and therefore the price will drop.

In general terms, it is safe to assume that Pt (and Pd) will follow the gold price, usually trading at a slight premium to gold, in the absence of any real fundamental issues. When there are real fundamental issues, Pt should trade at prices reflecting these.

As in other mineral mining sectors, there is a strong and growing belief in the PGM sector that there exists a "conspiracy" that somehow controls the PGM metal prices. Different lines of "evidence" are cited to support this "theory". An examination of PGM matte valuation prices (in constant Rand terms) reveals that they have remained static, or decreased slightly over the past six years. This would seem to suggest that the price is managed in Rand terms. No real benefit of the recent Rand/Dollar depreciation has filtered through to the South African PGM mining industry during 1996, due to synchronous drops in the metal Dollar prices. Another argument used is the continued willingness of major PGM refiners and fabricators (Johnson Matthey and Engelhard) to purchase all and any PGM's produced, which has, it is widely believed, been formalized in the form of contractual agreements.

Another factor in this argument is the manipulation of information on the part of these two refiners and fabricators. A sizeable proportion of Russian sales is in fact from recycled scrap material, which is toll refined by the Russian Krasnoyarsk Refinery for Johnson Matthey and Engelhard. This recycled PGM is reported under Russian sales rather than under autocatalyst recovery data. The autocatalyst recovery data that is reported accounts only for material recovered in North America and western Europe.

Sizeable inventories of old Russian PGM's are held by various concerns (banks and fabricators) in Switzerland and Japan. The movement of this stock, when sold and moved from one institution to another, is reported as "Russian sales". These factors have the effect of making the Russian sales appear to be much larger that they actually are. The Russian suppliers obviously are happy to go along with this illusion, as it appears to give them considerable sway in the market place. Like the South African producers, they prefer to reveal as little information as possible about their production. In fact, all information about Russian PGM production and sales is classified as a state secret, by right of a Presidential decree.

Another artificial factor in the reporting mechanism is that of reporting Investment bullion sales under the heading of Demand. Investors merely hold stocks, on a speculative basis, and these stocks are readily available to the market when the criteria of the "investor" are met by the market place. To include investment purchases in demand data distorts the total analysis by making the balance between the amount of new metal supply entering the market and "demand" always seem close to each other.

George Soros and Sir James Goldsmith were able to manipulate the gold price in 1993, despite the plethora of producers and abundant stocks held world-wide, and indeed George Soros in currently driving the value of the French Franc downwards at a dizzying pace. His motives are best known only to himself, but no doubt there is a strong profit motive involved. So perhaps there is some shred of truth or reality in the conspiracy theory as far as the PGM market is concerned.

In 1993, the average Pt price was $374/oz (Figure 1.3) with trading taking place within the range of $350-$403/oz. There was little price movement over the 1992 price, despite increased demand, as supply was also up, as discussed earlier. The average Pd price in 1993 increased by 39% over the 1992 price. However, rather than just following the Pt price as it is prone to, this price rise was driven by market fundamentals as well. In 1993, supply and demand were well matched, there being only 15 000 oz surplus, due to massive rise in Pd utilization in the electronics sector. Rhodium continues its long slide downwards due, once again, to market fundamentals: oversupply and limited applications. Rh traded at an average price of $1 133/oz in 1993, down from $2 478/oz in 1992.

Platinum continued its slow recovery during 1994, trading at an average price of $405/oz for the year. It reached a high of $419 in October 1994 but the price eased for the last three months of the year, as it tends to. Pd increased in price steadily in the first nine months of 1994, thereafter seeming to lose direction once again in the last three months.

The average price was $142/oz, having reached a high in November of $156/oz. Again the price was driven by strong demand, especially in the electronics sector. Rh continued to lose ground in 1994, trading at an average of only $743/oz in the face of oversupply.

1995 saw the Pt price rise to its highest level since 1990, reaching $460/oz in trading in late March - early April, but the average price rose to just $424/oz. Once again, and this is a

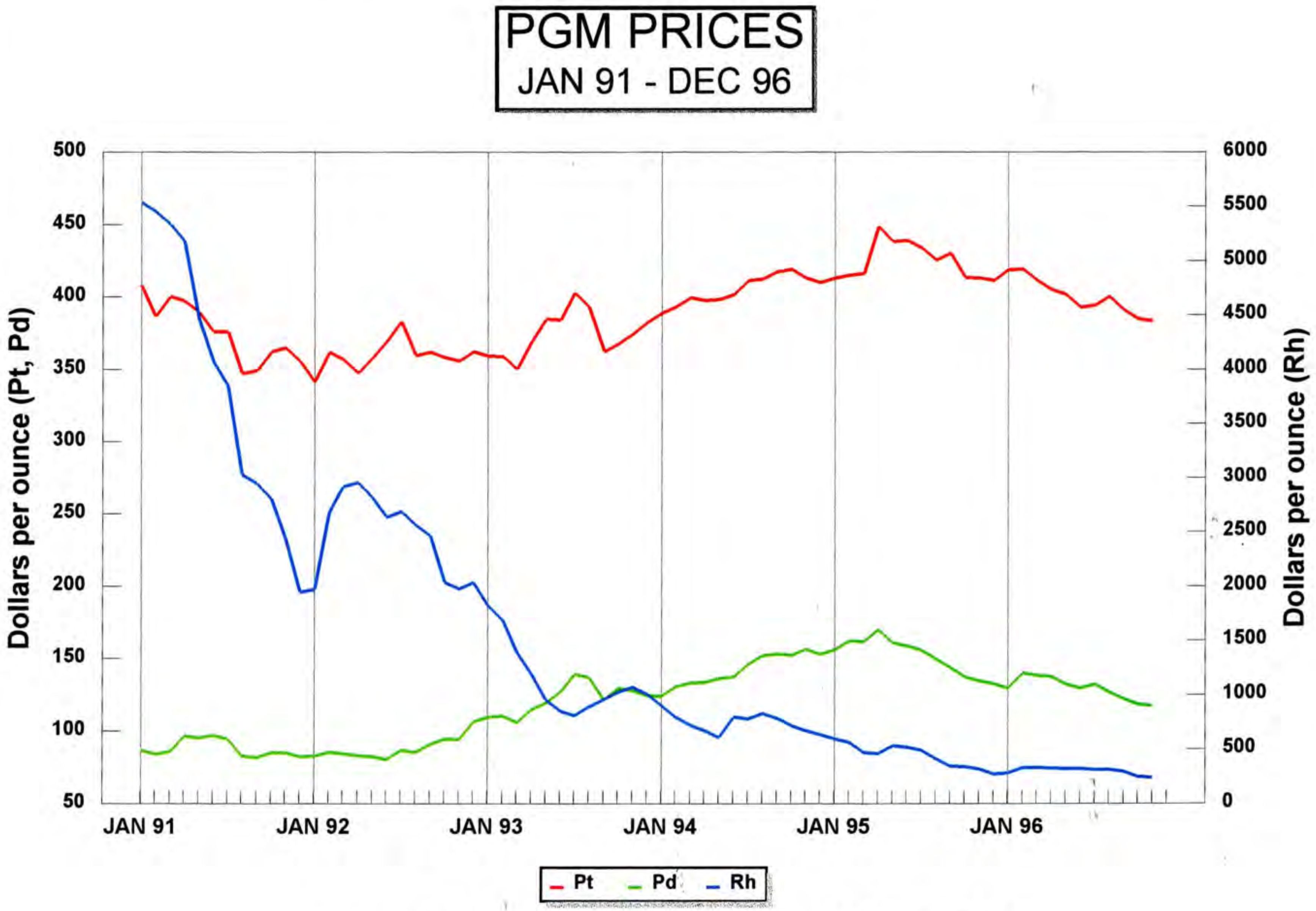


Figure 1.3 Pt, Pd, & Rh Prices 1991 - 1996

common feature of the Pt price, the price softened in the last three months of the year, the market seeming to lack direction. The early part of 1995 saw the collapse of the Mexican Peso and "Emerging Markets" sought refuge in precious metals pushing the Pt price up. Palladium continued its increase in price, although at a slower rate. The average price for 1995 was $151/oz, having reached a high of $178 in April. However, with continued massive shipments from Russia, the price declined to a 21 month low of $127/oz by the end of the year. Rhodium continued its downward spiral, halving its value to an average of $440/oz for the year. During August it fell below the level of Pt for the first time in over ten years. By December 1995 it had reached a 22 year low of $260/oz, despite being in short supply in the market (Table 1.3).

Early 1996 saw the Pt price remain fairly strong, trading between $400-$420/oz. However, since then it has declined to below $400/oz and is currently trading at $380-$390/oz. In the latter part of 1996 it has followed the fortunes of gold very closely, although the fact that Pt will once again be in oversupply this year provides fundamental reasons for this. Palladium has also suffered in 1996, with the average price dropping to $129/oz. This is, it is believed, more in sympathy with the falling Pt price than real fundamentals. With Russian shipments for the year expected to be approximately 3.8 Moz, and modest increases in demand, Pd should be in short supply, and obviously the price should increase. This is not happening and therefore the belief that it is the gold price that is driving the Pd price at present rather than real market fundamentals. This will undoubtedly have a marked effect on the price early in the new year, when the Pd price is expected to better reflect the balance of supply and demand.

1.7 The Future

Price prediction is, even at the best of times, a very dangerous occupation. These are not the best of times, and so, predictions will be attempted only in gross terms. Until the Russian stockpiles have been exhausted, the fundamentals for Pt look poor, especially in the face of declining demand. In the absence of market fundamentals, it would appear that Pt will follow the fortunes of gold in the short term. This was clearly demonstrated in October/November 1996, when the gold price began a dramatic drop in value, when the International Monetary Fund announced that it would be selling ±500t of gold and the Swiss Reserve Bank announced its intention to reduce its reserves of gold. The gold price fell below $370/oz at a time of year when it has traditionally strengthened, due to demand for jewellery over the Christmas period.

Early January 1997 saw the gold price predicted to fall to as little as $330/oz within the coming twelve months. The price of Pt, Pd and Rh all followed the downward trend set by gold, although there does not appear to be any real fundamental reason for them to have done so at this time. However, if the analysis of the history of the Pt prices is correct, Pt prices should peak early in the next century, around 2005 (Figure 1.3) when the next 25 year cycle peak is due. This is very tentative, and given recent performances, this upswing may never materialize.

Palladium, on the other hand, appears to be undergoing a dramatic increase in demand, in both the autocatalyst and electronics sectors, which will continue, provided that the price does not rise above about $170/oz. Above this level, the electronics sector will probably substitute Pd with base metals. This is a real threat, as practical problems of high temperature firing of ceramics (silicone chips) using base metals, such as nickel and silver, have been overcome. With increasingly stringent anti-pollution legislation being brought into force in North America, Europe, South America, South East Asia and Australia, the demand for Pd is assured to grow. However, so too is recovery from recycling, as the PGM's are virtually indestructible (Vermaak, 1995). As the clean-air legislation is expanded to include diesel powered vehicles, the demand for Rh will also increase. Because the Rh loading is so small in these types of catalytic converters, Rh may still find itself in oversupply. Intuitively, due to the rarity of Rh, it is felt that Rh should trade at a small premium to Pt. That this will be achieved in the short to medium term is unlikely. Rh will, in all likelihood, continue to trade at a price lower than Pt until demand increases and outstrips supply once again.

One of the greatest single influences on PGM prices is that of supply from Russia. Russia, as noted earlier, does not make data about their production and stockpiles available. However, estimates (Forrest, 1995) indicate that Russia may only have 1.88 Moz of Pt left in stockpiles. At the rate that Russia has been drawing down this inventory, in the region of 500 000 to 700 000 oz per annum, this could be exhausted in three years (i.e. by the year 2000). This will put demand in excess of supply by as much as 300 000 oz, discounting the investment sector, at current production capacity. The recent announcements by South African producers in terms of planned increased production may be able to cover this deficit. Supply and demand are expected to be fairly closely matched, but the price of Pt will probably rise, as fabricators build stocks to protect themselves in the short to medium term.

Russian stocks of Pd, on the other hand, are estimated to be nearly 10.0 Moz (Forrest, 1996). If Russia continues to sell Pd at rates comparable to recent years, this inventory could be completely depleted in five to six years.

At current production levels, it is unlikely that the NNC is contributing any new metal to inventories in stockpiles. It is also not known what proportion of Russian sales is sourced from recycled material supplied to them from the west. The Russians would obviously like to deplete their Pd stockpiles without affecting the Pd price too much, but also without precipitating substitution. The Pd price will therefore remain fairly static in real terms over the next five to six years. Once Russian stockpiles are depleted, there will be a substantial shortfall of Pd in the market, probably driving prices up. Some of this shortfall will be met by South African production, which will increase as Pt production is increased. The extent to which this will occur will be dictated to some extent, by where that expansion takes place, through increased Merensky or UG2 production. As much of the expansion will be within the Amplats Group, and specifically at PPRust, it is unlikely that South African production will be able to meet this shortfall, which could be as much as 1.5 Moz per annum. This will present a window of opportunity to the North American producers, and possibly boost exploration for Pd-rich deposits everywhere.

The increase in production in South Africa to meet the Pt supply towards the end of this century, will no doubt create a considerable oversupply of Rh. No real growth is expected in Rh demand, and this oversupply will further depress prices, possibly even below current levels in real terms. Russian stocks of Rh are unknown and it is thought that Russian Rh sales probably represent current production. A significant change in Russian sales is therefore felt to be unlikely.

1.7.1 Conclusions

This market summary has attempted to outline the problems faced by PGM producers as far as the market is concerned. Static or declining metal prices, when viewed against the rising production costs in the South African PGM industry, which are beyond the scope of this work but nonetheless very real, means that the profitability of the industry is being eroded at an alarming rate. The short- to medium-term market for PGM's looks bleak. Current metal prices severely restrict the profitability of PGM producers in South Africa and is requiring them to examine alternatives to address short- to medium-term production problems or "windows of opportunity" presented by overcapacity in processing plants. It is against this background that the remainder of this work will examine the potential of small-scale open cast mining of, specifically but not exclusively, the UG2 in the western Bushveld Complex.

1.8 The Bushveld Complex

The Bushveld Complex (BC) is the largest known layered complex in the world, occupying some 60000 km² mainly in the Northern and North West Provinces of South Africa. It extends for 370 km in an east-west direction and 365 km NW-SE. The total thickness of the BC is some 7500m. The BC consists of a series of interconnected (at depth) lobes or compartments, which were probably originally connected to a deep magma chamber (Sharpe & Snyman, 1980) of which five have been recognized. The most important of these from a PGM supply point of view are the eastern lobe, the western lobe and the northern lobe or Potgietersrus lobe (Figure 1.4).

The extrusion of large volumes of basaltic and felsic magmas marked the event that gave rise to the BC. The basaltic magmas formed the Dullstroom volcanics and the felsic magmas gave rise to the Rooiberg felsites, towards the end of the Transvaal sequence sedimentation. The Transvaal sediments were intruded by a series of pre-Bushveld sills and then by the mafic rocks of the BC (Vermaak, 1995).

The mafic magma spread upwards and outwards, probably following low-angled fractures and weaknesses (Sharpe & Snyman, 1980). This formed a series of seven shallow, overlapping cone-shaped intrusions, which enlarged with the introduction of more magma from depth into the system, producing the lobes of the BC as they exist today (Figure 1.4). The layering in the three main lobes, the western lobe, the eastern lobe and the Potgietersrus lobe, is essentially similar, especially the upper Critical and the Main Zones (Vermaak, 1995).

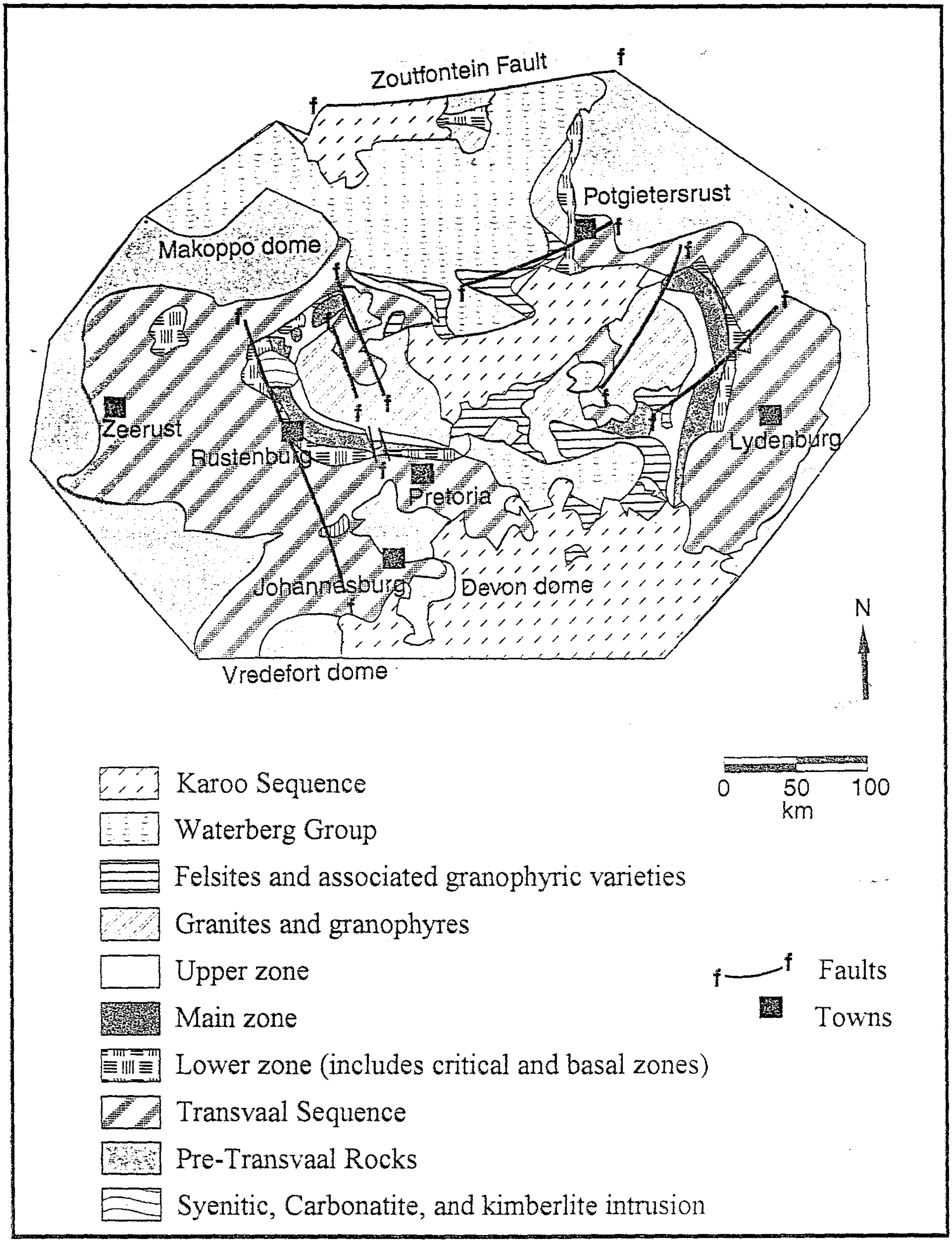


FIGURE 1.4

The Setting Of the Bushveld Complex in the Kaapvaal Craton

(After Vermaak, 1995)

1.9 Stratigraphy of the Bushveld Complex

The BC is subdivided into three suites; the Rustenburg Layered Suite (RLS), the Rashoop Granophyre Suite (RGS) and the Lebowa Granite Suite (LGS) (Tasble 1.4). The BC was emplaced into the Transvaal sequence, a 12km thick volcano-sedimentary sequence of rock within the Kaapvaal Craton. Both the Transvaal Basin, in which the Transvaal sequence were accumulating, and the BC are thought to be genetically related to the Ventersdorp Rift architecture (Clendinin *et al.*, 1988) and associated mantle plume (Houseman, 1990) respectively.

Unfortunately, an in-depth description and discussion of the BC is beyond the scope of this work and this section is only intended to introduce the reader in broad terms to the geology of the Rustenburg Layered Suite, specifically in the western BC lobe (Figure 1.5). A detailed review of the BC is provided by Eales *et al.* (1993) and Von Gruenewaldt *et al.* (1985). The stratigraphy is discussed with special emphasis on the UG2 Chromitite layer. The Merensky Reef is also described briefly.

The stratigraphy of the BC resembles that of other layered complexes, and is remarkably consistent within and between the various lobes of the complex. Individual layers in the RLS can be traced for tens of kilometres within lobes, and can be correlated between lobes, e.g. the Merensky Reef and the UG2. The BC displays discordant and pseudo-concordant relationships with the Transvaal sequence (Sharpe & Snyman, 1980).

1.10 The Rustenburg Layered Suite

The RLS contains the platiniferous layers of the BC, and this section will concentrate, for this reason, on the RLS. The RLS (Table 1.4), composed of the mafic and ultramafic phases of the BC, has been dated at 2050 ± 22 Ma (von Gruenewaldt *et al.*, 1985), and more recently at 2025 ± 40 Ma (Lee & Butcher, 1990). Both the Formal and Informal subdivisions of the RLS are presented in Table 1.4. More detailed work on the RLS in the western Bushveld Complex is provided by Eales *et al.* (1994) and Maier & Walters (1994).

TABLE 1.4

Formal Subdivision			Age (Ma)	Informal Subdivision
Bushveld Complex	Lebowa Granite Suite	Makhutso Granite Nebo Granite	1 670 ±30 U/Pb 2 010 ±20 U/Pb	Bushveld Granite
	Rashoop Granophyre Suite	Stavoren Granophyre	2 000 ±30 U/Pb	Bushveld Granophyre
	Rustenburg Layered Suite	Luipershoek Olivine Diorite Ironstone Magnetite Gabbro Magnet Heights Gabbronorite Mapoch Gabbronorite Leolo Mountain Gabbronorite Winnaarshoek Norite-Anorthosite Winterveld Norite-Anorthosite Zwartkoppies Pyroxenite Serokolo Bronzitite Jagdlust Harzburgite Rostock Bronzitite Clapham Bronzitite Shelter Norite	2 050 ±22 Rb/Sr	Upper Zone: subzone C, subzone B, subzone A Main Zone: subzone C, subzone B, subzone A Critical Zone: subzone B, subzone A Lower Zone: subzone D, subzone C, subzone B, subzone A Marginal Zone:
	Marico Diabase Suite		≥2 050 ≤2 224	Maruleng type Lydenburg type
Transvaal Sequence	Rooiberg Felsite Group	Selonsrivier Formation Damwal Formation	No reliable Data	Rooiberg Felsite

Subdivision and Chronological Framework of the Bushveld Complex.

(After Vermaak & Von Gruenewaldt, 1986)

1.10.1 The Marginal Zone

The Marginal Zone (Shelter Norite) consists of chilled pyroxenitic and noritic rocks. The marginal rocks of the BC are thought to be both pre- and syn-Bushveld sills and form the floor of the complex (von Gruenewaldt, 1985). The pre-Bushveld sills are thought to give the best estimate of the composition of the parental magma of the RSL (Hatton & Sharpe, 1983). This represents a critical step in understanding the Lower Zone, as the parental magma of the Lower Zone is boninitic in composition. This boninitic magma is believed to have played a major role in the formation of chromitite and PGE-mineralization within the RLS (Hatton & Sharpe, 1983).

The Marginal Zone rocks consist of norites and some quench-textured micro-orthopyroxenites (Sharpe & Hulbert, 1985) which enclose xenoliths of (now meta-) sedimentary and some mafic rocks, and form a 50-200 m thick sequence.

The boninitic composition of these rocks is confirmed by their enrichment in Zr, Rb and K_2O and high SiO_2 and MgO values. The importance of the presence of these boninitic rocks is seen to be twofold: firstly, the B1 boninites (Hatton & Sharpe, 1983) are thought to be "the best available representatives of the magma that gave rise to the Lower Zone and other pyroxenitic rocks of the Bushveld Complex". The pyroxenitic rocks are inevitably associated with chromitite and PGE mineralization in the RLS and the B1 magma is seen as the source of both the PGE and Cr in these layers. Secondly, the B1 boninites of the BC, being very similar to B1 boninites elsewhere, infer that they were formed in a subduction-related setting (Hatton & Sharpe, 1983).

1.10.2 The Lower Zone (LZ)

The Lower Zone is ±1700 m thick and divided into 4 subzones (A-D) (Cameron, 1978). It is best known from the eastern BC, where it crops out. The LZ consists mainly of dunites, harzburgites and orthopyroxenites (Figure 1.6). The Clapham and Rostock Bronzitites (subzone A & B of the Lower Zone) form a major (400 m thick) unit. This is overlain by the Jagdlust Harzburgite, (subzone C of the Lower Zone). The contact between these two subzones is taken to be the first appearance of cumulus olivine. The harzburgite subzone consists of a series of cyclical units: massive dunites followed by harzburgites. Intercumulus orthopyroxene (bronzite) increases upwards forming oikocrysts (Cameron, 1978). The layering within this sequence is well developed. The harzburgite subzone is followed, gradationally, by the upper Serokolo Bronzitite (subzone D of the Lower Zone). In the eastern BC this is considered to be part of the Lower Zone, but part of the lower Critical Zone in the western BC (Eales *et al.*, 1990).

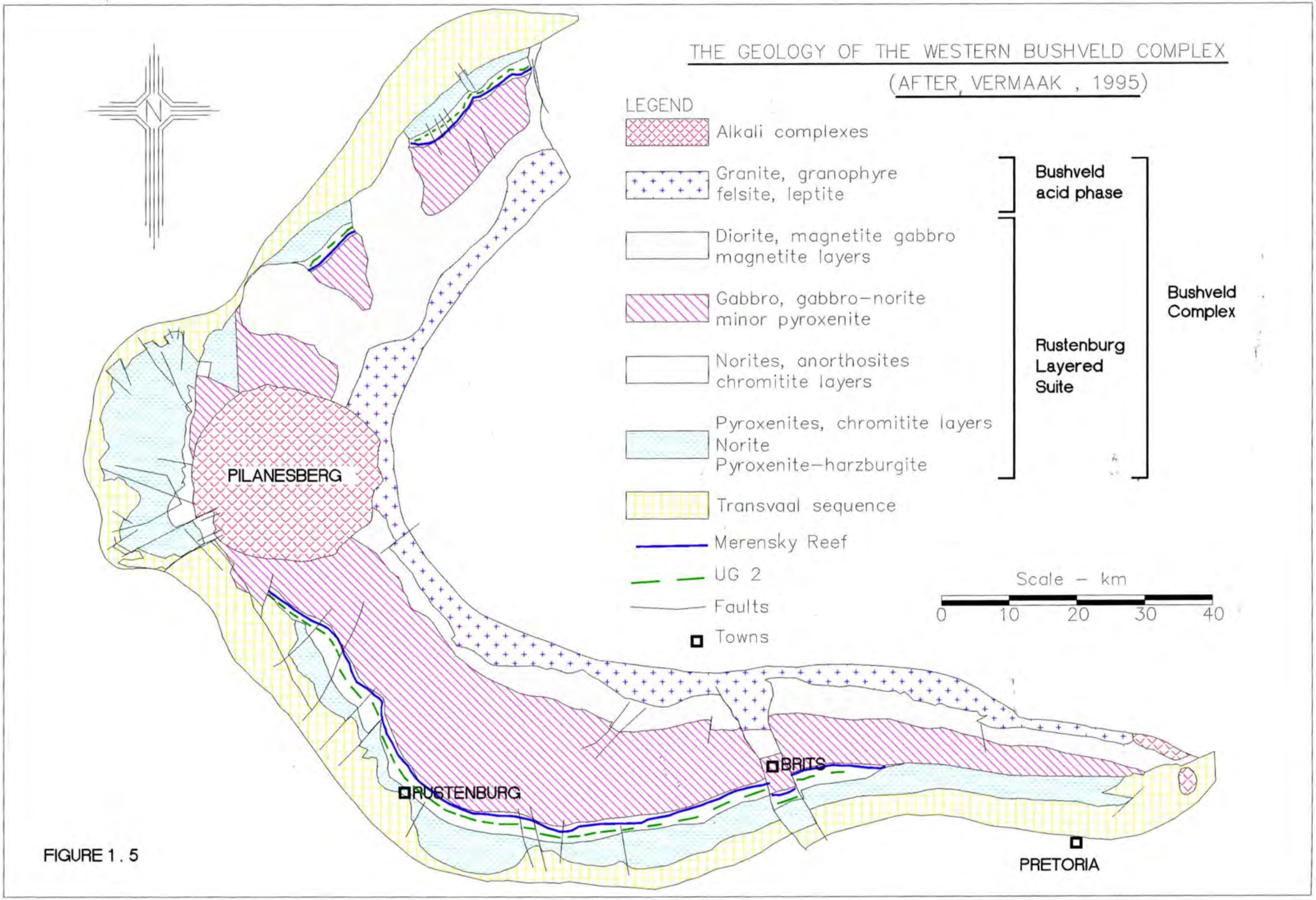

THE GEOLOGY OF THE WESTERN BUSHVELD COMPLEX
(AFTER, VERMAAK , 1995)
LEGEND
Alkali complexes
Granite, granophyre felsite, leptite
Bushveld acid phase
Diorite, magnetite gabbro magnetite layers
Gabbro, gabbro-norite minor pyroxenite
Norites, anorthosites chromitite layers
Pyroxenites, chromitite layers Norite Pyroxenite-harzburgite
Rustenburg Layered Suite
Bushveld Complex
Transvaal sequence
Merensky Reef
UG 2
Faults
Towns
Scale - km
0
10
20
30
40
N
PILANESBERG
RUSTENBURG
BRITS
PRETORIA

FIGURE 1 . 5

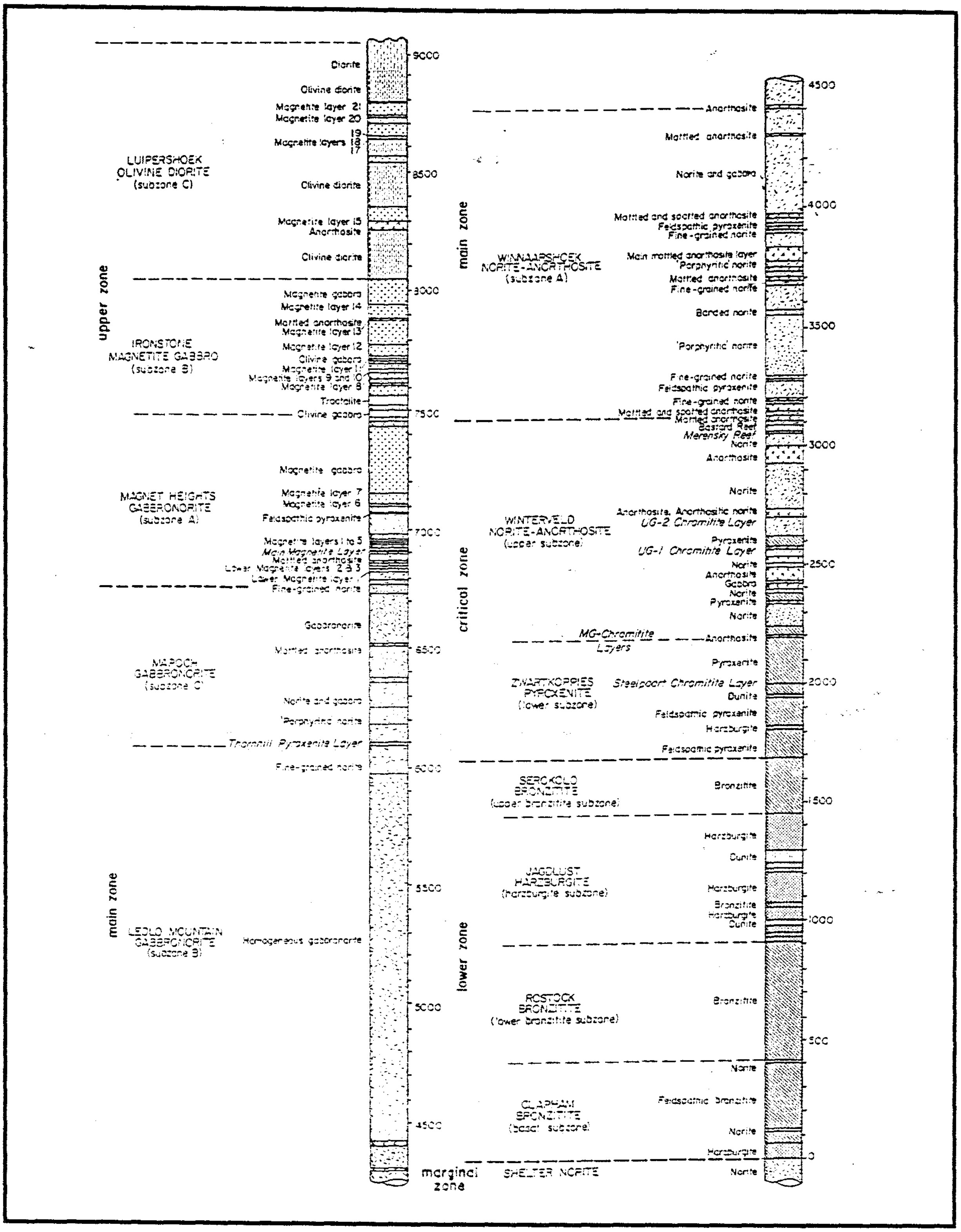


FIGURE 1.6

Lithostratigraphic Column of the Rustenburg Layered Suite

in the Eastern Bushveld Complex

(After Vermaak & Von Gruenewaldt, 1986)

However, the transition from LZ to lCZ in the eastern BC is marked by ±30 m of bronzitite containing layers of feldspathic bronzitite (Cameron, 1978). Generally, the first appearance of cumulate chromite defines the base of the lCZ.

1.10.3 The Critical Zone (CZ)

The Critical Zone (CZ) is subdivided into two subzones, namely the lower CZ (lCZ) and the upper CZ (uCZ) which correspond with the Zwartkoppies Pyroxenite and the Wintersveld Norite-Anorthosite respectively (Table 1.4). Traditionally the boundary between these two subzones has been taken to be the first appearance of cumulus plagioclase and chromite. However, it has been demonstrated (Teigler *et al.*, 1992) that this is not valid in the south-westward extension of the western lobe of the BC. In that part of the lobe abundant cumulus plagioclase is present in the lCZ. It has been suggested by Eales *et al.*, (1993) that either the norite or anorthosite layer occurring between the MG2 and MG3 chromitite layers would less ambiguously define the boundary between the lCZ and the uCZ.

1.10.3.1 Lower Critical Zone (lCZ)

The lithologies of the lCZ are feldspathic pyroxenites, harzburzites, minor dunites and chromitites (Figure 1.6). The chromitite layers vary in thickness from <1cm to the ± 1,2 m thick LG6 Chromitite layer. In general, the succession of the lCZ lacks the cyclical crystallization pattern present in the Lower Zone. Furthermore, the presence of dunites and harzburgites (present in places in the eastern BC) in the middle of the succession indicate reversals in the normal fractionation trend. The majority of chromitite layers are interlayered with pyroxenites or anorthosites and less commonly with norites, in the eastern sector of the BC (Cameron & Desborough, 1964).

The base of the lCZ has been re-defined by Eales *et al.*, (1990) for the western sector of the BC, as being the base of the thick package of orthopyroxenites in which the LG Chromitites are found. Traditionally the uCZ/lCZ boundary has been taken to be the first appearance of cumulus plagioclase.

1.10.3.2 Upper Critical Zone

The upper Critical Zone is marked by; the reappearance of olivine (after an absence of ±500m) in the ultramafic rocks; by the appearance of chromitite layers with sharp bottom contacts which are interlayered with anorthosites and/or anorthositic rocks (Figure 1.6); the most complete cyclic units known in the BC and the most evolved chromitites in the whole succession which are also sulfide bearing, and frequently platiniferous. The top of the uCZ is currently under debate, with some workers preferring the upper Giant Mottled Anorthosite (de Klerk, 1991; Eales *et al.*, 1993) and others placing it at the base of the Merensky Reef (Kruger, 1990). However, most workers prefer the use of the former, as the latter relies on the absence of the Giant Mottled Anorthosite in conjunction with the unconformity at the base of the Merensky Reef. The uCZ contains the Merensky Reef, Platreef and UG2 layers; the main platiniferous ore bodies of the BC. For more detailed description of the cumulate succession of the uCZ, as well as geochemical trends and their interpretation the reader is referred to Teigler *et al.* (1992), Eales *et al.* (1994) and Maier & Eales (1994).

1.10.3.2.1 The UG2 Chromitite Layer:

The UG2 is one of three important chromitite layers that lie within the uCZ. The other significant chromitite layers are the UG1 and a third chromitite layer is present in the eastern BC, designated the UG3 Chromitite (McLaren & De Villiers, 1982). The UG2 is the only one of these three chromitites that contains economic PGM mineralization. Apart from breaks created by the "Northern Gaps" north of the Pilanesberg and the Brits graben, where the entire RLS is disturbed by faulting, the UG2 in the western Bushveld Complex is traceable for extensive strike distances, of 50km or more. The UG2 in the eastern Bushveld Complex is similarly traceable along almost the entire strike length, except where it is interrupted by the Steelpoort and Wonderkop faults

The UG2 represents an incomplete cyclic unit, and the PGE-mineralization is mainly confined to the chromitite and the top of the underlying footwall unit. Although the UG2 shows enrichment of sulfides over its enclosing layers, its tenor is usually quite low, an order of magnitude lower than the Merensky Reef. In the north-west sector of the western lobe of the BC, the Merensky Reef and the UG2 are separated by between ±15m and ±370m of intervening cumulates (Table 1.5). The UG2 varies in thickness between ±15cm and ±255cm and dips towards the centre of the complex at between 5° and 70° (Table 1.5).

Table 1.5

	Western Bushveld Complex		Eastern Bushveld Complex		
	North	**South**	**Northwest**	**Central**	**South**
Thickness (cm)	90 to 155	15 to 255	?	60 to 140	90 to 220
UG2-MRPX (m)	15 to 150	85 to 220	±240	305 to 370	±145
Dip	15° to 30°	8° to 28°	55° to 70°	5° to 25°	6° to 45°

Geologic Data on the UG2 Chromitite Layer

(After McLaren & De Villiers, 1982)

The UG2 lies at the base of a ±10m-thick feldspathic pyroxenite layer within the anorthositic subzone of the upper Critical Zone (Winterveld Norite-Anorthosite). The UG2 can be massive or it can contain intercalated layers of pyroxenite, increasing the overall thickness of the unit to up to 5.7m (McLaren & De Villiers, 1982). In the southeastern sector of the western Bushveld Complex these intercalations can also be anorthosite or norite rather than pyroxenite.

The UG2 is composed of chromite (60-90% by volume), orthopyroxene (5-25%), plagioclase (5-15%) as well as accessory amounts of other minerals, of which the more important are clinopyroxene, base metal and other sulfides, platinum-group minerals, ilmenite and magnetite (McLaren & De Villiers, 1982). The UG2 frequently has a mottled appearance due to the presence of large (2-5cm) poikilitic bronzite crystals enclosing numerous small chromite crystals (Leeb-du Toit, 1986).

The average PGM+Au tenor of the UG2 is 7.0 g/t. There is a lateral as well as a vertical variation in both the tenor and chemistry of the PGM mineralization in the UG2. It invariably shows enrichment of PGM+Au at the bottom contact and usually has one or more PGM peaks in the middle or towards the top of the layer (McLaren & De Villiers, 1982). The PGM grade in the bottom contact can frequently exceed 20 g/t and high grades are present for several centimetres into the footwall, where the footwall is a pegmatoid. This is an important consideration in open cast mining as it is imperative that the bottom contact and some footwall be extracted to maximise the grade of the ore. However, the amount of footwall that is mineralized is highly variable and requires careful control. The majority of the PGM occurs along grain boundaries or associated with interstitial sulfides with a small proportion enclosed in chromite or silicates (McLaren & De Villiers, 1982; von Gruenewaldt, 1979).

The UG2 is usually directly underlain by a pegmatoidal feldspathic pyroxenite, often indistinguishable from Merensky pegmatoid (apart from the absence of visible sulfides), locally known as the UG2 Pegmatoid. This contact is usually very gradational and undulose, with mineralization extending into the pegmatoid for a variable distance. However, at Western Platinum Ltd. (WPL) and Karee Mine, the UG2 is underlain over extensive areas by an anorthosite (mottled), which is very distinctive and probably represents the primary footwall rock type. The footwall contact of the UG2 and anorthosite is sharp, though undulose, and mineralization is rarely present in the anorthosite footwall layer. On the perimeter of this area the incipient replacement of this anorthositic footwall has been observed, where a rapid change in footwall rock type occurs (often over less than 3m) from anorthosite to pegmatoid. On EPL, extensive areas of UG2 overlies a rhythmically layered norite. The contact between the UG2 and the footwall is a disconformity. In these areas the footwall layering is truncated by the UG2. It is believed that this contact is erosional, and represents the effects of the surge of new magma into the compartment. The absence of the UG2 pegmatoid is interpreted by Leeb-du Toit (*op cit.*) to be indicative of potholing. This is, however, not the case on EPL.

The UG2 contains the highest concentration of sulfides of any of the chromitite layers of the BC. The base metal sulfides are mainly pentlandite, pyrrhotite, pyrite and chalcopyrite and there are a host of noble metal sulfides, many of which are unnamed, as well as intermetalloids and solid solutions (electrum) (von Gruenewaldt, 1979).

The UG2 usually has a clear, sharp top contact and is overlain by a medium grained feldspathic pyroxenite. Within this hanging wall unit are usually present a series of chromite layers, known variously as the "Leaders", "Triplets" or "Marker". The distance from the top contact of the UG2 to these chromitite layers is highly variable over short distances and they are frequently seen to bifurcate from the main chromitite layer and rejoin it. Above the feldspathic pyroxenite the UG2 cyclic unit grades into a norite, followed by anorthosites

(Figure 1.6). On EPL the UG2 hanging wall pyroxenite is overlain by a thick sequence (±20m) of spotted and mottled anorthosites.

There is a substantial amount of descriptive work published on the UG2 in the western Bushveld Complex to which the reader is referred; Viljoen *et al.* (1986a); Viljoen *et al.* (1986b); Leeb-du Toit, (1986); Viljoen & Hieber, (1986) and Farquhar, (1986). Mossom, (1986) and Gain (1980 & 1985) provide descriptions of the UG2 in the eastern Bushveld Complex. The mineralization of the UG2 is discussed by Naldrett (1989) and McLaren & De Villiers *(op. cit.)*

Like the Merensky, the UG2 displays slumping structures known as potholes which affect the normal elevation of the UG2 due to "removal" of a portion of the footwall stratigraphy. These features are interpreted to be due to layer-parallel extensional slip (Carr *et al.*, 1994). Potholes are disruptive to mining and can cause considerable losses of mineable ore. They are known to occur also in the Merensky, where their origin and mode of formation has been debated since the late 1950's, in the LG and MG chromitite layers as well as the magnetite layers of the Main Zone. They are also described in the Stillwater Layered Complex (Mooney, 1995).

1.10.3.2.2 The Merensky Reef:

The Merensky occurs within a differentially graded cycle known as the Merensky Cyclic unit. The term Merensky Reef is unfortunate as it simplifies what is a very complex ore body. It is not a "reef" in any sense. Von Gruenewaldt (1979) expanded Vermaak & Hendriks' (1976) definition of the Merensky Reef as the basal pyroxenitic portion of the Merensky cyclic unit, including the porphyritic pyroxenite, the pegmatoidal pyroxenite and the associated chromitite stringers that may be developed. This essentially defines the Merensky Reef as that portion of the Merensky cyclic unit that hosts the PGM mineralization and it shows great lateral variation in style. It ranges from a thin (<50cm) pegmatoidal feldspathic pyroxenite, bounded top and bottom by thin chromitite layers, to a medium grained feldspathic pyroxenite 5m to 7m thick. Several distinct facies of Merensky Reef are recognized (Vermaak, 1976; Davey, 1993).

Generally, the Merensky "Reef" is used to describe that portion of the Merensky pyroxenite that is economically important. The economically important portion of the Merensky is invariably associated with one or more thin chromitite layers and the development of at least some pegmatoidal material. How well this pegmatoid is developed can vary greatly, both on the scale of the lobe and on the scale of a mine, or even an individual stope. Peak PGM values are invariably associated with the chromitite layers. It appears that the PGM tenor, measured in terms of grams per unit area, is surprisingly constant throughout the western BC. This being the case, the thinner the Merensky "Reef", the higher the grade per ton. The PGM mineralization in the Merensky is essentially cryptic. Where the Merensky pyroxenite is thin, the footwall rocks are often mineralized to economic grade. The mineralization appears to migrate upwards, becoming more diffuse, as the Merensky becomes progressively thicker towards the east. Despite this, peak PGM grades are still associated with the basal chromitite layer, where it is present.

The hanging wall rocks of the Merensky Reef are essentially anorthosites and overlying these is the Bastard Merensky Pyroxenite (Figure 1.6), which in places closely resembles the Merensky pyroxenite but often presents as a leuconorite. The Bastard Merensky is the last cyclic unit of the uCZ. The footwall sequence of the Merensky consists of a series of anorthosites and norites (Figure 1.6). The layering is well developed and quite distinct in these rocks, with individual layers ("markers") traceable over more than 50km along strike.

1.10.4 The Main Zone (MZ)

The base of the Main Zone is generally accepted as being the top of the Giant Mottled Anorthosite (Figure 1.6). Mineralogically it is marked by the loss of olivine and chromite and the appearance of augite and inverted-pigeonite. Lithologically it consists of norites and gabbro-norites but has anorthositic layers close to its base and at the top of the MZ. It is subdivided into 3 subzones. Pyroxenites are rare but present. The main pyroxenite "Marker" is used to mark the transition from subzone B to C (Leolo Mountain Gabbronorite to the Mapoch Gabbronorite, Table 1.4). This pyroxenite is significant in that it marks the rejuvenation of the whole-rock compositions, indicating a possible influx of new magma.

1.10.5 The Upper Zone (UZ)

The Upper Zone is defined mineralogically by the first appearance of magnetite as a cumulus phase, above the pyroxenite marker. Lithologically it consists of a series of magnetite layers, magnetite gabbros, anorthosites and ferrodiorites. Once again, the UZ is subdivided into 3 subzones A, B and C (Table 1.4). The base of subzone B (Ironstone Magnetite Gabbro) is defined by the reappearance of olivine and that of subzone C (Luipershoek Olivine Diorite), by the appearance of apatite, as cumulus phases. The UZ is ± 2000m thick (Figure 1.6) and contains economically important magnetite layers.

The following chapters will examine the history and methodology of open cast mining of the UG2 and compare the economics of open cast mining with underground mining as well as the metallurgical problems that must be overcome to profitably treat shallow UG2 ore.

Chapter 2

Chapter 2

2.0 Open Cast Mining for PGM's in the Bushveld Complex.

2.1 Background:

Although it has long been known that the chromitite layers of the Bushveld contain PGM's (Hall & Humphrey, 1908), few attempts were made to work them for their PGM content until the early 1980's, when Lonrho's Western Platinum Mine began mining the UG2. This was made possible by the development of metallurgical processes which overcame problems in this area that had previously marshalled against the extraction of PGM's from chromitites. A secondary factor was the comparative grades of the UG2 and the Merensky Reef. At Western Platinum, the UG2 grade is greater than that of the Merensky, and therein lay the attraction and driving force to overcome these metallurgical problems. Because of the greater areal extent of the UG2 compared to the Merensky Reef, the ability to exploit the UG2 would, and has, more than doubled the South African PGM resources (Mintek, Application Report No.1).

2.1.1 History of Open Cast Mining of the UG2 and Merensky Reef in the Bushveld Complex:

The first known open cast mining operation on the UG2 in the Bushveld Complex was at the Crocodile River Mine (CRM) No. 1 Pit, near Brits, which began operation in 1989 (Figure 2.1). This was followed by two more open cast operations on the farm Buffelsfontein, immediately south of EPL, and then one on the Latilla property, immediately to the east of EPL (Figure 2.1). The reason for open cast mining at the Crocodile River Mine, at that time, forms a recurring theme throughout the western Bushveld mines: a shortage of tonnage to keep the mill full because of production problems underground. CRM attempted to mine by trackless mining methods which were not as successful as they might have been, causing a slower build up of tonnage from underground than planned, and so their production fell behind and their mill was underutilized.

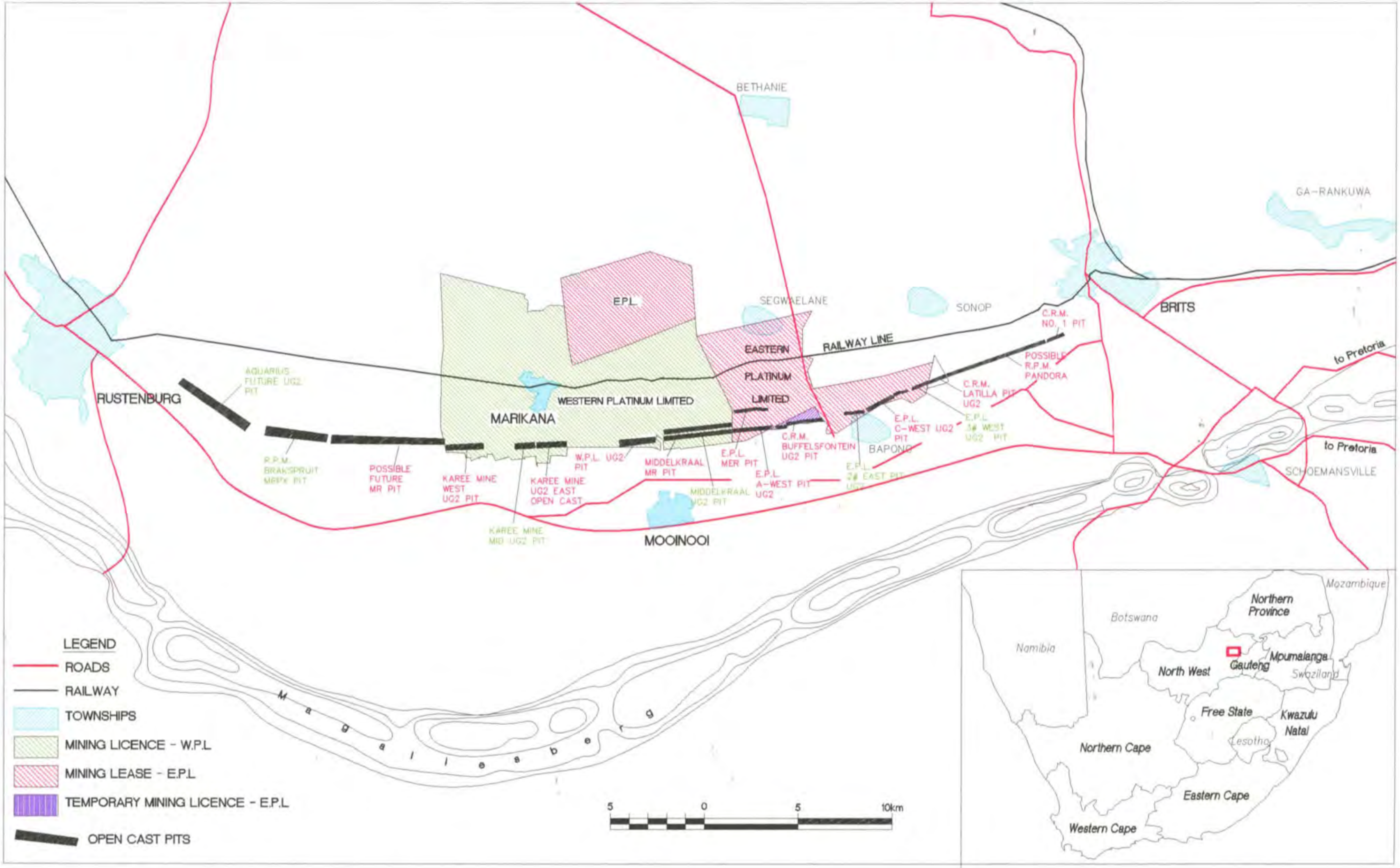


FIGURE 2.1

LOCATION PLAN OF OPEN CAST MINING OPERATIONS IN THE SOUTH - EASTERN PORTION OF THE WESTERN BUSHVELD COMPLEX

Underground development was stopped upon the purchase of the mine by Impala, and reserves were worked out. The mill throughput was maintained at a constant level of 160 000t/month by utilizing open cast ore (Brits, *pers comm.*, 1996). This was further supplemented by the purchase of shallow (0-8m) open cast ore from EPL once EPL began open cast mining operations in 1990 (see Chapter 5).

EPL began open cast operations (A-West Pit: Figure 2.1) in 1990 to supplement ore coming from underground. The reasons for open cast mining at that point in time were twofold: production problems underground providing a slower tonnage build up than had been planned or expected, and secondly, during the latter part of 1990, it became evident that the second mill stream, then under construction, would be underutilized. This led directly to the decision to proceed with a second open cast operation (C-West, Figure 2.1). During 1993, a similar situation prevailed on Karee Mine (WPL), which foresaw a drop in underground tonnage due to production problems related to encountering a large replacement pegmatoid body at No. 3 Shaft. At that stage a small UG2 open cast pit was mined to supplement tonnage.

In 1993, Western Platinum Mine too began a small UG2 open cast pit, once again in response to production problems underground. Unfortunately, this pit was not completed at the time, due to the fact that the contract mining company went into liquidation about half way through mining. This pit has subsequently been reopened and mining completed, late in 1996, in an effort to build a stockpile for the 1996 Christmas period, which traditionally suffers from a shortage of tonnage to mill due to mine closure over this period.

Until recently, no one seems to have examined the possibility of mining the Merensky Reef by open cast methods, in response to the problems cited above.

During 1993, RPM brought the PPRust Mine, on the Platreef, into production. It is believed that RPM researched for some 15 years to overcome metallurgical problems caused by the complex mineralogy of the Platreef. While this has nothing to do with the UG2, and its complex metallurgical problems, it is believed that this, along with the industrial problems that Amplats have been suffering, gave the impetus to RPM to examine the available Merensky reserves as potential open cast operations.

In 1995, RPM began open cast mining operations on a ± 4 km long Merensky pit, at Bleskop (Figure 2.1) which lies to the south of the Bleskop Shaft, near Rustenburg. The Merensky Reef in this area is "classic" RPM type Merensky Reef. It is a pegmatoidal feldspathic pyroxenite, between 60cm and 70cm in thickness. The average grade in the open cast operational area is in excess of 5.0 g/t, probably ±8.0 g/t (Harrison, *pers. comm.*, 1996). The mining rate there is in the region of 50 000t-60 000t per month. The Merensky Reef has a "frozen" bottom contact, which is removed by hand if necessary, due to the high grade reported along that contact. Ore is loaded on the basis of a visual inspection (of sulfides?), although the grade control is reported to be good (Harrison, *pers. comm.*, 1996). Mining costs are reported to be R44/t, down to a depth of 25 metres below surface, and the mining is contracted out.

Although metallurgical recoveries are reported to be as high as 64% (Harrison, *pers. comm.*, 1996), it is believed that this figure is based upon bench test results. It is highly likely that these bench tests involved at least some of the metallurgical processes/procedures developed for the Platreef at PPRust. The ore is transported by rail to the main RPM milling and processing plant where the open cast material is blended with underground Merensky ore. There appears to be no attempt to batch-mill or campaign the open cast ore separately, and it is therefore felt unlikely that RPM are indeed obtaining such high recoveries in practice.

An inspection of the Bleskop pit revealed the presence of deep weathering down to ± 20m below surface, with the Merensky Reef displaying extensive oxidation and pervasive development of clay minerals. Furthermore, due to the fact that is "policy" to mine through potholes and dykes, it is felt that the dilution is underestimated and that the mill-feed grade of ore sent to the mill must be substantially less than the 5.0 g/t quoted. Additionally, as this material is blended with underground ore, it is impossible to determine what the recovery really is.

The 1995/1996 period was a particularly bad period for production throughout the mines in the western Bushveld (see Chapter 1), with all three of the major producers suffering problems of low productivity and strikes on the part of their workforce. This created a "window of opportunity" to open cast shallow reserves, previously ignored or thought to be uneconomic.

In 1996, EPL began open cast operations, once again, on the UG2 (see Chapter 5), as has Karee Mine (Figure 2.1). An ongoing project is underway to examine the possibility of open cast mining of Merensky reef on Western Platinum Mine (Figure 2.1).

Amplats, hard hit by industrial unrest and obviously convinced of the success of the Bleskop pit, have initiated a number of open cast projects on both the UG2 and Merensky on several of their properties during 1996. A Merensky and a UG2 pit recently began production at Boschfontein, just south of Impala near Rustenburg. Union section has also started a UG2 pit. Unfortunately, little information is available about any of these pits, but Johnson Matthey (1996) estimate that some 5% of mined tonnage is being sourced from open cast mining at present, and this will probably expand over the coming year in view of the depressed metal prices.

Impala is also known to be examining the possibility of open cast mining of shallow Merensky reserves. Unfortunately Impala have built their main milling and processing plant on top of the largest stretch of UG2 open-castable shallow reserves.

2.2 Comparison of the Economics of Open Cast Mining with Underground Mining:

2.2.1 Background:

In this section the costs of mining open cast ore and underground ore are compared within the context of the Lonrho Platinum Division (LPD) operations. The open cast costs are based on several quotations obtained and actual working costs for both proposed Merensky and working UG2 open cast operations. All quotations were obtained on a 'cost-per-ton-of-ore-delivered' basis. This, it is believed, is the most cost effective method managing and operating an open cast pit. The alternative, favoured by some contractors, is for the client to pay for each individual task within the scope of work; topsoil stripping, soft overburden removal, hard overburden removal, extraction of ore and sweepings, followed by land reinstatement and rehabilitation. Each of these will be billable at a different rate and will therefore require careful measurement. This measurement can become an onerous and acrimonious task, causing continued conflict between the contractor and the client. It requires constant supervision and the skills of surveyors and accountants, which in itself represents a cost (often hidden) that should be borne by the open cast operation, further increasing costs. These types of contracts often even cover the contractor for standing time during inclement weather, and generally favour the contractor over the client.

Due to the current competitive conditions in the civil engineering/open cast mining industry it is possible to negotiate a mutually acceptable contract, wherein the contractor accepts some risk. This risk, it has been found, means that the contractor performs the mining more conscientiously. As the contractor is paid per ton of ore, little ore tends to be left in the pit. The risk to the client in this is of course that waste will be delivered as ore, to increase the tonnage. However, this can be catered for in the contract, where maximum acceptable dilution factors or even the grade of the delivered ore can be specified. Non-conformance to these conditions will mean non-payment.

Open cast mining in the western Bushveld, of either the UG2 or the Merensky, costs between R35 and R60/ton of ore. The Merensky is generally more costly for a given stoping width due to its lower specific gravity (3.2) than the UG2 (3.9). The UG2 provides more tons per unit area over the same stoping width and is therefore inevitably cheaper to mine than the Merensky to any given pit depth and ore zone thickness. The economic mining depth of an open cast operation is a function of a number of factors: grade, ore zone thickness, concentrator recovery, metal prices and mining costs are the most important of these. Break-even depth for the UG2 pit at EPL is approximately 40m. Cut-off depths will also vary as a function of time, which affects some of the factors cited above.

The cost of underground production falls in the range of R90 to R110/ton of UG2 ore at EPL. As with any operating mine, a large proportion of this cost is fixed, probably in the region of 60%-70%. As labour costs account for more than 50% of the total costs, the only real opportunity that exists to reduce the overall cost/t is that of increased productivity, a matter that is currently being addressed.

Milling and flotation costs for both underground and open cast ores are very similar. However, if the plant is kept full, the cost of both is somewhat reduced. The reason for this is the very high fixed cost at the plant and economies of scale. With the plant full, the milling and flotation cost per ton is $\pm$R17.50.

Smelting, on the other hand, is more expensive for concentrate produced from the open cast ore. This is due to the lower grade of concentrate produced from the open cast ore (see Chapter 3). It therefore provides fewer kilograms per ton of concentrate. However, despite the fact that the open cast concentrate can be half the grade of underground concentrate, its smelting costs are only marginally higher, at ±R6.50/t of ore, than underground ore smelting costs which is ±R6.00/t of ore. Notice that the smelting costs are quoted per ton of ore and not per ton of concentrate. The reason for the small differential in smelting costs between the two concentrates is more to do with accounting and billing procedures applied by the LPD smelters. Costs are calculated on a "tons of concentrate and contained kilograms of PGM" basis and back calculated to tons milled. The open cast concentrate makes up only a small proportion of the total concentrate dispatched and its effect is therefore greatly "diluted".

2.3 Mining Methods Employed in Open Cast Mining of the UG2:

The mining method employed by a variety of contractors on the UG2 is remarkably uniform throughout all the known pits in the western Bushveld. Open cast mining differs from open pit mining in that the spoil, or waste rock, is returned to the pit after the removal of the ore.

The open cast method employed in the western Bushveld, known as "convection mining" (because of the way material is moved from the working faces back into the backfill area), is essentially an adaptation of coal open cast mining methods, employed in the Witbank area. It is also the preferred method employed by many of the chromitite mines in the western Bushveld. The main difference since the UG2 chromitite dips at an angle of between 8° and 20° whereas most exploited coal deposits have a considerably lesser dip.

2.3.1 Mining Methodology:

"Convection mining" is especially well suited to the requirements of small operations. The method is efficient and incorporates land rehabilitation as an integral part of the mining cycle. The method was developed by Minesa Management Services (Pty) Ltd. in the early 1980's for the Goedenhoop Colliery, near Hendrina in the Mpumalanga Province. Interestingly, the method was developed from first principles, and implemented by a geologist (Patterson, 1982). The method has a number of important advantages over other methods:

- Its costs are significantly lower than other methods.
- It provides adjustable open pit limits.
- It limits highwall exposure.
- It limits waste handling
- It provides constant stripping ratios (depending on topography).
- It provides the opportunity for flexible grade control.
- It optimizes equipment utilization.
- It provides immediate land re-instatement.

The convection mining method was developed for flat lying coal seams with an overlying, sloping topography. The geometry of this is very similar to that of a shallowly dipping, tabular ore body with an overlying, flat topography, such as is the case in much of the Western Bushveld Complex. For this reason convection mining seems to be the preferred mining method employed in the small open cast operations there.

The mining strips are laid out parallel to the dip of the ore body, which provides two immediate advantages:

- The profile of each successive strip is similar to that of the first, maintaining a more-or-less constant stripping ratio, and hence constant costs.
- The highwall position of each strip can be moved (to some extent), either up-dip or down-dip. This means that the open pit limits and hence the unit mining cost can be adjusted, if required, in response to price fluctuations of the material being mined or other criteria.

Furthermore, as the exposure of the highwall is limited at any moment in time, pit operating safety is considerably improved (Patterson, 1982).

Another advantage of this mining method is that, where there is a distinct change in grade of the material being mined, and if the box-cut is placed centrally, the mining sequence can be planned so that the two resulting faces can retreat away from the box-cut. In this way the material from both faces can be blended to provide a fairly constant run-of-mine (ROM) grade material (Patterson, 1982). This has been done successfully in coal mines, but is not known if this advantage has been exploited in the western Bushveld. It is possible that doing so may confer some benefit when mining the Merensky Reef. A major advantage of this method, especially in view of the ever more stringent environmental legislation and public awareness of environmental issues, is that it allows almost immediate reinstatement of land to take place within the operating pit. The mining cycle operates in four phases, illustrated in figures 2.2 to 2.5.

Phase 1: The topsoil is removed and stockpiled on either the highwall or outcrop side of the pit (Figure 2.2). Due to the low specific gravity of soil, compared to most rock types, it is generally preferred not to move the topsoil too far from its original position. The topsoil may be removed from only a small area of the pit at one time, in which case it can be replaced once that section has been mined out and that area back filled, or, if the pit is small, from the whole pit at once.

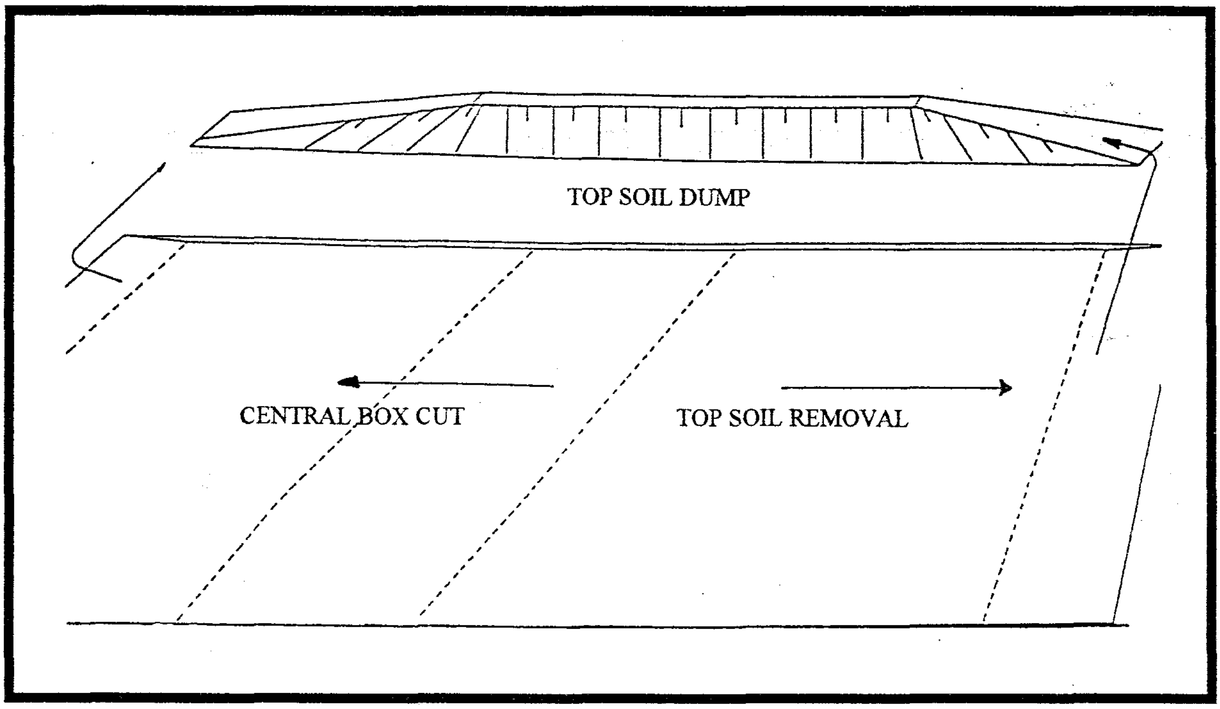


FIGURE 2.2

Schematic Representation of Convection Mining Method: Phase 1: Topsoil Removal
Note position of box-cut, central to the pit.

Some soil types may become essentially sterile if they are stockpiled for long periods of time. This may, in some cases, be a consideration in deciding how much topsoil to strip at once. In the western Bushveld, the most common soil type is black turf cotton soil. Experience has shown (at Crocodile River Mine, EPL, Western Platinum Mine and at Karee Mine) that this soil remains fertile for at least two years, and therefore requires little re-vegetation or treatment once put back in place. Therefore large areas of topsoil tend to be stripped at once.

The soil moved first is that overlying the box-cut area. The box-cut dimensions are dictated by the size of the machinery used. Some 40t trucks require as much as 50m within which to manoeuvre. The positioning of the box-cut may or may not be critical. A central box-cut (placed centrally along the strike length of the pit) provides more flexibility to both the mining contractor and to the client mining company. The drawback is that a central box-cut needs to be almost twice as large as one placed at either end of the pit to provide sufficient space to mine two faces simultaneously.

This has cost and cash-flow implications to either the client or the contractor, depending on the structure of the contract. If the contractor is paid on a "per ton delivered" basis, it doubles the cost of the waste removal that he must carry, until the ore is removed. If the contractor is paid for all work performed, then the client must bear the cost of a large volume of waste removal, before any ore is produced. However, it does, or may, provide benefits for both parties, as it provides the flexibility of having two faces. Where geological losses occur, such as potholes or dykes, it is likely to affect only one face and thus effecting only half of the production.

Phase 2: The waste rock overlying the ore body in the box-cut is removed by conventional blast, load-haul-dump operations and placed on the spoil dumps, either at the opposite end of the pit where the box-cut is placed at one end of the pit, or at either end of the pit where the box-cut is centrally placed. Obviously the mechanics of this vary greatly, depending on how fresh or weathered or fresh the rock is, and the type and size of equipment being used. If the rock is fresh, drilling and blasting may be necessary. Where this is done, it usually follows a pattern of 5m benches, a factor which is dictated to large extent by the reach of the equipment being used.

Throw blasting is not employed on the UG2 pits in the western Bushveld, which is another departure from the coal open cast method. The reasons for this is that the top contact of the

UG2 tends to be quite undulose and blasting would create a loss of technical control as in places part of the UG2 would be removed and trucked as spoil, while in other places waste material would be left in place and removed with reef, causing dilution. If even 20cm of the UG2 was lost through throw blasting, too much UG2 tonnage would be lost and the financial performance of the pit would be affected deleteriously.

The amount of spoil thus stockpiled must be sufficient to refill the final void. The positioning of these spoil dumps is supercritical, as it is the usual practice to bulldoze this spoil back into the final void. An error of only 20m in placing these spoil dumps can seriously effect the costs to the contractor. The reasons for this are that bulldozing is only cost effective over short distances. If the spoil dumps are wrongly placed, the material will have to be trucked back into the void, a much more costly exercise.

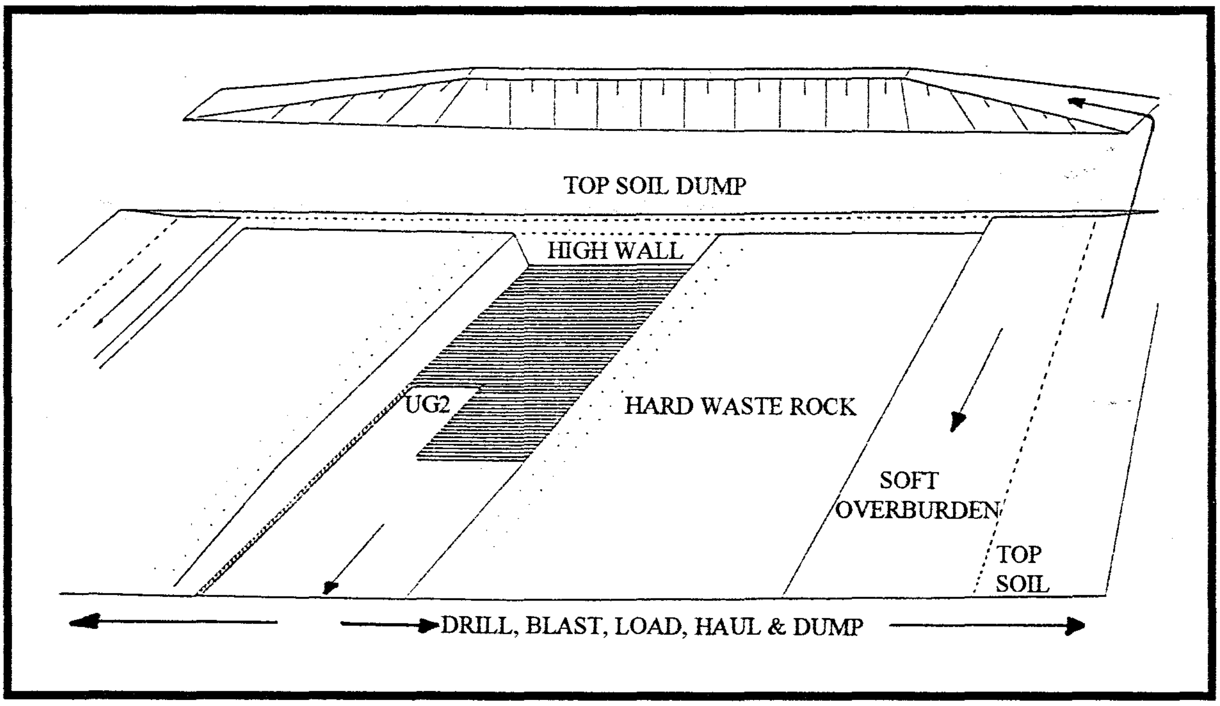


FIGURE 2.3

Schematic Representation of Convection Mining Method: Phase 2: Central Box-Cut
Removal of hanging wall waste followed by extraction of UG2.

Once sufficient waste rock is removed, the extraction of ore (Reefing) can commence by conventional blast, load-haul-dump methods. The first step in this process is to "sweep" the top contact of the UG2. This can involve manual sweeping with brooms, or mechanical cleaning of as much waste as is practical off the top contact of the UG2. This essentially is an aspect of grade control, minimizing the contamination of the ore.

Obviously the amount of blasting required depends on the degree of weathering that the UG2 has undergone, and to what depth. Both of these factors can be quite variable over short distances. To some extent it will also be determined by the size and type of the equipment being used in the pit. The larger the equipment the more ore can be "ripped" by excavators, hydraulic shovels, etc., depending on their breakout power. In places, UG2 only 2-3m below surface may have to be blasted and conversely, UG2 at 20m depth may be "rippable". The intrinsic strength of the rock is also major factor in determining the need to blast.

Blasting the UG2 has an effect in three major areas: firstly, the quality of the ore removed in terms of fine and coarse ore. The more fines that are produced the more ore will be lost through transport, handling, etc.. Secondly, the environmental considerations and impact of blasting, particularly near built-up/residential areas needs be kept to a minimum, and thirdly, cost. Blasting is expensive and increases the cost per ton. The scale of this is determined by the thickness of the ore. There is also a hidden cost to blasting, from the contractors point of view, in that it prevents work being carried out in close proximity to the area to be blasted for several hours (or even days, depending on the size of the area to be blasted) before the blast.

Phase 3: The waste overlying the ore body in the strip immediately adjoining the box-cut is removed and placed back in the box-cut, once the ore has been removed from it. This material is then compacted if necessary, and levelled. The mining process is sequential and the face(s) retreat from the original box-cut.

This phase demonstrates the main advantages of this mining method, which is that of the lack of double-handling of spoil. With the obvious exception of spoil from the box-cut, the spoil is handled only once. In large pits, this can also be applied to the removal and re-placement of topsoil, obviating the need to double-handle topsoil, and thus further reduce costs.

The lowermost part of this backfill is transported within the pit, and only when the upper portion of the backfill is placed do the load-haul trucks need to leave the pit. This reduces the travel distance and the cycle time per load, which assists in keeping the mining costs low.

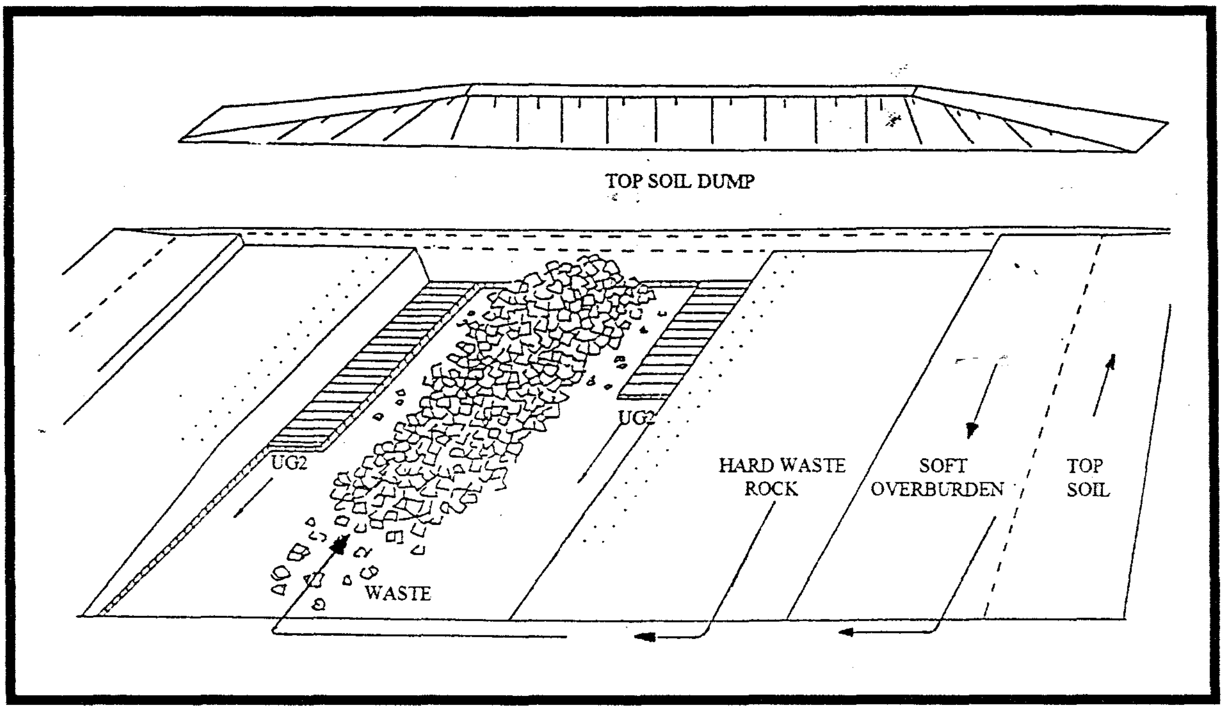


FIGURE 2.4

Schematic Representation of Convection Mining Method: Phase 3: Backfill Commences
Removal of waste from areas adjoining box-cut placed in box-cut.

Phase 4: Once the backfill is back in place and compacted, the topsoil can be replaced on top of the waste rock in the box-cut. The distance from the working face, at that stage, can be quite small. In the case of EPL, topsoil replacement is ±200m distant from the working faces. Final rehabilitation can now begin.

The replaced topsoil thickness is usually prescribed by the EMPR and levelling may be important. Too steep a slope will cause erosion, and further future expense in rectifying this. The mine remains responsible for the rehabilitated area in perpetuity and it is therefore worthwhile to ensure that the possibility of erosion is minimized.

On small scale pits, such as these, land revegetation can take place after mining has been completed and the whole pit area rehabilitated. On larger pits, revegetation may be an integral part of the mining and rehabilitation sequence. This is often the case on large open cast operations, for example the Richards Bay mineral sands mine on the Natal coast.

Depending on the mining schedule, this means that within a month or two of the extraction of the ore from the box-cut, land re-instatement and rehabilitation will be taking place. In the case of large pits (in terms of strike length), revegetation can commence soon after the extraction of ore has been completed in the box-cut.

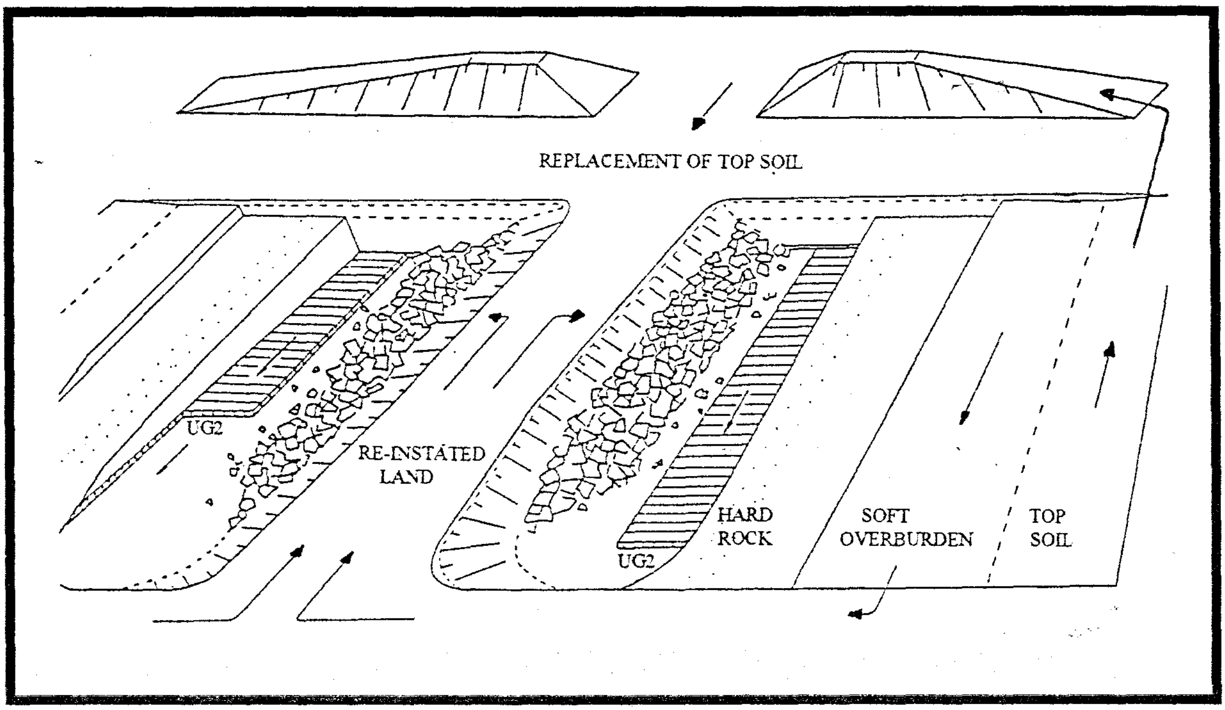


FIGURE 2.5

Schematic Representation of Convection Mining Method: Phase 4: Replacement of Topsoil
Topsoil is replaced and revegetation is performed.

2.4 Alternative Open Cast Mining Methods:

Various alternative mining methods can be envisaged for open cast mining of the UG2 in the Bushveld, such as drag line excavation, or bucket excavation. These mining methods are known to be extremely cheap and effective in coal mining environments. Their application in the Bushveld environment has been examined and a number of problems with these methods have been identified for application in that environment. Firstly, there is the problem of scale. For example, a small drag line costs in the region of R30 million. To justify this type of expenditure, an extensive pit is required. Few of the available UG2 outcrop areas are sufficiently large to make the use of a drag line attractive. This also applies to bucket excavators. Secondly, from an operational point of view, a drag line requires a level working surface, which would be difficult to obtain on the UG2, which dips at between 8° and 20° in the western Bushveld. Thirdly, due to the geometry of UG2 pits, the drag line would have to progress along strike, rather than on dip. This would require a considerable boom-reach to place the spoil behind the drag line, so that sufficient space would be created to enable the extraction of the ore.

Another alternative being examined is the new highwall mining system, developed by Cutting Edge Technology Pty. Ltd., an Australian company, in collaboration with the Australian coal mining industry known as the Super Auger. In open cast coal mining it is often the stripping ratio which decides the economic cut-off depth of mining. Cutting Edge Technology have developed an augering system which removes coal from beneath the highwall, without the necessity of removing the overlying spoil. The percentage extraction is a function of the rock strength, with pillar size being determined by this.

The Super Auger system can penetrate up to 200m down dip, to a maximum of 15°, from the highwall position in seams between 1.5 and 4.5m in thickness. The system also has active directional and steering capabilities in-seam. At present Cutting Edge Technology is attempting to adapt the technology to hard rock environments. The potential of a system such as this is obvious. A pit need only be mined to say 10m depth, to establish the dip and strike, reef thickness and consistency of the reef horizon, and then, by using the Super Auger technology, the reef horizon for say a further 100-150m down dip could be extracted. In Australia the costing structure for this is also quite attractive, as the work is subcontracted, and a similar arrangement would make this system very attractive in the South African mining industry. This technology is finding some application in the Witbank coal field at present.

Obviously there are still some problems to be resolved, if the Super Auger technology is to be successfully adapted to hard rock mining. The breakout force required for the UG2 will be significantly higher than for coal. The specific gravity of chromitites will require more energy to be expended in moving the broken ore from the bit-face/cutting-tool up-dip into the pit. The abrasiveness of the chromitites will require careful consideration, both in terms of cost of manufacture of this machinery and wear of cutting bits.

It is envisaged that the Super Auger would be set up in the box-cut and extract the ore down to a pre-determined depth, and then moved, following the mining direction, to the next position, where the process would be repeated. Obviously this would become an integral part of the mining process. It could be used to exploit reserves which are too deep to mine by conventional open cast methods, yet too shallow to be exploited by underground methods, due to ground conditions.

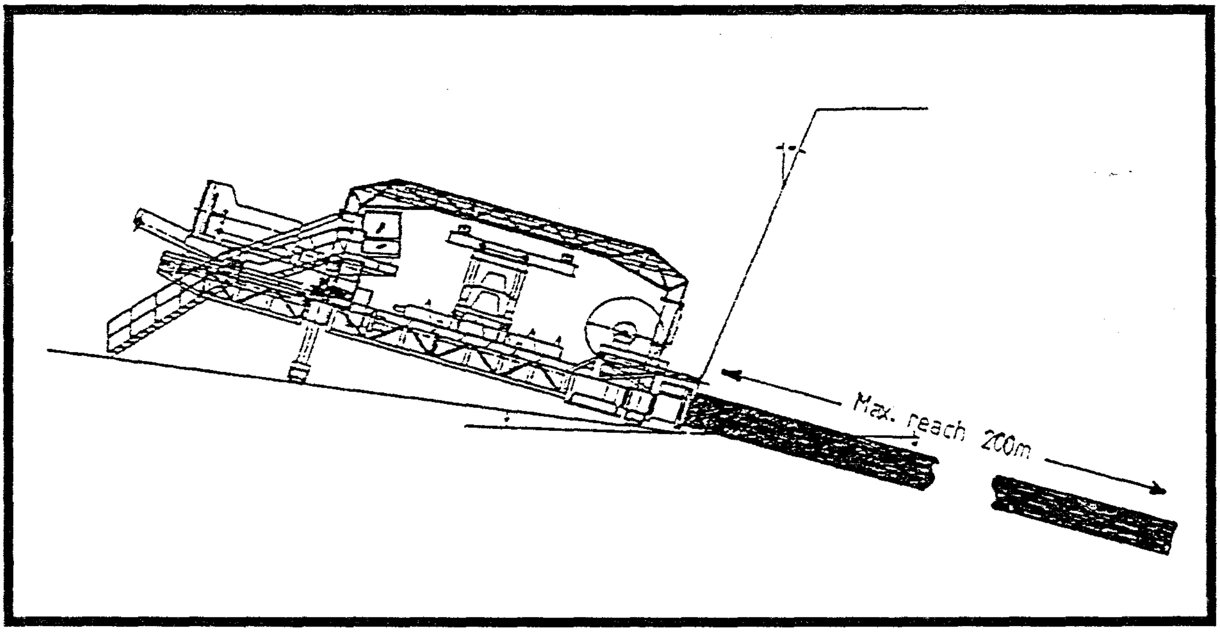
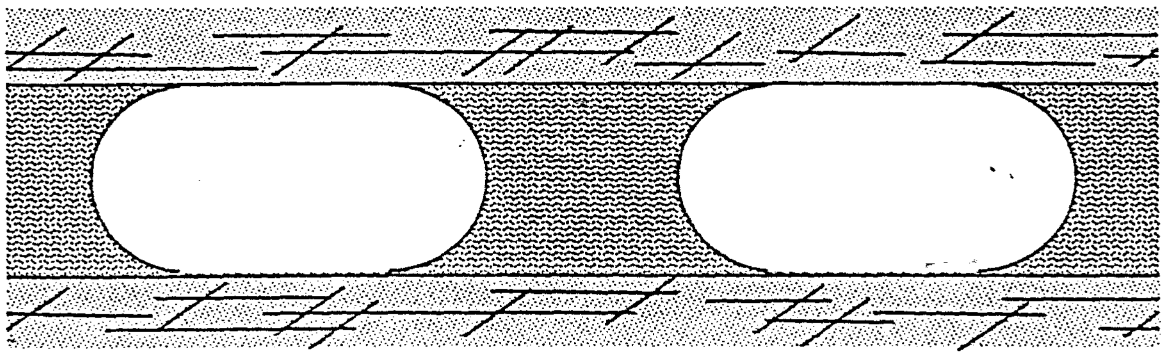


FIGURE 2.6

Super Auger at maximum working inclination (15°).
Machine operating on a 5° pit floor, tilted to mine at 15°.



Entry Geometry for Super Auger

FIGURE 2.7

The Super Auger cuts an excavation profile which maximizes the self supporting capacity of the ground, whilst achieving an acceptable level of recovery

2.5 In-House Mining v's Contract Mining:

The problems of deciding whether or not to mine a deposit in-house or to contract the work out is a complex one, and may depend on such diverse factors as Company philosophy, available capital, size of deposit and therefore the projected life of the mine, and even geographical location.

Most of the open cast operations in the western Bushveld Complex are being mined as stopgap measures, to overcome short term production problems, or to fill concentrator plants until underground tonnage is sufficient to do so. In this type of environment it makes little economic sense to expend large amounts of capital to enable in-house mining. The cost of plant and machinery for a 40 000t/month pit will be between R15 and R20 million and few of these pits could bear this kind of capital expenditure and still make a profit.

From the point of view of the mining house or mineral rights holder, it is probably not a viable proposition to even consider mining a deposit itself unless the deposit is large enough to support mining for 5-10 years. The rate of mining is also a consideration.

Using contractors also facilitates some flexibility. It allows production rates to be varied through time, at no extra expense to the client, if the contract is properly formulated. Variable production rates would be very difficult to justify if the pit was being mined in house and substantial capital had been spent on plant and machinery. Once purchased, this equipment must be kept fully occupied to justify their purchase. Variable production would mean that the plant would at times be fully utilized and underutilized at others.

Another issue is that of staffing: both the overhead that it would create, and the need for the pit to bear this cost, and the problem of obtaining the skilled personnel required. Once the pit is worked out, there would exist the problem of what to do with these personnel.

However, for a large pit, these issues would have to be examined carefully. Amplats has several areas where pits of ±20km of both UG2 and Merensky could be mined by open cast methods. In the case of such large pit it might be more cost effective to mine these in-house.

Chapter 3

Chapter 3

3.0 Metallurgical Considerations

The information contained in this chapter is a précis of information from various Mintek reports, Lonrho Platinum Division internal research and development reports, Mineral and Metal Extraction: An Overview, which are cited at the back of this work in the Bibliography and Reference section, as well as information obtained from various metallurgists, acknowledged in the text.

3.1 Background:

The concentration/recovery process employed by South African PGM producers for both the Merensky and the UG2 is essentially a sulfide flotation process. Froth flotation exploits the surface properties of minerals to achieve mineral separation. Surface properties are determined by the type of chemical bonding and chemical composition of a mineral, and are specific to that mineral species. Because of these specific properties, flotation offers the capability of selective separation.

Flotation processes achieve mineral separation in an agitated slurry, through which air bubbles are allowed to stream. The process aims to achieve the attachment of a specific class of material to the air bubbles. Once attached to the air bubble, the material is raised through the slurry and forms a froth (due to the action of a frother added to the slurry) on the surface. The froth is removed at the top of the flotation cell and the material that it has selectively removed from the slurry forms the concentrate.

There is a very definite limit on the size of the particles that this method can successfully separate. Particles need to be sufficiently small so that they can be effected by the surface tension of the bubbles and "attached" to the bubble, and so that it can overcome the effects of gravity. Flotation can be used to remove the target material or gangue, dependant upon the surface characteristics.

Flotation depends on whether a mineral is hydrophilic or hydrophobic as only hydrophobic minerals will selectively attach themselves to the bubbles. However, where a mineral is not sufficiently hydrophobic, the material (in the form of the slurry) may be pre-treated or conditioned, to improve its hydrophobic characteristics and ensure that it will float.

Some minerals are highly hydrophobic and will bond spontaneously to the air bubbles. If such minerals are plentiful in the ore, they may entirely coat the air bubbles. Where these minerals are not the target mineral and are gangue minerals, suppressants may have to be added to the slurry to prevent this and to facilitate the flotation of only the target mineral. Fairly common examples of highly hydrophobic minerals that bond spontaneously to bubbles are graphite and talc.

As with any other low-grade deposit type, recovery of metal from the ore is a crucial consideration in both the Merensky and the UG2. The processes for both ore types are now very well established, but these processes are not optimal for oxidized ores, and indeed until recently, this was not even a consideration. Yet the scale of open cast operations does not (in most cases) justify either the commissioning of new concentrators, nor of extensive modification to existing concentrators, to optimize recoveries. It falls to the metallurgists, therefore, to optimize recoveries by making adjustments as best they can, within existing processes, to achieve this when processing open cast ore, especially when this ore is being batch processed.

If, on the other hand, open cast ore is simply blended with other, underground, ROM ore, the overall process cannot really be adjusted in any meaningful way to optimize recoveries from the open cast ore. In fact, if the open cast ore is oxidized and weathered and if it is blended with underground ore, it may have a detrimental affect on the overall recovery. Once two ore types are blended, it is impossible to determine the specific effects one may have on recovery, and to what degree.

There are two specific problems which need to be addressed in the western Bushveld when processing open cast ore. Firstly, when dealing with the Merensky, the tenor of sulfides in the ore is critical. Secondly, and a problem specific to the UG2 ore, is the problem of the chromite content of the concentrate.

In both cases, and as a result of treating shallow oxidized ore, the problem of introducing clay minerals and talc into the concentrator needs to be addressed, and managed carefully.

In the case of the Merensky, the sulfides are intimately associated with the PGM mineralization. If the sulfides are absent (through oxidation) recoveries will be low. Bulk metallurgical test-work on highly oxidized Merensky ore at WPL, initially reported recoveries of only 15%-20% using the standard Merensky flotation process. However, it has subsequently been established that recoveries could be expected to rise to between 55%-60% if the concentrator process were to be modified. These modifications would cost several million Rand and would involve, amongst other modifications, the sulfidization of the milled ore at the conditioning stage. However, even at 55%-60% recovery, this ore would be sub-economic, due to its low PGM tenor (see Chapter 5).

Amplats, on the other hand, report recovery in the region of 64% from their Bleskop Merensky Pit (Harrison, *pers. comm.*, 1996). The Merensky ore from this pit is less weathered and oxidized than that from the WPL area, and contains some fresh sulfides. This recovery figure of 64% is based on bench tests, and was probably achieved using some of the technology and processes developed for the Platreef at PPRust. The ore from the Bleskop pit is railed to the main RPM processing plant and blended with ROM underground ore. It is therefore highly unlikely that Amplats know what the actual achieved recovery from this ore is. As it is processed with ROM ore in a "standard" concentrator process, and as it is weathered and oxidized to some extent, it is unlikely that a recovery of 64% is achieved, and indeed, it probably affects the recovery of the ore with which it is blended, unless it is substantially diluted.

In the case of the UG2, which has a sulfide tenor an order of magnitude lower than the Merensky Reef, the lack of sulfide is not such a large problem as it is in the Merensky. However, the oxidized UG2 ore from open cast operations tends to be much more friable than underground ore and hence, during the milling process it tends to be over milled, giving a high percentage of particles <10 μm in size. Unfortunately, this finer material is mainly made up of chromite, and it tends to float and reports to the concentrate, due to hydraulic entrainment. Chromite concentration in concentrate is one of the major problems that must be overcome in the processing of UG2 ore as it creates major difficulties in the smelting process.

The chromite content of the concentrate from open cast ore would be expected to be higher than the limit set by the smelter. This limit is 3%, and based on experience, could be expected to be in the region of 5% for open cast concentrate. The chromite content is affected by the "mass pull" at the plant. The mass pull is the ratio between the mass of material milled to the mass of the concentrate. For example, if 100 000t of ore is milled and the mass of the concentrate is 3 000t, the mass pull is 3%. There is also a direct relationship between the mass pull and the percentage chromite content of the concentrate. The higher the mass pull, the higher the chromite content. The chromite content in turn affects the smelter. This is due to the refractory nature of chromite (Assad, *pers. comm.*, 1996)

As the smelting temperature of the smelters is not sufficiently high to smelt chromite if it is in excess of ±3.5% due to its refractory nature, the chromite builds up in the smelter (in the form of a spinel-matte) and becomes very corrosive. The chromite content in the smelter has two main effects: firstly, it tends to settle out on the furnace hearth, and can, with time, considerably reduce the effective volume of the furnace and secondly, it may give rise to tapping problems. To overcome these problems, a smelting process was designed jointly by Mintek and WPL which would overcome these problems. This process involves the use of higher smelting temperatures and power densities (watts per square metre of furnace area), causing the chromium introduced as chromite, to dissolve in the matte and slag. This is then removed when the furnace is tapped, and negates the build up problem.

However, this process will only work effectively when the chromite content of the charge is less than ±3.5%. Once the chromite content of the charge is higher, it builds up in the furnace and causes the problems cited above. High chromite concentrations in the furnace also cause a change in the basicity within the furnace, and the contents become more corrosive than it is designed to cope with. This requires that the smelter be "taken down" and re-bricked more frequently than the planned interval of once every six months, at a cost of ± R200 000.

3.2 Other Considerations:

One of the most serious effects of milling and processing oxidized and weathered ore is that of sliming. This is a particular problem where process water is recycled. Very fine material becomes suspended in the process water and will not settle out easily. As the water is recycled, the amount of suspended fine material increases. This then begins to affect the recovery in the flotation process, due to the fact that this fine material coats the bubbles in the froth. The flotation process then becomes very efficient in floating very fine particle of gangue and not much else. Sliming is a function of particle size, and not so much the surface characteristics or chemistry. Concentrators usually cope with fine material in settling dams (concentrate or tailings settlers) where the residency duration and lack of agitation are normally sufficient to allow the fine material to settle out of the process water. Once sliming has occurred it is very difficult to resolve. Flocculents and coagulants on their own have proved ineffective. However, recent work at EPL has shown a combination of the two to be effective. At present the problem is being managed by the addition of large quantities of depressants to the slurry (Streczynski, *pers. comm.*, 1996).

Due to the chemical and physical nature of oxidized ore, especially in the UG2, this material needs to be treated and floated quite aggressively and this has two important effects: firstly, it means that the mass-pull is greater (and hence the concentrate grade lower) than for underground ore, and secondly, it increases the chromite content of the concentrate.

At present at the EPL concentrator, the mass-pull for underground ore ranges between 0.9% and 1.1%, whereas the mass-pull for open cast ore varies between 1.1% and 1.8%. Obviously this has a marked affect on the concentrate grade: underground ore produces a concentrate grading ±330 g/t whereas that produced from open cast ore grades between ±160 g/t to ±245 g/t depending on whether shallow or deep open cast ore is being processed (Streczynski, *pers. comm.*, 1996). From a smelting point of view, the lower the volume (mass-pull) and the higher the grade of the concentrate, the better. This has to be balanced with consideration of the chromite content, especially as the open cast ore tends to produce more very fine chromite particle in the milling stage, due to its weathered nature.

The chromite content of the concentrate is intimately associated with the mass-pull and concentrate grade. There is a fine balance between the mass-pull, concentrate grade and chromite content. Herein lies the art of the UG2 concentrator metallurgist. To obtain an acceptable concentrate grade, the mass pull needs to be decreased. Decreasing the mass-pull will decrease the chromite content of the concentrate. Adjusting the kinematics of the froth to reduce chromite entrainment will also reduce the PGM recovery. Reducing the mass-pull, to increase the concentrate grade, will increase the chromite content of the concentrate. The metallurgist must balance these two evils to produce a concentrate of acceptable grade with an acceptable chromite content.

3.3 Flotation Bench-Testing Methodology:

One of the most difficult factors to quantify when examining a complex polymetallic ore-body such as the UG2 is that of metallurgical recoveries in the sulfide froth flotation process. Because of these difficulties a methodology has been developed jointly by LPD and Mintek, and that process is outlined here and some actual data from the EPL open cast ore, which will be discussed later in Chapter 5. The real problem lies in relating bench-test scale results to what will actually occur in the concentrator when the ore in processed on a production scale.

3.3.1 Method

Two 1 kilogram samples of the ore are weighed out, as well as a ±250g sample (for head grade assay). This assayed grade is later checked against the calculated figure. Reagent solutions are made up and added to one of the 1 kilogram samples which is put into a rougher cell and temperature, pH and mV readings are taken. The rougher cell is switched on and agitation is started. The rotor speed is set at 1000 rpm. The required amount of reagents and frother are added and conditioning is allowed to take place. The rotor speed is increased to 1200 rpm and air is introduced on the rotometre at a rate of 11l/minute.

The concentrate is drawn off at 15 second intervals to provide five concentrate samples:

- First Concentrate: 0 - 1 minute.
- Second Concentrate: 1 -3 minutes.
- Third Concentrate: 3 -8 minutes.
- Fourth Concentrate: 8 - 15 minutes.
- Fifth Concentrate: 15 - 20 minutes.

The whole process above is repeated for the second 1 kilogram sample. The concentrates can either be combined with the (time equivalents) of the first test or kept separate. The concentrates are then weighed, pressure filtered and dried, and then the dry sample is weighed, and the concentrate samples are submitted for assay. Nickle-sulfide assays are performed, which provides individual PGM grades, as well as Ni, Cu and Cr values.

3.3.2 Interpretation of Flotation Bench-Test Results

The interpretation of the data is essentially a matter of fitting the data to a Kelsall model curve. In the case of the UG2, the fast floating fraction is taken to represent the expected recovery in the concentrator. The interpretation also involves the prediction of the chromite percentage concentration in the concentrate (Streczynski, *pers. comm.*, 1996).

Table 3.1

Metallurgical Test Results

Ore Description	Lab. Grind %-75 µm	Chromite %		PGM+Au Recovery	Assay Results PGM + Au		
		Feed	RC	%	Feed	RT	RC
Plant Feed	71.0%	27.1%	8.4%	90.0%	4.23	0.46	35.4
Shallow Open Cast	74.7%	27.4%	12.5%	62.4%	4.75	1.97	31.8
Medium Open Cast	74.7%	29.7%	12.5%	71.9%	3.96	1.26	23.6
Deep Open Cast	74.0%	28.0%	12.2%	70.3%	4.44	1.44	38.5

The plant feed was normal ROM underground ore

RT = Rougher Tailings **RC** = Rougher Concentrate

Table 3.1 is the raw data and a few of the data merit some discussion. Firstly, notice that in all cases for the open cast ore the grind shows a greater percentage of material finer than -75 µm than for the normal (underground ore) plant feed. This is a time dependant factor, but as EPL's mill has a constant, fixed feed rate, this means that inevitably open cast ore is over-milled and produces a larger proportion of very fine material than does underground ore. This is directly related to the issue of the degree of weathering that the ore has undergone and it is this very fine material that causes sliming in the concentrator. Secondly, the chromite percentage in the rougher concentrate is considerably higher for the open cast ore than for the plant feed. Thirdly, the rougher tailing grade for the open cast ore is considerably higher than for the normal plant feed, even allowing for the differences in the feed grade. Finally, this data is based on the overall recovery of the flow rate tests, after 20 minutes.

The underlying presumption is that the material being tested in the laboratory will behave in a similar fashion on a production scale. It is known that the froth generated on a bench scale is "thinner" and has slightly different kinematics than that in the plant. These data are therefore taken to be the lower limit (minimum percentage recovery) of what can be expected in production in the plant.

Table 3.2

Expected Minimum Recoveries: PGM + Au

(Fast Floating Fraction)

Ore Description	PGM + Au Fast Floating Fraction (%)	Expected Minimum Plant PGM + Au Recovery (%)
Plant Feed	79.0%	80.0%
Shallow Open Cast	-	-
Medium Open Cast	53.8%	**53.8%**
Deep Open Cast	57.0%	**57.0%**

The percentage recoveries are fitted to a Kelsall curve and from that a prediction is made as to the recoveries that can be expected from the ore in question, once processed through the plant. This data is presented in Table 3.2, above. In fitting the data to the Kelsall curves, only the data relating to the fast floating fraction is used. Unfortunately, there was insufficient sample concentrate for the fast floating fraction of the shallow open cast ore on which to perform a nickel-sulfide assay test, and there was therefore no data available for this ore.

The accuracy of the predicted recoveries, based on these bench tests, will be discussed further in chapter five. Chapter four will examine some of the environmental considerations that need to be addresses before mining can proceed.

Chapter 4

Chapter 4

4.0 Environmental Considerations of Open cast Mining

4.1 Background:

The Minerals Act, No 50, 1991 was enacted:

"to regulate the prospecting for and the optimal exploitation, processing and utilization of minerals; to provide for the safety and health of persons concerned in mines and works; to regulate the orderly utilization and the *rehabilitation of the surface of land during and after prospecting and mining operations; and to provide for matters connected therewith.*"
(The italics are the writer's.)

In terms of this act, an Environmental Management Programme Report (EMPR) is required for any proposed prospecting and mining. The main aim of an EMPR is to determine the environmental management criteria to minimize negative impacts and maximize positive impacts in the exploration and mining milieu. The important philosophy behind the Act and the requirements of the EMPR is that a mine site should affect as little land as possible, and those effects should be managed to limit the long term damage. This means that the land usage after mining must be planned and managed throughout the life of the mine. One of the objectives of the Act is to provide a closure certificate for the mine, once all the requirements of the EMPR have been adequately addressed. Even after the issuance of a closure certificate, the mining company or the holder of the mineral rights remains responsible in **perpetuity** for, *inter alia,* the management and treatment of mine drainage water and tailings dumps.

4.1.1 Specific Requirements for Open Cast Operations

Neither the Act nor the EMPR Aide Mémoir relates any specific requirements for open cast operations. This is due to the fact that the Minerals Act redefined the term mine to include:

(i) "any excavation in the earth, including the portion of the sea or under the water or in any tailings, as well as any borehole, whether being worked or not, made for the purpose of searching for or winning a mineral"

(ii) "any other place where a mineral deposit is being exploited, including the mining area and all buildings, structures, machinery, mine dumps, access roads or objects situated on such area and which are used or intended to be used in connection with such searching, winning or exploration or for the processing of such mineral:..."

Under the previous Act, quarries and sand and gravel pits were not included in the definition. The EMPR requirements for all "mines" are now the same, subject to certain exemptions.

4.2 Environmental Management Programme Report:

The Minerals Act, 1991, requires every mine owner to submit, and obtain approval of, an EMPR before mining or exploration operations begin. The purpose of the EMPR is to compile an environmental management programme that is acceptable to all concerned, "with a view to leaving a useful heritage to future generations after the mineral resources have been extracted" (Aide Mémoir). The EMPR is a wide ranging document and on submittal, is scruitinized be several Government Departments, including:

- The Department of Mineral and Energy Affairs
- The Department of Water Affairs and Forestry
- The Department of Environmental Affairs
- The Department of Agriculture
- The Department of National Health and Population Development
- The Department of Finance

The first three listed here take an active role in assisting to draft the EMPR, and ensure that various criteria are met. The latter three are only interested in the information contained in the approved EMPR document and do not involve themselves in the drafting process.

4.2.1 Purpose of the EMPR

The purpose of the EMPR is to provide a single document that satisfies the authorities concerned with regulation of the environmental impacts of mining by describing the pre-mining environment at the proposed mine site and assessing the significant impacts that the mine is likely to have on the environment both during and after mining.

The EMPR must also describe how the negative impacts of mining will be managed, and the positive impacts will be maximized. The positive impacts are mainly seen to be socio-economic, and therefore the project must be motivated and the benefits described, either on a local or national scale. The most important purpose of the EMPR is that of setting out the actual management programme, which will be applied throughout the life of the project so that stated and agreed land capability and closure objectives can be achieved, and a closure certificate issued, and of course, to ensure that sufficient funds are set aside to implement the environmental management programme.

4.2.2 Requirements of the EMPR

The EMPR takes the form of a report divided into 11 parts:

Part 1: Part 1 is a brief project description including data on the proponent and on the proposed project:

- Administrative detail.

Part 2: Part 2 is essentially a description of pre-mining environment, optimally, as it was before mining commenced. It requires rather comprehensive and detailed accounts of following topics:

- Geology - representative sections, borehole logs, geological map, and an assessment of how the material to be mined will affect the water quality.
- Climate - a description of the climate in the area.
- Topography - Topographical plans.
- Soils - Description and plan of soil types.

- Pre-Mining Land capability - Land-use classification maps and existing structures.
- Natural Vegetation - Vegetation maps.
- Surface Water - Complete hydrological assessment of the area to be mined, including the catchment and water quality.
- Ground Water - Complete hydrological assessment and effects of mining thereto.
- Air Quality - Complete survey of existing air quality and an assessment of the impact mining will have on it.
- Noise - Identification of existing and an estimate of impact of mining on noise levels.
- Archaeological & Cultural Sites - Such sites need be noted.
- Sensitive Landscapes - Description of areas of statutory protection is required.
- Visual Aspects - Assessment of visual impact of mine site.
- Regional Socio-Economic Structure - Assessment of the impact of the mine on socio-economic structure, on a regional or national basis.
- Interested and Affected Parties - Parties with whom consultation may be required.

Part 3: Part 3 requires a fairly detailed motivation of the proposed project requiring financial data as well as an assessment of the socio-economic impact of it:

- Benefits of the Project - Estimates of expenditure, labour force, multiplier effect and market.
- Alternative Projects - A record of all alternative options considered for the mine site.

Part 4: Part 4 requires a detailed description of the proposed project. This will obviously be used to assess what the impact will be on the pre-mining environment, described in part 2:

- Surface Infrastructure - All proposed infrastructure, waste management facilities, water pollution and treatment management facilities, mineral processing plant, housing, transport, water balance diagram, disturbances of water courses and storm water management.
- Construction Phase - Brief description.
- Operational phase - Detailed description of impact on the environment of infrastructure, mineral processing, plant residue disposal, transport, river diversions.

Part 5: Part 5 is the environmental impact assessment. In this section the proponent is required to demonstrate an in-depth understanding of the impact that the proposed project will have, and how those impacts will be managed:

- All the foregoing factors must be covered in such a way that the mining proponent can demonstrate that he has considered and understood the potential or expected impacts of the project on the environment, for all stages of the project: construction, operational, decommissioning and post-closure in terms of all the topics described in part 2.

Part 6: Part 6 details the actual Environmental Management Programme:
Note: This is the only part of the EMPR which becomes legally binding, once the EMPR is approved.

- Management programme for all the environmental impacts described in Part 5 of this document for all stages of the proposed project, including a proposed timetable, duration and sequence of events, for both mining and exploration and financial provisions that will be made.

Part 7: Part 7 forms the conclusions:

- Detail of the overall net impact of the proposed project on the environment.

The last four parts (Parts 8-11) deal with documentation, permits for the project, statutory requirements, references and supporting documentation. Part 11 provides for financial and propriety material which is included in the EMPR but which the proponent may wish to remain confidential.

Part 8: Statutory Requirements:

- A list of all permission granted, pertaining to the proposed project.

Part 9: Amendments to EMPR:

Part 10: References and Supporting Documentation:

Part 11: Confidential Material:

4.3 Environmental Implication of Open Cast Mining at EPL:

Lonrho was the first platinum mining house to have obtained approval of its EMPR in South Africa, for Western Platinum Mine. Eastern Platinum Ltd. was in the process of finalizing their EMPR when the open cast project began, and at that stage it was decided to incorporate all possible open cast areas into it. However, as the EMPR is designed to be a dynamic document, subject to change in some respects, according to changing circumstances, had the EMPR already been approved revisions to it could have been submitted to the Regional Director of the Department of Mineral and Energy Affairs for approval. This was the case on WPL when they decided to proceed with open cast operations there.

The main considerations, as far as the EMPR is concerned, are those of ground water, rivers and land rehabilitation and use after mining. The issue of land use after mining can, to some extent, be determined by the land owner. If the land was veld before mining, but is suitable for agricultural purposes, the land owner can opt to have the rehabilitated mine area converted to agricultural use after mining. Obviously this will depend to a large extent on land use in the surrounding area, and is subject to approval by the Departments of Environment Affairs, Agriculture and Water Affairs.

An environmental risk analysis of the proposed open cast areas was performed and a number of problematic factors were identified. The 2# East Pit area is traversed by a non-perennial stream, and dwellings lay within a 500m radius of the pit. The 3# West Pit also has dwellings within 500m radius of it and lies adjacent to a provincial road. The 3# East Pit is also traversed by a non-perennial stream.

Of these problems the most difficult to overcome is the existence of streams traversing the open cast areas. Blasting can be tailored to minimize the effect on nearby dwellings and roads. All the nearby dwellings lie to the south of the proposed pits and this could be used to the mine's advantage. As the dip-slopes of the pits dip towards the north, when blasting occurred the shockwave would be directed upwards (into the air) in a southerly direction. The presence of a thick cover of Black Turf Cotton Soil would also cushion the effects of blasting as it is not conducive to propagating a surface shock wave.

Various options were examined to deal with the problems of the existence of streams traversing the pits, under the guidance of a consulting hydrogeologist. The Department of Water Affairs and Forestry (DWAF) were consulted and, initially at least, they required a fully fledged river diversion **around** the pit. An estimate of costs for the scope of work required ran to several million Rand. This would have severely affected the profitability of the pits concerned. Negotiations were held with the DWAF, wherein hydrological data (flow histories, volumes, total dissolved solids (TDS) data, etc.) was presented to them, as well as alternative solutions to deal with the streams. Eventually permission was granted to mine without a stream diversion, subject to certain conditions being met, to wit: the stream area must be mined and re-instated **within** one dry season; the landfill below the re-instated stream must be compacted to specific specifications; the re-instated stream bed must be clay lined and rock-clad; and finally the re-instated stream bed must be grassed to reduce erosion and TDS.

The DWAF also imposed several other requirements. No spoil or topsoil could be dumped or stockpiled within a fixed distance of the stream bed and berm walls had to be constructed (to certain specifications) on either side of the stream once it was reinstated. The stream flow and water quality must be measured and monitored whenever it flows. The main consideration here is that silt must be prevented from entering the catchment so that down-stream users should not be adversely affected by the mining of the pit.

The DWAF have reserved the right to immediately halt all operations in the pit if any of their stipulations are not being adhered to, and to impose severe fines on EPL in such an event.

Any water made in the pit during the operational phase must be pumped into settling dams and allowed to settle before it is released into the river catchment system, also a measure to keep silt out of the river system.

Water monitoring boreholes were required to be in place before mining commenced, both up-stream and down-stream of the pit limits. Once mining is complete, further water monitoring borehole are required on, or next to, any geological feature (fault, dyke, etc.) that extends beyond the limits of the pit.

The blasting license was granted without any problem, or onerous requirements, based on the blast design submitted to the Department of Mineral and Energy Affairs. However, there is provision made from the profits of the open cast operation, to cover any potential claims against the company for damage to dwellings and other structures.

Chapter 5

Chapter 5

5.0 Case Study: UG2 Open CastProject at Eastern Platinum Ltd.

5.1 Background:

Eastern Platinum Ltd. (Figure 5.1), the subject of this case study, began shaft sinking in 1987 following several years of exploration. It was originally planned to mine 160 000t per month, but due to improvements in the plant and revision of mine planning, now mines ±200 000t per month. In February 1990 the first UG2 open cast operation (A-West Pit, Figure 5.1) was started and in September 1990 a second pit (C-West Pit, Figure 5.1) was begun, in an attempt to fill the plant, which was running under capacity due to a slower build up of tonnage from underground than planned. At that time, Eastern Platinum Ltd. fell within the former Bophuthatswana Homeland and obtaining mining permission was relatively easy. These pits were planned to mine to 40m depth at a rate of 35 000t and 57 000t per month. At the time these pits were started, and for much of their life, PGM prices were high. It was during this period that rhodium peaked at just over $7 000/oz, and platinum was above $500/oz (Chapter 1). The shallow ore (0-8m) was sold to Crocodile River Mine whose plant reported good recoveries from the ore; better than EPL believed could be obtained from its own plant at the time.

However, due to contractual problems, little profit was made from these two pits, despite the high PGM prices prevailing at the time. In fact, an analysis of the data from those pits shows that the mining costs were R112/t of ore mined for a total of 1.658 Mt extracted from the two pits. Total profit, for the two pits was R5.2 million, giving a return based on cash flows in real terms, of 3.9%. Various factors were identified which contributed to the poor return, but essentially it is felt that too little work, if any, was put into the feasibility study. For example the first pit was not drilled at all, and the tonnage and grades were based on "guesstimates" derived from the mineral resource documents. This document itself was unreliable due to the newness of the mine and the paucity of data available at that point in time. This translated into an overestimate of the Reef thickness as well as an underestimate of the dip, creating a discrepancy between planned and actual tons of 100 000 t.

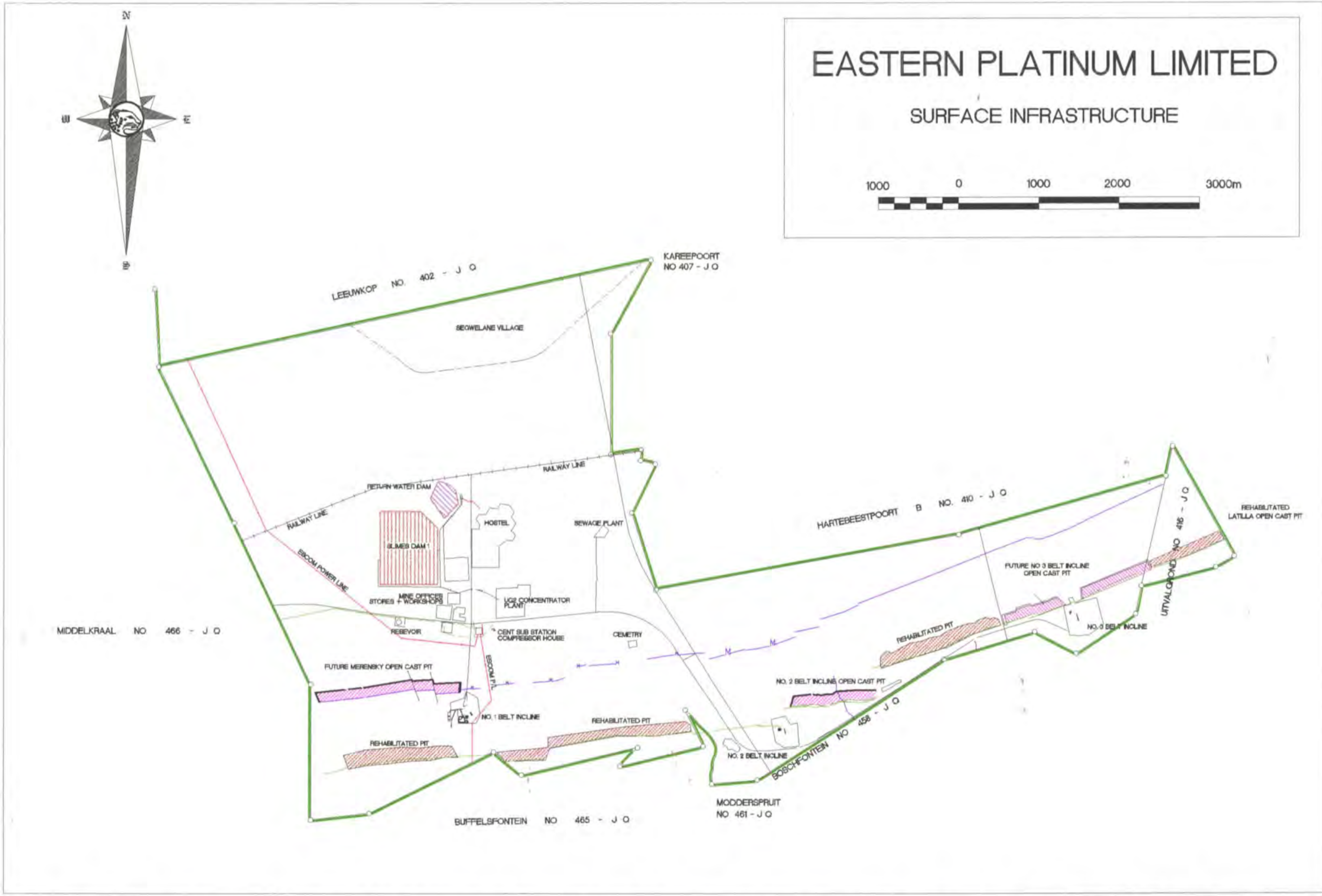


FIGURE 5 . 1 EASTERN PLATINUM LIMITED - DETAILED SURFACE INFRUSTRUCTURE PLAN WITH OPEN CAST PITS

During the latter part of 1995 underground production began to decline quite dramatically, due to massive ground losses, which affected all three declines simultaneously. These losses were due to a series of, for EPL, large potholes as well as several levels encountering large faults (30m). Although many of these potholes had been identified beforehand using ground magnetics, the planning had not taken account of the effect that they would have on production. These potholes were subsequently found to be large (>200m in diameter) and to overlap each other. The time needed to re-establish the levels, which had encountered those faults, on the other side of the faults is, on average, 14 months. At the same time there was a major drive on within the Lonrho group to increase the underground proved ore reserves (proved through development) from 12 to 18 months. It became obvious that unless something drastic was done the company would not only not be able to meet its production quotas, and thereby fulfil metals contracts, but would also not be able to increase its proved ore reserves. Compounding these problems was that of inexplicably low productivity in the underground operations (see Chapter 1).

Several areas of shallow UG2 are present on the property which had not previously been exploited (Figure 5.3), either by open cast mining or from underground and a project was initiated by the Chief Geologist to examine the possibility of mining them profitably. This would provide tonnage to overcome the problems encountered underground, and increase the proved ore reserve while maintaining production levels.

Due to known resistance at senior management levels, because of their previous experiences with open cast mining on the properties and its perceived lack of profitability, a very cautious approach was adopted. It was clear that a simple approach, quoting grades and tonnages would not persuade management to proceed with the project. Whatever approach used would have to show a worthwhile bottom-line profit. It was known that a return of about 30% would be acceptable. Anything substantially less would not be approved.

The above factors required that a fairly sophisticated model would have to be developed, incorporating procedures used for assessing underground potential and mirroring them as closely as possible. The data and factors used must be as accurate as possible and the costs and profit must be well defined.

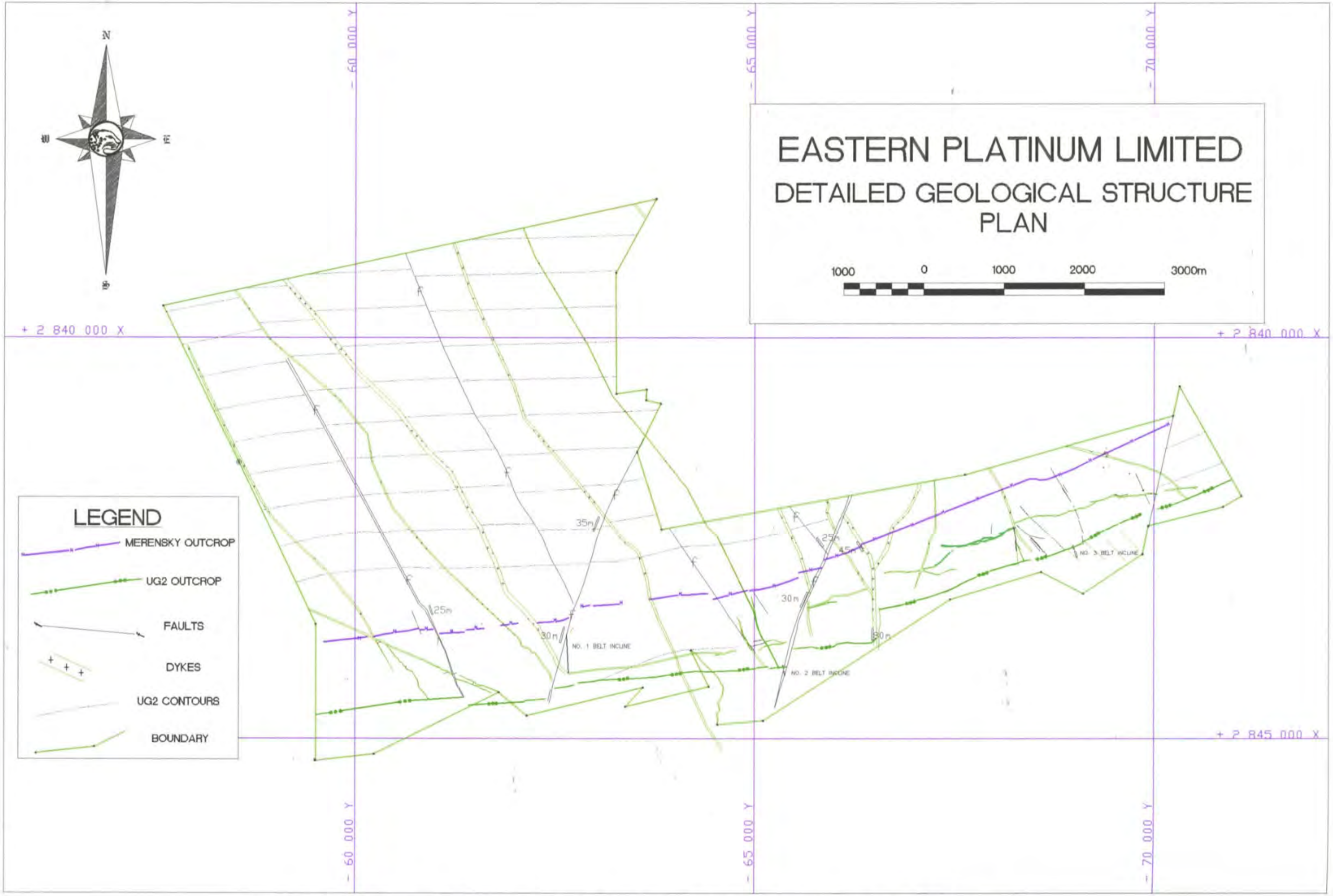


FIGURE 5 . 2 DETAILED GEOLOGICAL PLAN - EASTERN PLATINUM LIMITED

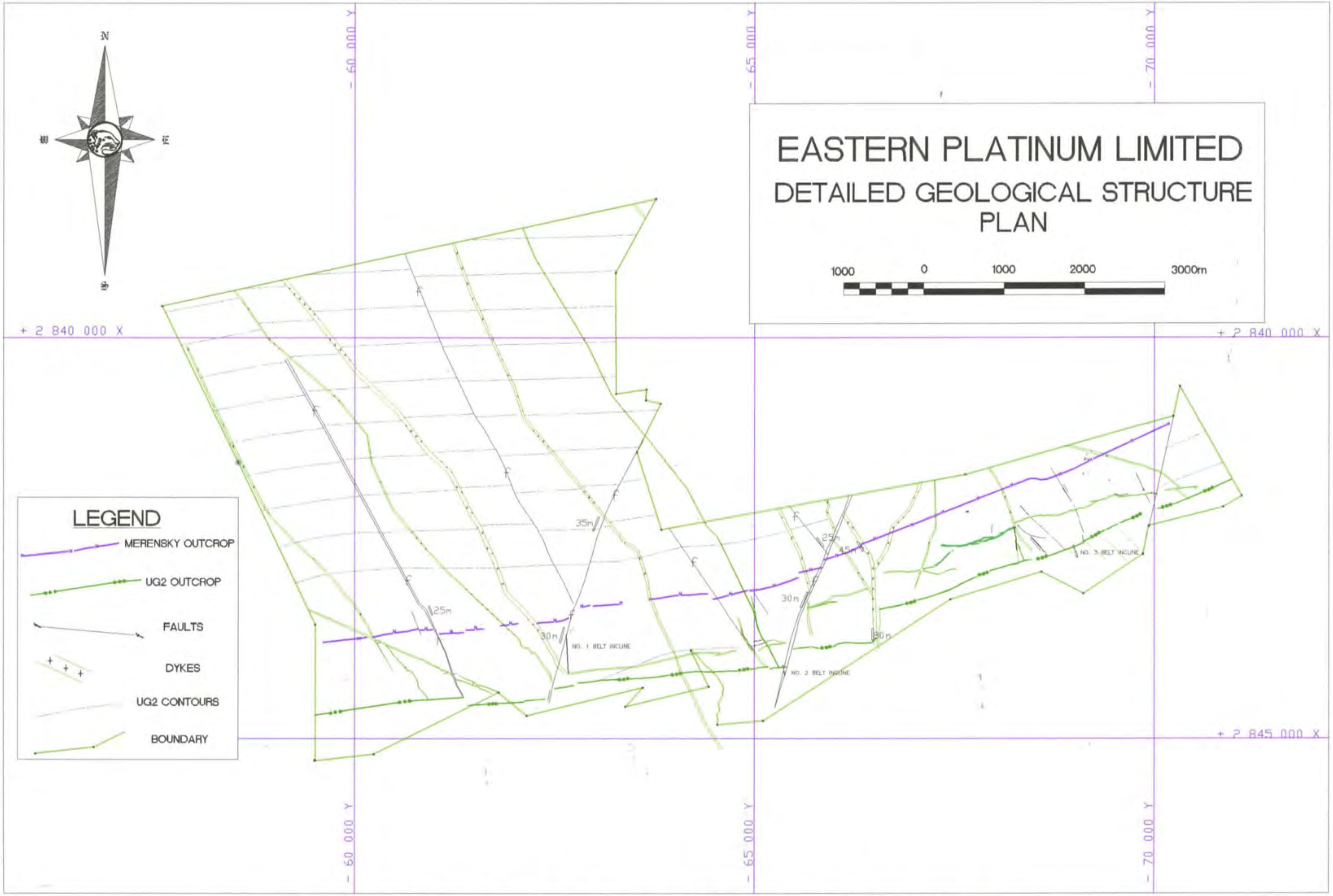


FIGURE 5 . 2 DETAILED GEOLOGICAL PLAN - EASTERN PLATINUM LIMITED

5.2 Pre-Feasibility Study:

Initially, three areas with potential for open cast mining the UG2 were identified: an area immediately to the east of No. 2 Incline (called the 2# East Pit); and two areas immediately to the east (3# East Pit) and west (3# West Pit) of No. 3 Incline (Figure 5.3). The area to the west of No. 2 Incline was deemed to be too small, being bordered to the west by the Sun City Road. The depths to which these pits could be mined would be determined by the depth of underground workings (Figure 5.3). The pit boundaries were essentially pre-determined by faults or dykes (east and west of the 2# East Pit as well as to the east of the 3# West Pit), underground workings on dip in the case of all three pits, or previously open cast areas (as is the case at the 3# East Pit) (Figure 5.3). In all three cases the outcrop had been trenched and sampled during 1990 and that sampling data was available (Table 5.1).

Table 5.1

Trench sampling results from the 2# East Pit

	UG2 thickness	Grade g/t	Comment
Trench 01	60 cm	Not sampled	Potholed
Trench 02	175 cm	6.1 g/t	Split Reef
Trench 03	130 cm	10.2 g/t	Normal
Trench 04	150 cm	9.2 g/t	Normal
Trench 05	50 cm	Not sampled	Potholed
Trench 06	150 cm	9.4 g/t	Normal
Trench 07	120 cm	4.23 g/t	Normal
Trench 08	45 cm	Not sampled	Potholed
Trench 09	150 cm	5.43 g/t	Normal
Trench 10	130 cm	4.65 g/t	Faulted

Several other parameters needed to be estimated for each of the pits, such as grade, tonnage, geological losses, etc., and these are discussed below:

5.2.1 Borehole Data:

None of the prospective areas had any boreholes, and at that stage no budget was available to drill any boreholes. The ore resources on the Lonrho properties are estimated using an inverse distance method, which is blocked out in 500m x 500m blocks. It was deemed that this would be sufficient for an initial "first pass" financial model. The grade for each pit was estimated from the annual ore resource document and the tonnage, of both ore and waste was estimated using a spreadsheet template (Mallinson, 1995).

5.2.2 Geological Losses:

Geological losses underground, due to potholes, faults and dykes, are excluded from the ore resources. For proved mineral resources, the amount deducted is based on empirical data and it is applied to a specific level, based on current logging and that level's history. For probable mineral resources, a factor of 15% is applied, based on some 25 years of mining history. As all three of the proposed pits had underground workings immediately down dip of them, empirical data for areas close at hand was available. The geological losses for these areas varied between 7% and 12%. However, despite this a conservative figure of 20% was used.

5.2.3 Metallurgy:

Discussions with metallurgists in the company indicated that "shallow" and "deep" ore would have to be treated separately, as the shallow ore would, based on experience, be more oxidized, and that different recoveries could be expected from each. In view of this, the financial model would have to reflect this. Fortunately, the Metallurgical Manager at EPL had worked at Crocodile River Mine and hence had experience of milling and flotation of UG2 open cast ore. He estimated that, due to the general metallurgical improvements in place at the EPL plant, a recovery of 68% could be expected from the shallow ore and a recovery of 74% from the deep ore, based on a cut-off of 15m vertical depth. In the past when EPL had milled open cast ore, a recovery of 63.8% was obtained from deep open-cast ore and 70.1% recovery from underground ore. However, recoveries of 68% for shallow ore and 74% for deep ore were felt to be too high. At the time recovery for underground ore was approximately 82%. It was therefore decided to err on the side of conservatism, and the recoveries used were reduced to 64% and 68 % respectively for shallow and deep ore.

As the concentration plant was running under capacity, the milling and concentrator costs were running higher than they would if the plant was full. This is due to a rather large fixed overhead. Despite the fact that the open cast ore would fill the mill it was decided to use the then current milling and concentrating cost/t, which amounted to R17.44/t. The smelting cost was fixed at R5.00/t of ore, the current smelting cost/ton, as these economies of scale would not be evident at the smelter, due to the small additional amount of concentrate that it would produce.

It was decided to apply the normal smelter recovery factor of 99%. It was felt that the amount of open cast concentrate would be such a small proportion of the total smelted, and sufficient dilution would occur, to make the additional chromite quite insignificant. Also, as the mass pull had recently been reduced (and hence the concentrate grade increased), and although this had increased the percentage of chromite in the concentrate, the absolute mass of chromite had been reduced.

5.2.4 Transport:

Ore from all three inclines on EPL is trucked from the shafts, over a weighbridge, to the plant. The distances vary from just over 2 km to 9.5 km. The transport is contracted out to a transport specialist and their cost/t/km was employed for the transport cost of ore from the open pits to the plant. That cost amounts to 42c/t/km. However, this cost operates on a fuel "rise and fall" basis and account would have to be taken of this.

5.2.5 Contract Mining:

Due to the limited tonnage available from prospective open pits, it was obvious right from the beginning that to mine them would be a short term solution to the problem of filling the plant. This is reinforced by the fact that due to lower recoveries it was expected to be less profitable than filling the plant with unoxidized, higher grade underground ore. Capital expenditure on mining plant and equipment to mine these resources was therefore out of the question. Furthermore, it was recognized that this form of mining is very different to EPL's normal core-business: underground mining. If it was a viable proposition, the mining of the open pits would be contracted out. This had the advantage of requiring virtually no capital expenditure. For the purposes of modelling the proposed mining, mining costs of between R40 and R50/t of ore were used. These costs were based on the then current mining costs known to be incurred by the Amplats group at their Bleskop Merensky open cast operation (R44/t), near Kroondal in the North West Province.

5.2.6 Delineation Costs:

Some small amount of capital expenditure would, however, be required. The pit areas would have to be drilled to prove ore resources and resolve as much of the structure as possible before mining commenced. These costs were estimated and the data fed into the template.

5.3 Financial Template - A First Pass Analysis:

A financial template was developed, incorporating all of the above data and factors (Figure 5.4). In addition to this a number of other factors had to be incorporated, to make the model as robust as possible. To remove the effects of inflation and a volatile exchange rate, the prevailing metal prices and $:R exchange rates for February 1996 were used. This was necessary as the operating profit is calculated twice monthly, based on actual metal prices and exchange rates.

A further complication in this regard is the fact that future revenue predicted from mine planning and production is based on an 18 month forecast by market consultants. The February 1996 metal prices were not particularly good (Figure 1.3) and the $:R exchange rate had begun its long downhill slide. The metal prices at prevailing exchange rates are used to calculate a Matte-Valuation, which is the Net Realization per kg of metal in Matte (Converter Matte). It reflects the actual proportional recoveries of each of the metals for Pt, Pd, Rh, Ir, Os and Au as well as Cu and Ni for a specific ore. As Rh was in oversupply in the marketplace at the time (Chapter 1) and Lonrho was stockpiling it rather than selling it, it was also decided to set the Rh value at zero in the matte valuation metals' basket. This provided a very conservative matte valuation of R29 754/kg based on an exchange rate of $3.6525:R1.

5.3.1 Preliminary Results:

All this data was then fed into the financial template (Figure 5.4). Various scenarios were then examined, wherein the various factors and costs were varied within certain parameters. This involved examining the effect of changing the tonnages, (by adjusting the highwall position up to the maximum as dictated by the depth of underground workings), varying the Matte-Valuation (as well as exchange rates), examining the effects of different recoveries, geological losses, dilution factors and fuel prices.

In all, twenty scenarios were examined for each pit using an iterative approach. All of these scenarios were profitable, and the profit varied between R6.0 million and almost R20.0 million. Interestingly, in every one of the different scenarios, the 2# East pit was the most profitable. This was surprising because No.2 Incline has historically had the lowest grade of the three inclines. However, this proposed pit is closest to the plant, and hence attracts the lowest transport costs. Additionally, despite having the same strike length as the 3# East Pit, this pit holds the greatest tonnage because it could me mined to 30m depth along its entire strike length (the ground conditions were so poor in the underground workings that no mining could take place above 35m below surface) whereas the 3# East pit can only be mined to 25m below surface. Furthermore, the UG2 channel (reef) thickness, from the mineral resource document (130cm) was much thicker that the channel thickness at either of the other two pits.

A selection of the most reasonable scenarios (their "reasonableness" decided by the General Manager and the Chief Geologist. Figure 5.4 was considered to be the most reasonable for the 2# East Pit.) as well as a proposed drilling programme were presented to senior management on the 4th of March 1996. Approval was sought and obtained to proceed with a drilling programme required to prove the mineral resources and obtain sufficient sample to perform metallurgical testing. In fact, senior management indicated that the proposed drilling programme be considerably expanded to avoid problems encountered in the previous pits due to a lack of drilling in them before they were begun. Permission was also obtained to approach mining contractors to obtain provisional quotations on mining costs. It was decided to concentrate on the 2# East Pit, in preference to the 3# East and West Pits. Nonetheless, these pits must too be further investigated.

FIGURE 5.4

PROPOSED OPEN-CAST AREA EAST OF 2 INCLINE: FIRST PASS ANALYSIS

PRESUMPTIONS:
Matte Valuation R29 754/kg
Dilution grade 0.3 g/t
Mining costs R40-R50/t of ore
Mining Depth: 30mbs
Rand/$: R3.6525:$1

		R40/t	R42.50/t	R45/t	R47.50/t	R50/t
RESERVES (TONS)	0-15 m	321,911	321,911	321,911	321,911	321,911
GEOLOGICAL LOSS	20%	64,382	64,382	64,382	64,382	64,382
MINEABLE TONS	(ORE)	257,529	257,529	257,529	257,529	257,529
MINING DILUTION	15%	38,629	38,629	38,629	38,629	38,629
% RECOVERY		64%	64%	64%	64%	64%
PGE RECOVERED	kg	897	897	897	897	897
RESERVES (TONS)	15-30 m	321,911	321,911	321,911	321,911	321,911
GEOLOGICAL LOSS	20%	64,382	64,382	64,382	64,382	64,382
MINEABLE TONS	(ORE)	257,529	257,529	257,529	257,529	257,529
MINING DILUTION	15%	38,629	38,629	38,629	38,629	38,629
% RECOVERY		68%	68%	68%	68%	68%
PGE RECOVERED	kg	954	954	954	954	954
REEF TONS	TOTAL	643,822	643,822	643,822	643,822	643,822
MILLED TONS		592,316	592,316	592,316	592,316	592,316
TONS MINED		643,822	643,822	643,822	643,822	643,822
PGE kg in conc.		1851	1851	1851	1851	1851
PGE GRADE	g/t	5.40	5.40	5.40	5.40	5.40
PGE kg in matte	kg	1795	1795	1795	1795	1795
MATTE VALUATION	R/kg	29,754	29,754	29,754	29,754	29,754
ON-MINE COSTS						
MINING/t	Rand	40.00	42.50	45.00	47.50	50.00
MILLING/t		17.44	17.44	17.44	17.44	17.44
SMELTING COST/t	Rand	5.00	5.00	5.00	5.00	5.00
TRANSPORT @42c/t/km	Rand	4.92	4.92	4.92	4.92	4.92
TOTAL COSTS/t OF ORE	Rand	67.43	69.93	72.43	74.93	77.43
REVENUE/t	Rand	90.19	90.19	90.19	90.19	90.19
DELINEATION COSTS	Rand	41,250	41,250	41,250	41,250	41,250
GROSS REVENUE	Rand	53,421,332	53,421,332	53,421,332	53,421,332	53,421,332
GROSS COSTS	Rand	39,939,672	41,420,463	42,901,253	44,382,044	45,862,834
GROSS PROFIT/t	Rand	22.76	20.26	17.76	15.26	12.76
GROSS PROFIT (TOTAL)	Rand	13,481,660	12,000,869	10,520,079	9,039,288	7,558,498
COST/kg	Rand	22,245	23,070	23,895	24,719	25,544
PROFIT/kg	Rand	7,509	6,684	5,859	5,035	4,210
MARGIN % (Gross Profit/ Gross Revenue)*100	Rand	25%	22%	20%	17%	14%

Tons & Grades from 1995 Mineral Resources Document

5.4 Feasibility Study:

5.4.1 Drilling Programme:

A drilling programme was designed for all three pits. As mentioned previously, trenching had been performed in 1990 on the entire outcrop strike length of all three pits. Three phases of drilling were planned (Figure 5.5). The first phase, with holes ±200m apart on strike along lines ±35 m apart on dip were to be diamond drill holes, obtaining UG2 reef intersections of NQ size (47.5mm diameter). The second phase would be in-fill drilling, with the holes placed midway between the phase 1 holes. The decision as to whether these holes would be diamond drill holes or reverse circulation percussion holes (102mm diameter) would be based on the core recovery and general condition of the core from the phase 1 drilling. The third phase would consist of further in-fill drilling, between the phase 2 holes. Once again the holes would be either diamond drill holes or reverse circulation percussion holes dependent upon the core recovery and condition of the core obtained from phases 1 and 2.

Phase 1 diamond drilling began on the 6th of March and was completed at the 2# East Pit on the 17th of March. A total of 365m in 15 holes was drilled. The core recovery was remarkably good (>95%), although it was badly broken and deeply weathered in some holes. It was decided at that point to continue with the phase 2 drilling, using reverse circulation drilling methods. This second phase began on the 29th of March and was completed at the 2# East Pit on the 1st of April 1996. A total of 348m in 16 boreholes was drilled.

The core from the Phase 1 diamond drilling was logged and the UG2 layer sampled. Whole core sampling was performed as only this would provide sufficient mass for both assaying and metallurgical testing. The UG2 was crushed, pulverized and split. One half of the split of each borehole core was sent to Mintek for assay of individual PGM's, Au, Ni & Cu. The other half of the samples were composited and sent for metallurgical testing in-house. These composited samples were depth specific. All the samples from the shallow holes were composited; all the samples from the deep holes were composited and all the samples from the medium depth holes were composited, to provide three samples for the metallurgical test work.

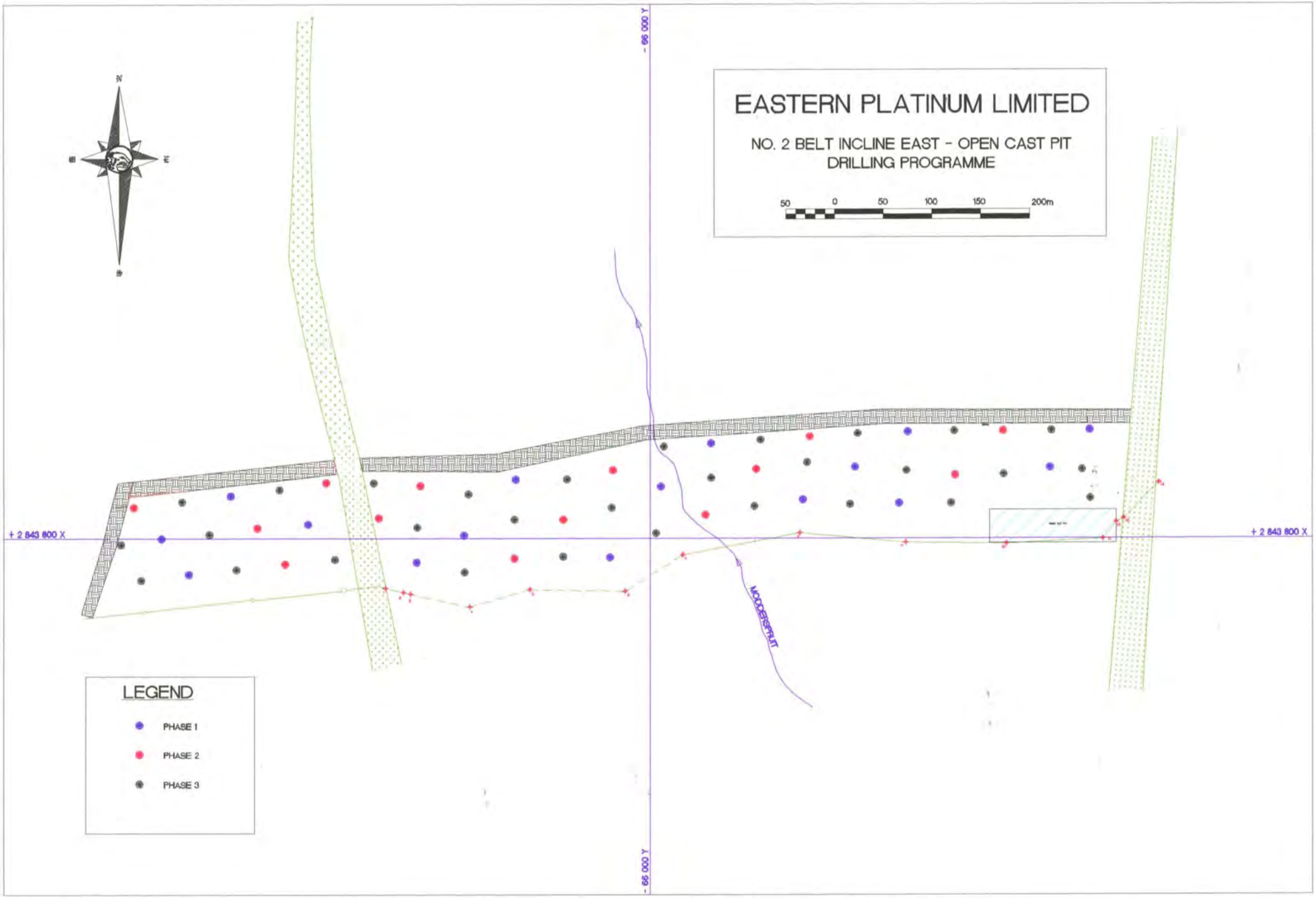


FIGURE 5 . 5 NO. 2 BELT INCLINE EAST - OPEN CAST PIT - DRILLING PROGRAMME

5.4.2 Drilling Results:

Of the 15 phase 1 diamond drill holes, six intersections of the UG2 contained **Split Reef.** This is a widespread phenomenon on EPL and is often, though not necessarily, associated with potholing of the reef horizon. In split reef areas, the UG2 contains lenses of either anorthosite (usually mottled in texture), feldspathic pyroxenite or rarely, pegmatoid (pegmatoidal feldspathic pyroxenite). They can cover areas of up to 200m x 200m and frequently will pothole towards their centre (Figure 5.6). The lens contained within the reef, be it anorthosite, pyroxenite or pegmatoid, never carries any value and therefore stoping such split reef requires careful control. The reverse circulation boreholes indicated two definite potholes and five split reef areas, one of which was definitely potholed. This data, as well as data from underground and other sources (ground-magnetics, trenching, etc.) was used to construct a structural map of the 2# East Pit (Figure 5.7). By convention, the split reef areas delimit only those areas where the internal waste is >50cm. A pothole was thought (from information about previous mining at the east end of the pit area) to exist on the eastern end of the pit, which was supported by ground magnetics interpretation. This has subsequently been confirmed by mining at the pit (Figure 5.7).

The assay results for the boreholes are given in Table 5.2. The drilling revealed a slightly steeper dip, of 15°, rather than the originally estimated 13°. The average reef thickness of 147.5cm (channel width) was also considerably more that the mineral resources estimate of 130cm. This dictated that the tonnage per unit area would be greater than anticipated. The average grade of the boreholes was almost identical to the predicted grade from the mineral resource estimate. This in itself was quite impressive as the pit area had no previous borehole information. The average grade of the boreholes was 5.37 g/t (corrected) as opposed to 5.4 g/t (corrected) as estimated. The Pt:Pd ratios are rather skewed. This is due to the mobility of Pd in the secondary environment. Some of the Pt:Pd ratios are as high as 10:1 (Table 5.2).

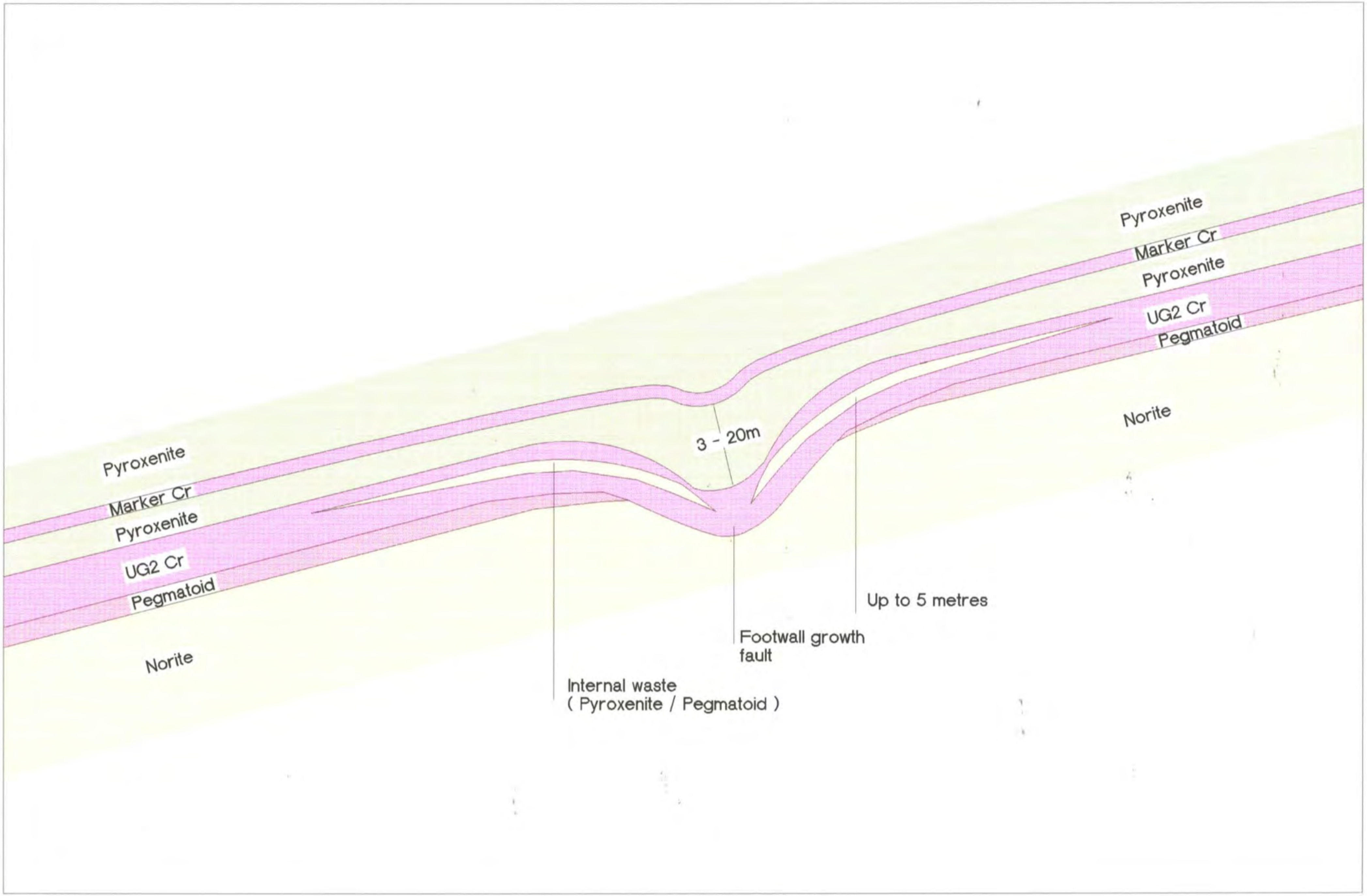


FIGURE 5 . 6 SCHEMATIC DIAGRAM (SECTION) OF A SPLIT REEF AREA WITH A POTHOLE

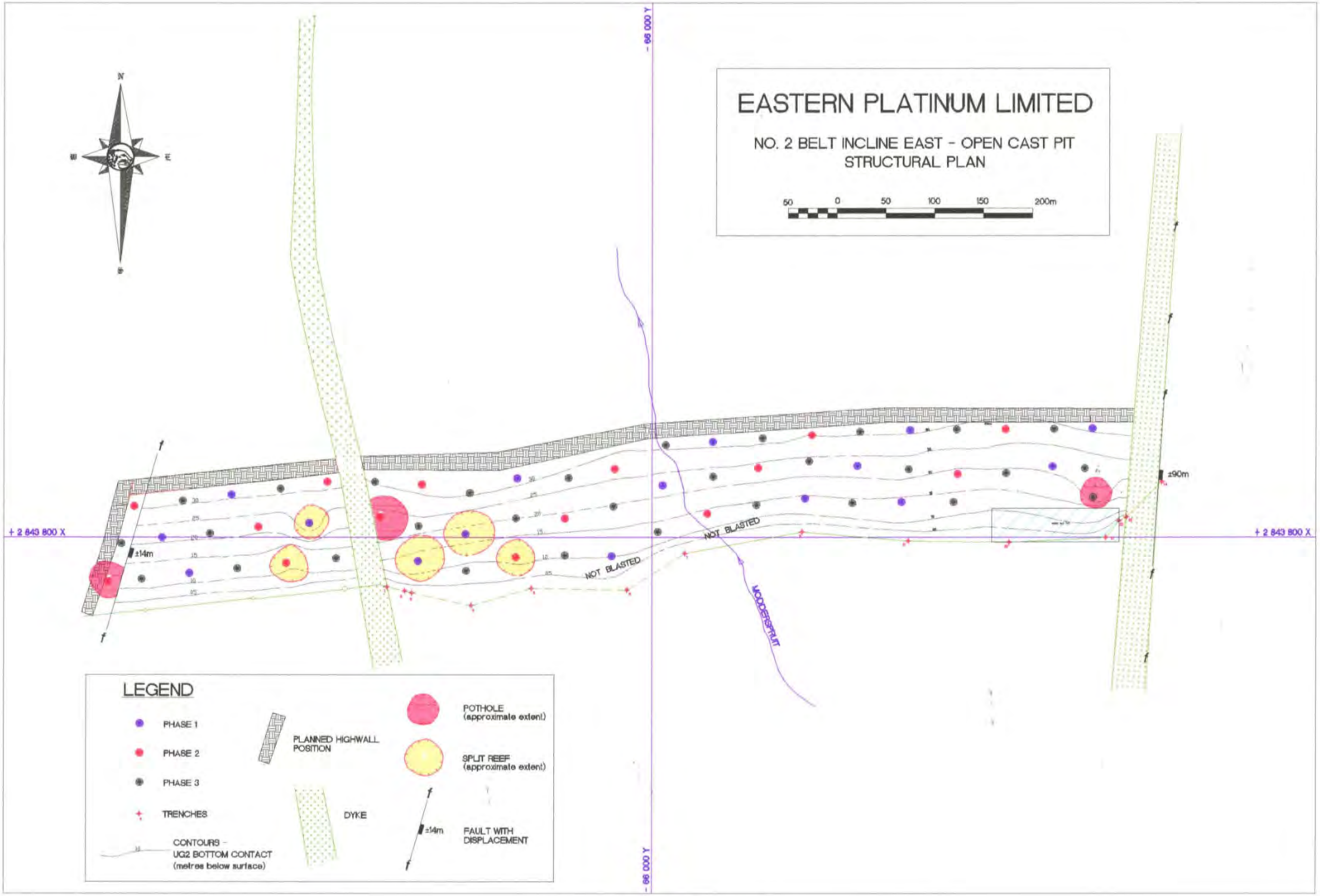


FIGURE 5 . 7 STRUCTURAL PLAN OF 2# EAST PIT

UG2Cr : PGM Splits

		g/t								%						Corr	Ratios	
Borehole	Assay Lab.	Pt	Pd	Rh	Au	Ru	Ir	3PGM+Au	5PGM+Au	Pt	Pd	Rh	Au	Ru	Ir	Fact	Pt : Pd	Pt : Rh
C1	Mintek	2.20	1.05	0.40	0.04	0.77	0.18	3.69	4.64	47.4	22.6	8.6	0.9	16.6	3.9	1.26	2.10	5.50
C2	Mintek	2.75	0.90	0.54	0.01	0.90	0.21	4.20	5.31	51.8	16.9	10.2	0.2	16.9	4.0	1.26	3.06	5.09
C3	Mintek	2.35	1.04	0.42	0.01	0.71	0.17	3.82	4.70	50.0	22.1	8.9	0.2	15.1	3.6	1.23	2.26	5.60
C4T	Mintek	2.60	0.45	0.39	0.00	0.76	0.18	3.44	4.38	59.4	10.3	8.9	0.0	17.4	4.1	1.27	5.78	6.67
C4B	Mintek	2.50	1.15	0.47	0.01	0.80	0.19	4.13	5.12	48.8	22.5	9.2	0.2	15.6	3.7	1.24	2.17	5.32
C5T	Mintek	2.90	1.35	0.66	0.01	0.89	0.24	4.92	6.05	47.9	22.3	10.9	0.2	14.7	4.0	1.23	2.15	4.39
C5B	Mintek	1.80	0.18	0.27	0.01	0.74	0.17	2.26	3.17	56.8	5.7	8.5	0.3	23.3	5.4	1.40	10.00	6.67
C6T	Mintek	3.30	2.10	0.56	0.06	1.00	0.23	6.02	7.25	45.5	29.0	7.7	0.8	13.8	3.2	1.20	1.57	5.89
C6B	Mintek	2.70	1.10	0.40	0.02	0.86	0.18	4.22	5.26	51.3	20.9	7.6	0.4	16.3	3.4	1.25	2.45	6.75
C7	Mintek	2.60	1.50	0.48	0.03	0.90	0.20	4.61	5.71	45.5	26.3	8.4	0.5	15.8	3.5	1.24	1.73	5.42
C8	Mintek	2.75	1.45	0.51	0.01	0.89	0.20	4.72	5.81	47.3	25.0	8.8	0.2	15.3	3.4	1.23	1.90	5.39
C9	Mintek	2.40	0.76	0.41	0.01	0.78	0.18	3.58	4.54	52.9	16.7	9.0	0.2	17.2	4.0	1.27	3.16	5.85
C10	Mintek	2.40	1.15	0.42	0.02	0.78	0.18	3.99	4.95	48.5	23.2	8.5	0.4	15.8	3.6	1.24	2.09	5.71
C11	Mintek	2.75	0.86	0.48	0.01	0.90	0.20	4.10	5.20	52.9	16.5	9.2	0.2	17.3	3.8	1.27	3.20	5.73
C12	Mintek	2.45	1.05	0.42	0.02	0.85	0.20	3.94	4.99	49.1	21.0	8.4	0.4	17.0	4.0	1.27	2.33	5.83
C13	Mintek	2.95	1.00	0.54	0.01	0.96	0.23	4.50	5.69	51.8	17.6	9.5	0.2	16.9	4.0	1.26	2.95	5.46
C14	Mintek	2.80	1.65	0.52	0.02	0.91	0.20	4.99	6.10	45.9	27.0	8.5	0.3	14.9	3.3	1.22	1.70	5.38
C16	Mintek	3.30	2.55	0.56	0.05	1.10	0.24	6.46	7.80	42.3	32.7	7.2	0.6	14.1	3.1	1.21	1.29	5.89
Ave.		2.64	1.18	0.47	0.02	0.86	0.20	4.31	5.37	49.1	22.0	8.7	0.4	16.0	3.7	1.25	2.23	5.62

TABLE 5.2: 2# East Open Cast Pit: Borehole Assay Results.

5.4.3 Metallurgical Test Results

The metallurgical test results were very disappointing. The metallurgists were not even able to provide a percentage recovery for the shallow ore. The medium depth ore showed a recovery of only 53.8% and the deep ore only 57% (Table 3.2). However, it was pointed out by the Metallurgical Manager that these results were minimum expected recoveries, based on applying the standard flotation method as applied to underground ore, to open cast samples. No attempt had been made to adjust the reagents used. A further problem was noted in that there was considerable resistance, on the part of the metallurgists, to having to process open cast ore. It was felt that it would create a greater workload for them and further stretch a fairly limited resource: themselves. This was complicated by identifying the relative positions of the samples and by not having had a control sample supplied by the geologist.

At this point the Metallurgical Manager indicated a fair level of confidence in obtaining at least a 60% recovery. To maintain a degree of conservatism it was decided to use the metallurgical test result percentages of 54% for the shallow ore and 57% for the deep ore.

5.4.4 Mineral Resource Statement

In the underground environment, split reef areas (Figure 5.6) may be stoped where the internal waste (lens) is less than 50cm thick. Therefore, areas of split reef where the internal waste is less than 50cm in thickness are included in the ore resource statement. The grade is calculated over the entire width, including the internal waste, which effectively dilutes the grade.

This same procedure was applied to the borehole data. Only two of the six split-reef boreholes contained split reef that was less than 50cm in thickness. Normally the boreholes with internal waste greater than 50cm would not be utilized in the estimation of the mineral resource, and would be written off as a geological loss. Due to the mining method employed in open cast mining, where the reef is essentially mined from above rather than from the side, as is the case in underground mining, there was a logical and intuitive argument to include at least the uppermost portion of split reef areas in the mineral resource statement. If the split reef area was potholed, the geological loss would be catered for by the normal allowances. Where the internal waste of split reef areas was sufficiently thick, it was felt that it might even be possible to remove the internal waste and access the bottom portion of the reef as well, thus resulting in no loss, or a much reduced loss, of potential tonnage.

In order that the entire open cast project data be directly comparable to data from other, competing, projects, the mineral resource statement was estimated in exactly the same manner as for underground ore, with the exception noted above. This data is presented in Figure 5.8. Obviously, in an open cast environment, there would be no need to leave pillars, so there are no losses attributed to support. The mineral resource estimate is based on 4 PGM's + Au, and the correction factor is a weighted average of the correction factors of the assay results.

The mineral resource statement provides the pit tonnage, an estimate of tons to be sent to the mill (allowing for geological losses and dilution), an estimated mill head-grade as well as an estimate of the kilograms of PGM produced in concentrate.

5.4.5 Mining Costs

In all, five open cast mining contractors were approached and requested to provide quotations for the mining of the open pits. There were some stipulations from management, however. The primary and most important stipulation was that EPL wanted only one monthly billable item: tons of ore delivered as measured by the mine's weigh bridge. The company did not wish to have to perform survey measurements in the pits (see Chapter 2). This had been a major point of contention when the previous open pits had been mined. Furthermore it was felt that billing in any other way made it difficult to fix a mining cost per ton of ore, as in any month material (waste) will be stripped from above reef that will only be mined at some later time. Secondly, the cost must include rehabilitation of the open cast area after mining has finished, but not revegetation. As EPL has extensive experience of re-vegetating open pits, it was felt that it could be performed more cost effectively by the mine. Thirdly the cost must be a "loaded" cost. EPL wished to award the contract of the ore to the local Bapo Ma-Mogale tribe, on whose behalf EPL mines, as part of an ongoing socio-economic programme. The mining cost must therefore include the cost of loading the ore from a stockpile at the pit onto the contractor's transport trucks.

dissert - 1/9/97

EASTERN PLATINUM LIMITED

UG2 2# OPENCAST

MINERAL RESOURCES STATEMENT FOR 2# EASTOPEN PIT AS AT 30 MAY 1996

BLOCK	IN SITU SQUARE METRES NOT DISCOUNTED OR ADJUSTED			GROUND LOSSES (PERCENTAGES)		AVAILABLE FOR STOPING (IN SITU UNCORRECTED GRADE)							FACTOR FOR TONS MILLED PER SQUARE METRE BROKEN	ESTIMATED TONNES SENT TO MILL	FACTOR OF IN SITU GRADE TO BUILT UP HEAD GRADE	UNCORRECTED		25.0% CORR MILL HEAD GRADE	
	HORI-ZONTAL	DIP DEGR	ON PLANE OF REEF	GEO-LOGICAL	SUPPORT	SQUARE METRES	CHANNEL WIDTH (cms)	DENSITY	TONNES	CHANNEL VALUE PGM. (g/t)	cmg/t	GRAMS PER SQUARE METRE				MILL HEAD GRADE	CONTENTS (KGS)		
PROVED MINERAL RESERVE	*	*	*	%	%		*	*		*	*				%				
2E1	1,855	15	1,920	10.00%	0.00%	1,728	158.9	3.790	10,409	3.72	591	22.4	6.7181	11,611	0.8179	3.04	35.329	3.80	(pay)
2E2	4,450	15	4,607	10.00%	0.00%	4,146	137.3	3.790	21,576	3.76	516	19.6	5.9024	24,473	0.8179	3.08	75.262	3.84	(pay)
2E3	4,685	15	4,850	10.00%	0.00%	4,365	107.4	3.790	17,769	4.05	435	16.5	4.7752	20,845	0.8179	3.31	69.049	4.14	(pay)
2E4	4,175	15	4,322	10.00%	0.00%	3,890	108.0	3.790	15,923	4.46	482	18.3	4.7978	18,664	0.8179	3.65	68.082	4.56	(pay)
2E5	3,020	15	3,127	10.00%	0.00%	2,814	141.3	3.790	15,069	4.53	640	24.3	6.0534	17,033	0.8179	3.71	63.110	4.63	(pay)
2E6	3,530	15	3,655	10.00%	0.00%	3,289	139.4	3.790	17,377	4.32	602	22.8	5.9816	19,674	0.8179	3.53	69.515	4.42	(pay)
2E7	5,460	15	5,653	10.00%	0.00%	5,087	147.9	3.790	28,517	4.47	661	25.1	6.3026	32,063	0.8179	3.66	117.224	4.57	(pay)
2E8	5,430	15	5,622	10.00%	0.00%	5,059	148.8	3.790	28,533	4.43	659	25.0	6.3365	32,059	0.8179	3.62	116.160	4.53	(pay)
2E9	5,110	15	5,290	10.00%	0.00%	4,761	176.1	3.790	31,777	4.36	768	29.1	7.3681	35,081	0.8179	3.57	125.102	4.46	(pay)
2E10	4,395	15	4,550	10.00%	0.00%	4,095	170.2	3.790	26,415	4.71	802	30.4	7.1451	29,259	0.8179	3.85	112.716	4.82	(pay)
2E11	3,345	15	3,463	10.00%	0.00%	3,117	189.3	3.790	22,361	4.93	933	35.4	7.8672	24,520	0.8179	4.03	98.870	5.04	(pay)
2E12	5,150	15	5,332	10.00%	0.00%	4,799	165.3	3.790	30,062	3.74	618	23.4	6.9599	33,397	0.8179	3.06	102.160	3.82	(pay)
2E13	7,505	15	7,770	10.00%	0.00%	6,993	156.5	3.790	41,477	3.79	593	22.5	6.6274	46,344	0.8179	3.10	143.659	3.87	(pay)
2E14	7,290	15	7,547	10.00%	0.00%	6,792	104.9	3.790	27,005	4.06	426	16.1	4.6811	31,796	0.8179	3.32	105.585	4.15	(pay)
2E15	8,400	15	8,696	10.00%	0.00%	7,827	51.1	3.790	15,158	4.86	248	9.4	2.6720	20,913	0.8179	3.97	83.129	4.97	(pay)
2E16	9,190	15	9,514	10.00%	0.00%	8,563	115.6	3.790	37,516	4.55	526	19.9	5.0840	43,533	0.8179	3.72	162.007	4.65	(pay)
2E17	9,235	15	9,561	10.00%	0.00%	8,605	163.6	3.790	53,353	4.10	671	25.4	6.8957	59,335	0.8179	3.35	198.973	4.19	(pay)
2E18	6,320	15	6,543	10.00%	0.00%	5,889	147.6	3.790	32,941	4.35	642	24.3	6.2912	37,047	0.8179	3.56	131.808	4.45	(pay)
2E19	5,660	15	5,860	10.00%	0.00%	5,274	140.4	3.790	28,062	4.50	632	23.9	6.0194	31,744	0.8179	3.68	116.837	4.60	(pay)
2E20	6,865	15	7,107	10.00%	0.00%	6,396	146.5	3.790	35,515	5.30	776	29.4	6.2497	39,976	0.8179	4.33	173.290	5.42	(pay)
2E21	6065	15	6,279	10.00%	0.00%	5,651	163.0	3.790	34,911	4.87	794	30.1	6.8730	38,840	0.8179	3.98	154.705	4.98	(pay)
2E22	4615	15	4,778	10.00%	0.00%	4,300	179.3	3.790	29,221	4.89	877	33.2	7.4891	32,203	0.8179	4.00	128.798	5.00	(pay)
TOTAL OPENCAST	121,750	15	126,045	10.00%	0.00%	113,440	147.5	3.790	600,945	4.41	650	23.3	5.9980	680,412	0.8178	3.60	2,451.367	4.50	(pay)
TOTAL 2# O/CAST	121,750	15	126,045	10.00%	0.00%	113,440	147.5	3.790	600,945	4.41	650	23.3	5.9980	680,412	0.8176	3.60	2,451.367	4.50	(pay)

EPUG2# * Pay Limit applied to Mill Head Grade = 3.763 g/t corrected p:\home\technica\opencast\2#east\epug2#.wk1 11:08 AM 30-May-96

FIGURE 5.8

Accommodation for the mining contractors' personnel would be provided by EPL at the existing hostel. This prevented the situation where EPL would invoice the contractor for accommodation costs, only to have it built into the cost/t, with the addition of a 20% administration fee. Although this cost is borne by EPL, it forms part of the sundry costs (variable) in the financial model. Various other operational stipulations were also made to the contractors.

Obviously the first stipulation means that there is an increased risk to the contractor and indirectly to the geologist. It requires that the confidence level in, above all, the tonnage estimates must be high. If there is less tonnage in the pit than estimated, for whatever reason, the contractor's profit margin will be reduced. If there is more tonnage in the pit, the contractor will be quite happy as he can deliver more ore tonnage for the same amount of waste stripping, but the client will pay an unnecessarily high cost per ton for every ton of ore.

In fact so onerous is this stipulation that one reputable contractor withdrew from the tender process because they were not prepared to carry that risk. The other contractors tendered on that basis and the costs ranged from ± R34/t of ore to ± R56/t of ore.

5.4.6 Sundry Costs

Some expenditure had at this stage been incurred by EPL, in the form of drilling and assay costs and some environmental work in terms of the Environmental Mine Programme Report required by law, would be necessary (Chapter 4). These were now defined as Sunk Costs. If the pit did not proceed these costs would be non-recoverable. In addition to this, and if the open cast mining costs went ahead, additional working costs would be incurred. These costs would be made up of in-pit sampling (assay costs), revegetation of the mined out areas, a grade control officer to control the ore grade as best as possible during mining, and damage provision for any damage that might occur to dwellings close to the proposed pit areas as a result of blasting. Additionally, the cost of hostel accommodation for the mining contractor's personnel would form part of these costs. These costs would vary in proportion to the tonnage removed, and are therefore deemed Variable Costs. A breakdown of these sundry costs is presented in Figure 5.9. The sunk costs and variable costs were estimated, and apportioned per ton of ore and appear on the template as Sundry Costs. At this stage sufficient information

PROPOSED 2# EAST OPEN-CAST

Sundry Costs Breakdown

	Rand
MINING R / ton (of ore)	35.34
TRANSPORT / ton	4.92
MILLING / ton	17.44
SMELTING / ton	6.50
SUB-TOTAL:	**64.20**
SUNK COSTS:	
DRILLING & DELINEATION	85,867
EMPR BOREHOLES	53,770
SUB-TOTAL:	**139,637**
SUNK COST/TON	**0.21**
VARIABLE COSTS	
SAMPLING/VALUATION	12,667
HOSTEL COSTS	240,000
RE-VEGETATION	22,249
DAMAGE PROVISION	158,147
GRADE-CONTROL/GEOLOGIST	236,444
TOTAL VARIABLE COSTS:	**669,507**
TOTAL SUNDRY COSTS:	**809,144**
TONS (ORE)	680,412
VARIABLE COST / ton	**0.98**
TOTAL COSTS / ton	**65.39**

FIGURE 5.9

with respect to costs, grade, tonnage and recoveries was available to return to the financial model template and re-examine the financial viability of the proposed open pits.

5.5 Financial Model Template:

All of the foregoing factors were finalized as follows:

Reserve: 704 624t as estimated by the spreadsheet template. This estimated that some 697 597t of ore would be sent to the mill and was acceptably close to the Mineral Resource Statement (Figure 5.8) estimated tonnage sent to the mill of 680 412t.

Geological losses: Due to the different mining method that would be employed, in open cast as opposed to underground mining, it was felt that there was considerable justification in expecting a greater extraction percentage from the open cast than is the case from underground. Because of the mining method, it is very often impossible (either physically or in terms of cost) to, for example, mine the base of a pothole, or often even the flanks of a pothole. In an open cast environment, however, potholes could be more extensively mined, except if they occur at or near the highwall. It was therefore decided to apply a factor of 10% for geological losses.

Mining Dilution: It is essential to extract the bottom contact of the UG2, as the lowest portion of the UG2 contains the highest grade, frequently >20 g/t. It was therefore decided to allow for 10% dilution, essentially of footwall material. In underground stoping operations a similar footwall overbreak allowance is applied, and based on extensive sampling data, a value of 0.3 g/t is ascribed to this dilution. This value was also applied to dilution for the open cast ore.

Recovery: A recovery of 54% was applied to the shallow ore (0-15m depth) and 57% to the deep ore (15-30m) as per the metallurgical test results (Table 3.2). A smelter recovery of 99% was applied.

Sundry Costs: The sundry costs were estimated for the open cast and amounted to R809 144. A breakdown of these costs is presented in Figure 5.9.

Grade: The grade used is the estimated grade from the mineral resource statement (Figure 5.8) of 4.41 g/t (3 PGM's + Au) which was then corrected by the correction factor of 25%, giving a 5 PGM + Au grade of 5.51 g/t.

Matte Valuation: At this stage it was decided to use a Matte Valuation of R33 208/kg, the April 1996 matte valuation figure based on the then current PGM prices and a $:R exchange rate of $4.3725:R1. This matte valuation includes Rh.

Mining Costs: The tendered mining cost of R35.34/t of ore used in the template.

Note: At this point it was discovered that the previously used smelting cost/t was wrong. Where a cost of R5.00/t had been used in the first pass analysis, the final analysis used a cost of R6.50/t of ore.

The formulae used in the financial template are presented in Appendix 1.

5.5.1 Results:

The financial template was run using all the above factors and the results are presented in Figure 5.10. This template also shows the estimated costs and profits for average recoveries of 59%-62%. These were included because the recovery percentage predicted were deemed to be the least dependably defined of all the factors. Despite the conservatism applied in all of these factors, the results were very promising. The gross revenue was estimated to be R64 103 332 against a gross cost of R 45 593 636, giving a gross profit of R18 509 696. No allowance was made for tax, as all financial estimates performed within the Lonrho Group are done in gross terms. The cost/kg of PGM in matte was estimated to be R23 619, against a matte valuation of R33 208, giving a gross profit/kg of R9 589. The margin, calculated on the basis of gross profit expressed as a percentage of gross revenue, is 29%. Some other analyses were attempted, such as calculating an IRR, but the result (8787%) were felt to be somewhat meaningless. Two specific problems exist in calculating the IRR: firstly the short life of the pit (± 18 months) and secondly the very low capital outlay ±R140 000 which combine to make these measures effectively useless. An NPV was calculated and the result, at 10% p.a. discount rate, amounted to R16 177 450.

FIGURE 5.10

2# EAST OPEN CAST: FINAL FINANCIAL TEMPLATE

PRESUMPTIONS: Matte Valuation R33 208/kg
Dilution grade 0.3 g/t
Mining costs R35.34/t of ore
Mining Depth: 30mbs
Rand/$: R4.3725:$1

		Predicted Recovery	59% Recovery	60% Recovery	61% Recovery	62% Recovery
RESERVES (tons)	0-15 m	352,312	352,312	352,312	352,312	352,312
GEOLOGICAL LOSS	10.0%	35,231	35,231	35,231	35,231	35,231
MINEABLE TONS	(ORE)	317,081	317,081	317,081	317,081	317,081
MINING DILUTION	10%	31,708	31,708	31,708	31,708	31,708
% RECOVERY		54%	59%	60%	61%	62%
PGE RECOVERED	kg	949	1036	1054	1072	1089
RESERVES (tons)	15-30 m	352312	352312	352312	352312	352312
GEOLOGICAL LOSS	10.0%	35231	35231	35231	35231	35231
MINEABLE TONS	(ORE)	317081	317081	317081	317081	317081
MINING DILUTION	10%	31708	31708	31708	31708	31708
% RECOVERY		57%	59%	60%	61%	62%
PGE RECOVERED	kg	1001	1036	1054	1072	1089
REEF TONS	TOTAL	704,624	704,624	704,624	704,624	704,624
MILLED TONS		697,578	697,578	697,578	697,578	697,578
TONS MINED		5,120,283	5,120,283	5,120,283	5,120,283	5,120,283
PGE kg in conc.		1950	2073	2108	2143	2178
PGE GRADE	g/t	5.51	5.51	5.51	5.51	5.51
PGE kg in matte	kg	1930	2052	2087	2122	2156
MATTE VALUATION	R/kg	33,208	33,208	33,208	33,208	33,208
ON-MINE COSTS						
MINING/t	Rand	35.34	35.34	35.34	35.34	35.34
MILLING/t	Rand	17.44	17.44	17.44	17.44	17.44
SMELTING COST/t	Rand	6.50	6.50	6.50	6.50	6.50
TRANSPORT @42c/t/km	Rand	4.92	4.92	4.92	4.92	4.92
TOTAL COSTS/t OF ORE	Rand	65.36	65.36	65.36	65.36	65.36
REVENUE/t	Rand	91.89	97.69	99.35	101.00	102.66
SUNDRY COSTS	Rand	809,144	809,144	809,144	809,144	809,144
GROSS REVENUE	Rand	64,103,332	68,145,884	69,300,899	70,455,914	71,610,929
GROSS COSTS	Rand	45,593,636	45,593,636	45,593,636	45,593,636	45,593,636
GROSS PROFIT/t	Rand	26.53	32.33	33.99	35.64	37.30
GROSS PROFIT (TOTAL)	Rand	18,509,696	22,552,248	23,707,263	24,862,278	26,017,293
COST/kg	Rand	23,619	22,218	21,848	21,490	21,143
PROFIT/kg	Rand	9,589	10,990	11,360	11,718	12,065
MARGIN %	Rand	29%	33%	34%	35%	36%

Due to the short life of the pit the next step was to produce a month by month cash flow for the pit. Despite the effort and cost expended on defining the mineral resource of the pit, conservatism was urged in the estimation of a cash flow. Experience with the two previous pits had shown that the actual tonnage extracted always was less that was predicted. This meant that some of the sunk costs and even some of the variable costs were left uncovered. For this reason it was decided to use the more conservative tonnage figure from the mineral resource statement, of 680 412t, and to cover all of the sundry costs (based on the estimated tonnage) in the first half of the pit's life. At a required tonnage of 40 000t/month delivered, the pit had a life of ±18 months. The cash flow must also reflect the build-up of tonnage from the pit and the run-down of the tonnage towards the end of the pit life. This is presented in Figure 5.11. By loading all of the sundry costs onto the tonnage mined during the first ten months of the pit life, it was felt that the risk of mining out the pit (at a reduced tonnage) and still having outstanding sundry costs which had not been covered, was greatly reduced. The differences between the results on Figure 5.10 and Figure 5.11 are due to different tonnages being used.

However, it must be remembered that the open cast tonnage would be displacing potential underground tonnage, which, while having a similar grade, would yield higher recoveries. If the same tonnage as was expected from the open cast was sourced from underground it would yield 3 096kg in matte (5.36g/t x 704 624t x 82.8% x 99%). At R33 208/kg, this would provide a revenue of R102.81 million, with costs of R70.76 million, yielding a gross profit of R32.05 million. The theoretical difference in gross profit between this and the gross profit of the open cast is some R 13.54 million. This is based on the assumption that all factors would remain equal (matte valuation, mining costs, processing costs, etc.).

This meant that a strategic decision was required because, should the underground production recover, the logical action would be to close the pit and process the underground ore. To close the pit with only a portion of the tonnage extracted would effectively sterilize the remaining resource there, due to technicalities in the mining plan and EMPR requirements. A decision to proceed with the pit required commitment to see the mining through to completion. This required a close examination of production trends, planned production, public holidays, official shutdowns, etc., across the entire Lonrho Group platinum mines. This was done and it was decided to proceed with the 2# East Pit.

2# EAST OPEN PIT: MONTHLY CASH FLOW (CONSTANT MONEY TERMS)

		Milled Tons	Gross Costs	ew kg' n Matt	Cost/kg Rand	st/To Rand	Revenue per Month	Profit per Month	Profit per kg	Cumulative Profit
Month	1	5,000	332,900	13.699	24,301	66.58	454,916	122,016	8,907	122,016
Month	2	20,000	1,331,600	54.796	24,301	66.58	1,819,666	488,066	8,907	610,082
Month	3	35,000	2,330,300	95.893	24,301	66.58	3,184,415	854,115	8,907	1,464,197
Month	4	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	2,440,328
Month	5	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	3,416,459
Month	6	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	4,392,590
Month	7	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	5,368,721
Month	8	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	6,344,852
Month	9	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	7,320,984
Month	10	40,000	2,663,200	109.592	24,301	66.58	3,639,331	976,131	8,907	8,297,115
Month	11	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	9,368,446
Month	12	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	10,439,777
Month	13	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	11,511,108
Month	14	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	12,582,439
Month	15	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	13,653,770
Month	16	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	14,725,101
Month	17	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	15,796,433
Month	18	40,000	2,568,000	109.592	23,432	64.20	3,639,331	1,071,331	9,776	16,867,764
Month	19	20,412	1,310,450	55.925	23,432	64.20	1,857,157	546,707	9,776	17,414,471
TOTALS:		680,412	44,491,650	1864			61,906,121	17,414,471		17,414,471

* COST/TON: Includes mining costs, sunk costs, variable costs,transport, milling costs and smelting

FIGURE 5.11

5.6 UG2 and Merensky Open Cast Pits: A Comparison.

To date, Lonrho has not mined any of their extensive Merensky reserves by open cast methods. The reasons for this are manifold, but the overriding consideration has been the mineralogy of the Merensky Reef in the near surface environment. There is also a large degree of "wait-and-see" what their competitors will do.

An area of Merensky Reef, thought to be suitable for open cast mining was delineated on EPL, although at present EPL does not mine the Merensky Reef from underground. As EPL does not produce any Merensky Reef, a surcharge is levied by the smelter to cover the cost of sulfides added in the smelting stage. The reasons for this are complex and due to internal costing structures. Were EPL to produce its own Merensky reef, this surcharge would be dropped or reduced, and the overall cost of smelting would be reduced. There was also a problem with keeping one of the Merensky plants full, due to various operational problems, so a "window of opportunity" existed.

An analysis, similar in most respects to that for the UG2, was performed (Figure 5.12). Initially, the metallurgists felt that recovery could be as high as 84%. After discussions with Amplats, on their Merensky open cast operations at Bleskop, it was felt that it would be more realistic to apply a recovery of 64% as WPL could batch mill open cast ore (Chapter 2). With a Matte Valuation ±50% higher than that of the UG2 (Table 5.3), it was expected that the Merensky would be a viable open cast project. The mining costs worked out to some R56/t of ore. Grades (in situ) were available from the mineral resource document.

Initially, the results looked promising (Figure 5.12) and the go-ahead was given to perform diamond drilling and to define the mineral resource, along the same lines as for the UG2. The results were, however, very disappointing, giving an average grade for the boreholes drilled of 2.49 g/t over a cut of 100cm. The area is also transected by a number of large (40m wide) dykes, which appear to have metasomatized the adjoining Merensky Reef, causing extensive replacement and alteration. The sulfide content looked reasonably good in fresh intersections. This information was fed back into the financial template (Figure 5.13), and it was realized that the project was not viable. At this point in time a strategic decision was taken to concentrate on the UG2 projects.

FIGURE 5.12

FIRST PASS FINANCIAL ANALYSIS OF PROPOSED MERENSKY OPEN CAST PIT

PRESUMPTIONS:

Matte Valuation R39 000/kg (February 1996)
Dilution grade 0.3 g/t
Mining costs R55.00/t of ore
Mining Depth: 25mbs
R:$ Exchange Rate: R3.7:$1

			64% Recovery	65% Recovery	65% Recovery	66% Recovery	67% Recovery
RESERVES	(TONS)	0-15 m	383,590	383,590	383,590	383,590	383,590
GEOLOGICAL LOSS		10%	38,359	38,359	38,359	38,359	38,359
MINEABLE TONS		(ORE)	345,231	345,231	345,231	345,231	345,231
MINING DILUTION		10%	34,523	34,523	34,523	34,523	34,523
% RECOVERY			63%	64%	65%	66%	67%
PGE RECOVERED		kg	1037	1054	1070	1087	1103
RESERVES	(TONS)	15-25 m	255,726	255,726	255,726	255,726	255,726
GEOLOGICAL LOSS		10%	51,145	51,145	51,145	51,145	51,145
MINEABLE TONS		(ORE)	204,581	204,581	204,581	204,581	204,581
MINING DILUTION		10%	20,458	20,458	20,458	20,458	20,458
% RECOVERY			63%	64%	65%	66%	67%
PGE RECOVERED		kg	615	625	634	644	654
REEF TONS		TOTAL	639,316	639,316	639,316	639,316	639,316
MILLED TONS			604,793	604,793	604,793	604,793	604,793
TONS MINED			8,676,215	8,676,215	8,676,215	8,676,215	8,676,215
PGE in conc.		kg	1,652	1,678	1,705	1,731	1,757
PGE GRADE		g/t	4.74	4.74	4.74	4.74	4.74
PGE in matte		kg	1636	1662	1688	1714	1740
MATTE VAL $/oz		Rand	328.86	328.86	328.86	328.86	328.86
ON-MINE COSTS							
MINING/t		Rand	4.05	4.05	4.05	4.05	4.05
MILLING/t		Rand	15.97	15.97	15.97	15.97	15.97
SMELTING COSTS/t		Rand	15.96	15.96	15.96	15.96	15.96
TRANSPORT @42c/t/km		Rand	6.30	6.30	6.30	6.30	6.30
TOTAL COSTS/t OF ORE		Rand	96.37	96.37	96.37	96.37	96.37
REVENUE/t		Rand	105.48	107.15	108.83	110.50	112.18
SUNDRY COSTS		Rand	572,927	572,927	572,927	572,927	572,927
GROSS REVENUE		Rand	63,792,965	64,805,552	65,818,138	66,830,725	67,843,312
GROSS COSTS		Rand	58,856,543	58,856,543	58,856,543	58,856,543	58,856,543
GROSS PROFIT/t		Rand	8.16	9.84	11.51	13.18	14.86
GROSS PROFIT (TOTAL)		Rand	4,936,422	5,949,009	6,961,596	7,974,182	8,986,769
Cost/kg		Rand	35,982	35,420	34,875	34,347	33,834
Profit/kg		Rand	3,018	3,580	4,125	4,653	5,166
Margin %		Rand	8%	10%	12%	14%	15%

FIGURE 5.13

FINAL FINANCIAL ANALYSIS OF PROPOSED MERENSKY OPEN CAST PIT

PRESUMPTIONS:
Matte Valuation R39 000/kg (February 1996)
Dilution grade 0.3 g/t
Mining costs R55.00/t of ore
Mining Depth: 25mbs
R:$ Exchange Rate: R3.7:$1

			64% Recovery	65% Recovery	65% Recovery	66% Recovery	67% Recovery
RESERVES	(TONS)	0-15 m	383,590	383,590	383,590	383,590	383,590
GEOLOGICAL LOSS		10%	38,359	38,359	38,359	38,359	38,359
MINEABLE TONS		(ORE)	345,231	345,231	345,231	345,231	345,231
MINING DILUTION		10%	34,523	34,523	34,523	34,523	34,523
% RECOVERY			63%	64%	65%	66%	67%
PGE RECOVERED		kg	548	557	565	574	583
RESERVES	(TONS)	15-25 m	255,726	255,726	255,726	255,726	255,726
GEOLOGICAL LOSS		10%	51,145	51,145	51,145	51,145	51,145
MINEABLE TONS		(ORE)	204,581	204,581	204,581	204,581	204,581
MINING DILUTION		10%	20,458	20,458	20,458	20,458	20,458
% RECOVERY			63%	64%	65%	66%	67%
PGE RECOVERED		kg	325	330	335	340	345
REEF TONS		TOTAL	639,316	639,316	639,316	639,316	639,316
MILLED TONS			604,793	604,793	604,793	604,793	604,793
TONS MINED			8,676,215	8,676,215	8,676,215	8,676,215	8,676,215
PGE in conc.		kg	873	887	901	914	928
PGE GRADE		g/t	2.49	2.49	2.49	2.49	2.49
PGE in matte		kg	864	878	892	905	919
MATTE VAL $/oz		Rand	328.86	328.86	328.86	328.86	328.86
ON-MINE COSTS							
MINING/t		Rand	4.05	4.05	4.05	4.05	4.05
MILLING/t		Rand	15.97	15.97	15.97	15.97	15.97
SMELTING COSTS/t		Rand	15.96	15.96	15.96	15.96	15.96
TRANSPORT @42c/t/km		Rand	6.30	6.30	6.30	6.30	6.30
TOTAL COSTS/t OF ORE		Rand	96.37	96.37	96.37	96.37	96.37
REVENUE/t		Rand	55.72	56.61	57.49	58.38	59.26
SUNDRY COSTS		Rand	572,927	572,927	572,927	572,927	572,927
GROSS REVENUE		Rand	33,701,944	34,236,895	34,771,847	35,306,798	35,841,750
GROSS COSTS		Rand	58,856,543	58,856,543	58,856,543	58,856,543	58,856,543
GROSS PROFIT/t		Rand	-41.59	-40.71	-39.82	-38.94	-38.05
GROSS PROFIT (TOTAL)		Rand	-25,154,599	-24,619,647	-24,084,696	-23,549,745	-23,014,793
Cost/kg		Rand	68,109	67,045	66,013	65,013	64,043
Profit/kg		Rand	-29,109	-28,045	-27,013	-26,013	-25,043

5.6.1 Merensky Open Cast on Western Platinum Ltd.

An exercise, similar in most respects to that performed on the UG2 at EPL, was performed on a potential Merensky open cast area on Western Platinum Mine. The area was drilled and the boreholes sampled. Due to core loss in the diamond drill holes there was little confidence in the results of this drilling. The areas was therefore trenched down to a depth of ±7m and the Merensky sampled in these trenches. The sampling results from the trenches were reasnonable and it was decided to obtain a bulk sample (±5 000t) to enable metallurgical testwork to be performed on a production scale.

The Merensky Reef, being a feldspathic pyroxenite or pegmatoidal feldspathic pyroxenite, tends to form clay minerals on weathering, which have a deleterious effect on the flotation process, and therefore creates major problems for the metallurgists (see Chapter 3). Another problem identified is that the proportions of metals contained in the weathered environment tends to be quite different to the Merensky in the unweathered environment. On first examination of these differences it could be thought that this process has in fact improved the oxidized Merensky Reef (Table 5.3). The Pt and Au content of the ore is enriched at the expense of Pd. In terms of revenue, this should improve the matte valuation for this ore. Pt forms some 77% of the contained metal and Au forms 10% of the contained metal, while Pd

Table 5.3

Comparative Merensky & UG2 Matte Valuations

	Merensky Underground Ore	Merensky Open Cast Ore	UG2 Underground Ore	UG2 Open Cast Ore
Pt \$380.50/oz	59.5%	77.5%	49.5%	48.6%
Pd \$116.00/oz	25.0%	10.2%	23.8%	22.6%
Au \$379.45/oz	5.0%	10.0%	0.4%	0.4%
Rh \$240.00/oz	2.9%	0.4%	8.7%	8.7%
Ru \$50.00/oz	6.2%	1.0%	14.0%	16.0%
Ir \$65.00/oz	1.4%	1.0%	3.6%	3.7%
Ni \$7,162.50/t	265.44 kg/kg PGM	27.92 kg/kg PGM	23.56 kg/kg PGM	24.62 kg/kg PGM
Cu \$1941.50/t	181.66 kg/kg PGM	20.94 kg/kg PGM	13.06 kg/kg PGM	12.41 kg/kg PGM
Matte Valuation	**R43,559 /kg**	**R47,491 /kg**	**R32,663 /kg**	**R32,101 /kg**

forms just over 10%, and Rh is also low. As Pt and Au are the two most valuable metals, the matte valuation was in excess of R47 000/kg at the then current metal prices.

Other problems also exist. The hanging wall succession of the Merensky Reef is a sequence of mottled and spotted anorthosites, which are remarkably fresh right up to outcrop. Indeed, they tend to form the only outcrop close to the Merensky Reef. This hanging wall succession requires blasting from outcrop which increases the mining costs. The areas of Merensky Reef available for open cast mining are of the Western Platinum Merensky facies (Davey, 1993), which is a relatively thick (3-5m) feldspathic pyroxenite, within which the PGM mineralization is essentially cryptic. This means that grade control within the pit would be required to determine the selection of ore from within a homogeneous rock type that has been shattered and confused by blasting and removal of the hanging wall succession.

The next problem, that of the degree to which the Merensky Reef is weathered in the near surface environment, causes several problems. Firstly, there is the problem of proving mineral resources as diamond drill core recovery is generally quite poor. Secondly, and as discussed previously, the problem of selective mining of a weathered and highly friable ore zone, having blasted the overlying hanging wall rocks, is extremely difficult. Thirdly, the metallurgical problems of extracting the PGM's in the virtual absence of sulfides must be overcome.

The drilling problem could be overcome possibly by large hole diametre drilling or reverse circulation drilling. The later was attempted on the Merensky Reef but unfortunately at the time of writing, those results are not yet available.

The virtual absence (Table 5.3) of sulfides would mean that visual identification of ore will be almost impossible. All of these factors would mean that extensive drilling and sampling (of SMU's in the region of 5 x 5 x 1m) would have to be conducted and that the mining cycle would therefore have to be 2-3 days to allow for the return of assay results. This is a very real problem as the Merensky pyroxenite in this area is 3m to 5m thick, with only 1m to 1.4m of economic PGM mineralization within it. Even in unweathered underground workings visual identification of the merensky "Reef" is often impossible. The pit would be too small to facilitate this, as the mining rate would effectively be reduced to 10 000t-15 000t/month. Furthermore, due to lower specific gravity of the Merensky Reef compared to the UG2, the mining cost is higher. The lower specific gravity means that fewer tons of ore are extracted per unit area, and as the hanging wall rocks are very similar in specific gravity to those of the UG2, the stripping ratio and hence the cost increases.

To some extent these problems can be quantified by bulk sampling, which was performed at Western Platinum Ltd.. However, this is a very expensive exercise, as the pit must first be designed, and the box cut placed. The reason for this is that if the pit goes ahead, the box cut can then be used for the production mining of the pit. At Western Platinum, this was done and a 5 700t bulk sample of the Merensky Reef removed, sampled and processed at a concentrator plant. This yielded a recovery of 15-20% in a plant that historically yields 97% recovery. When processing this ore, the plant was adversely affected due to the occurrence of sliming. The cost of mining for this bulk sample was R314 000.

However, the sulfide content, and hence the Ni-Cu values (Table 5.3), is very low, being of the same order of magnitude as in the normal, underground, UG2 ore. The combination of the low sulfide content and the presence of significant amounts of clay minerals means that the flotation process for this ore has very low recoveries. So, despite the exceedingly attractive matte-valuation of more than R47 000/kg, the Merensky on WPL in just not viable. The matte valuation is based on assay values, not on metallurgically recoverable PGM's. If the metallurgical problems can be resolved, the Merensky Reef on WPL may become a viable operation. Extensive test work at Mintek and within Lonrho has led to the development of a process whereby a recovery of ± 50% could be expected. Considering the low in situ grade (3.76 g/t) and the problems outlined above, the Merensky is not really a viable proposition on the Lonrho properties at present.

It has been suggested that this Merensky ore could be treated with UG2 ore (because of its low sulfide tenor), but as LPD only really mines the Merensky Reef for its sulfides to act as collectors for PGM's of the UG2 in the smelting process, this did not make the project any more attractive.

5.7 Results of Mining 2# East UG2 Open Cast Pit:

Work began on the 2# East pit in June 1996, when the contractors came on site. The pit was laid out and stripping of the topsoil commenced, followed by waste rock removal from the box-cut. Towards the end of July, the first ore was produced from the pit. By the end of August the pit had reached full production of 40 000t/month. From August through to December 1996, production has ranged between 40 000t and 68 000t/month. The reason for this is a requirement on the part of the mine that there be a stockpile of at least 40 000t for the Christmas period mining closedown, during which time the concentrator will continue to operate. Waste mining at the pit has ranged between 210 000t and 230 000t/month.

By August 1996, underground production had picked up to record levels. There appears to be no rational explanation for this, although there are several sciolists who gladly accept credit for the up-swing. This appeared to pose somewhat of a problem as, strictly speaking, the mine no longer needed the tonnage from the open cast operation. However, a significant level of commitment existed to see the pit through to completion. A decision was taken in September 1996 to build a substantial stockpile of ore to see the concentrator through the planned Christmas closedown. Furthermore, other mines in the Lonrho Group had excess concentrator capacity and began to take delivery of ore from EPL, for both immediate milling purposes and to build stockpiles for the Christmas period.

For the first time in several years the EPL concentrator has successfully milled through the Christmas mining shutdown without running out of ore to mill, and indeed have broken milling records over that period. This is something that has never before occurred and it is due, to a large extent, to the availability of ore from the 2# East open cast pit.

The open cast ore has been batch-milled at EPL in campaigns ranging in length from two to ten days since July 1996. The results for the period July to November 1996 are presented in Table 5.4 below. The December 1996 figures will only be available in mid-January 1997.

At this point in time, with only some 20% of the pit tonnage having been milled and processed, 28% of the expected kilograms have been recovered. If the pit continues to perform as it has for the first five months, the gross profit will be R38.151 million, some R19.641 million more than projected.

TABLE 5.4

2# East Open Cast Pit Results: July-November 1996

	Tons Milled	Mill Head Grade	Actual Recovery	kg PGM Produced	Cost: Rand/kg	Profit: Rand/kg
July	13,180	4.63	69.3%	60.96	14,071	18,884
August	15,134	4.57	61.2%	69.1	15,049	18,117
September	18,229	4.4	58.7%	80.28	15,770	15,685
October	50,731	4.11	60.4%	208.54	17,261	15,402
November	41,165	4.86	68.7%	121.46	24,304	6,093
TOTALS:	140,439			540.339kg		
Weighted Averages:		4.40 g/t	62.7%		R17,980	R14,092

The October and November data quoted in Table 5.4 are somewhat anomalous and require some explanation. The kilograms recovered reported are impossibly high, taking the tonnage and grade into account. The reason for this anomalously high figure of 208 kg, is a function of metals' accounting and allocation practices. During the period July to September 1996, the LPD smelter and base metal refinery complex under-recovered PGM's, which then was recovered from lock-up in October 1996. During October, the smelters reported a recovery of 125%. This represents a ± 99% recovery for October production plus excess metal that had been locked up in the system for several months prior to that. The locked-up metal is apportioned to each operation on a *pro rata* basis. Had this lock-up not occurred, the kilograms produced for each of the first three months would have been slightly higher, and the kilograms produced for October would have been slightly lower. In November, the smelters reported a net recovery of only 75%, and hence there is once again an under-recovery of PGM's, which will be credited to the pit at some later date.

From an operational point of view, the open cast pit has been performing very well, and in terms of this a number of points merit some discussion:

➢ Geological Losses:

To date, virtually no geological losses have been incurred. Several small potholes have been exposed and, after investigation, were discovered to contain UG2 Chromitite at their base. This chromitite was often substantially thicker than normal, sometimes exceeding 2.0m in thickness. This material has successfully been extracted in all cases thus far.

➢ Split Reef:

Split reef has, unfortunately, been a common feature of the pit in the area mined to date. However, through careful management and vigilance, both on the part of the mine and the contractors, the internal waste of the UG2 in these areas has been extracted separately from the UG2. In all cases, both the top and bottom portions of the UG2 in split reef areas have been mined and sent to the mill. It will be remembered that the bottom portion of the UG2 in known split reef areas was not included in the Mineral Resource Statement for the open cast operation. The fact that it is successfully being mined will have the effect of increasing the ore tonnage from the pit, without materially affecting the stripping ratio. At present the west face is advancing into an area known to contain a large proportion of split reef (Figure 5.7), in which the internal waste is up to 3.0m thick, and at this stage there is a fair level of confidence in being able to successfully remove both the top and bottom portions of the UG2, with minimal dilution.

➢ Grade:

In-pit sampling, based on 127 sections, indicates an *in situ* grade of 3.87 g/t, (4.84 g/t corrected) and truck sampling indicates a head grade of 4.38 g/t. Every face is sampled prior to blasting. These figures correlate very well with the predicted head grade (Figure 5.8).

➢ Dilution:

Dilution of the run-of-mine ore from the pit is measured to be <5% of ore delivered. In the pit, the percentage may be a little higher but waste "picking" is performed on the ore after blasting and on the stockpiles at the pit that has the effect of reducing waste presentations.

➢ Blasting:

To date only two waste blasts have been necessary. However, and despite the weathered nature of the UG2 close to surface, most of the UG2 (60-70%) so far extracted, has required light blasting before removal could be effected. No complaints have been received from local communities or residents in respect of blast damage or noise.

➢ Faulting:

Several small faults have been negotiated in the pit since it began. Most faults encountered so far have been normal faults with a N-S strike, and vertical displacements of up to 1.0m. One exception to this was a very low-angle reverse fault (<10°), also trending N-S, which caused a considerable gain of ground, throughout its entire exposure in the pit (±100m). In places along a ±20m wide N-S trending zone, a stoping width of >2.5m existed and was successfully mined.

➢ Landfill:

Landfill is ongoing. The box-cut is, at the time of writing, back in place with the Moddersspruit river-bed re-instated. Recent rain has caused this stream to flow, and the flow successfully traversed the rehabilitated/re-instated section across the pit, with no problems reported and without flooding the working pits.

➢ Concentrator Recoveries:

The average recovery (weighted) reported by the plant, is 62.7%, some 7.5% better than predicted. There appears to be a definite correlation between percentage recovery and the duration of the milling campaigns: the longer the campaign, the higher the recovery. Although it was planned that the shallow (0-15m) ore and the deep (15-30m) should be milled in separate campaigns, due to the fact that they should be treated differently, it has been found that this is not necessarily true. Therefore, the deep and shallow ores are often blended, with little detrimental effect on recoveries discernible. Sliming in the concentrator has been a problem when shallow open cast ore has been campaigned for longer than 3-4 days on its own.

5.7.1 Production Costs and Profitability:

Although, and as has been noted earlier, the small open cast operations do not carry a portion of the total mine overhead, due to their relatively short-term nature, it is worthwhile examining their production costs and profitability compared to that of underground operations. Table 5.5

below is a simple comparison of these data for a five month period on EPL. However, these data are at the core of the business: minimizing costs and maximizing profits.

TABLE 5.5

Comparison of cost and profit data for underground and open cast production

	Underground		Open Cast	
	Cost/kg	Profit/kg	Cost/kg	Profit/kg
July '96	R20,703	R12,828	R14,071	R18,884
August '96	R23,744	R 9,957	R15,049	R18,117
September '96	R20,714	R11,258	R15,770	R15,685
October '96	R31,933	R 332	R17,261	R15,402
November '96	R27,871	R 3,014	R24,304	R 6,093
Averages:	**R24,993**	**R 7,478**	**R17,979**	**R14,092**

Given the current recoveries, of 62.7%, and despite being ± 7% over budget mainly due to the cost of stockpiles, the open cast operation is far more profitable, ton-for-ton, than the underground operations. Note also that the high cost/kg for November is due to the smelter under-recovery discussed above. This figure will be 'normalized' once the metal currently locked up, is released.

The 2# East Pit is performing far better than predicted in terms of cost/kg and profit/kg. The predicted cost/kg was R23 619/kg, which is R5 640/kg more than the average current cost. The predicted profit/kg was R9 589/kg, which is R4 503/kg less than the current average figure of R14 092/kg. To a large extent the performance is due to two factors: firstly the better than predicted recovery at the concentrator, and secondly, the high degree of conservatism applied in the pre-feasibility and feasibility work.

Chapter 6

Chapter 6

6.0 Conclusions

6.1 Future Potential for Open Cast Mining in the Bushveld Complex

Small scale open cast mining of the UG2, and the Merensky Reef has proved itself to be feasible and profitable. It enjoys a number of advantages which will be, and are being, exploited. It provides a readily available, and often as yet unexploited, resource that was previously thought to be unexploitable. It can be brought into production rapidly, as shown in Chapter 5. It requires little or no capital, if contract mining is employed. The problems of (comparatively) low grade and low metallurgical recoveries are more than offset by very low mining costs. The metallurgical treatment costs are similar to those incurred by underground ore. The technology, knowledge and experience now exists to facilitate the profitable mining and treatment of shallow Merensky and UG2 ores. Viewed against a background of low metal prices and low underground productivity, the prospect of mining these resources must continue to be very attractive.

There has been a sudden and great interest in exploiting these shallow reserves in the western Bushveld Complex during 1996. This interest will probably remain focused on the western Bushveld because of a number of factors. Firstly, because that is where most of the concentrators are, and the scale of mining being considered here is opportunistic. The concentrator and mining capacity in the western Bushveld far exceeds that of the eastern Bushveld, and therefore more opportunity exists in the western Bushveld for small scale open cast pits to redress production shortfalls. Secondly, the combination of physiographic and structural factors tend to favour the western Bushveld. In general terms, the topography tends to be flatter and more uniform in the western Bushveld than in the eastern Bushveld. The dip of both the UG2 and the Merensky tends to be flatter in the western Bushveld than in the eastern Bushveld, while the ore thicknesses remain much the same. These factors would severely limit the profitability of small pit in the eastern Bushveld. However, some scope exists in the eastern Bushveld for this, especially at the start-up stage of underground mining.

Lonrho is currently mining two small UG2 open cast pits, with two more planned over the next two years (Figure 2.1). There is also the possibility that a third UG2 pit will be mined during that period. The possibility of mining shallow Merensky resources is still a possibility, though remote, but work is ongoing on this. In the medium term, Lonrho has a 4 km stretch of virgin ground, at Middlekraal (Figure 2.1), with both UG2 and Merensky open cast potential. This, it is felt will only be exploited once the planned shaft system there is begun, and will reduce the capital requirements considerably, by providing ore from the beginning of the project, even before the shafts begin to produce ore from underground.

Amplats too, seems convinced of the benefits of exploiting shallow Merensky and UG2 resources, previously ignored. The Bleskop pit at Rustenburg Section (Figure 6.1), the first of their recent forays into small scale open cast mining in the western Bushveld, is nearing completion. Amplats have recently announced a further R200 million investment in PPRust (Potgietersrus Platinum, Figure 6.1), which will facilitate a doubling of production there. While strictly not comparable to the type and scale of operation being examined here, it is, and will become an increasingly important factor in the PGM supply industry, simply because it can react to the market demand much more rapidly than almost any of the other South African (underground) operations. Its success has almost certainly influenced decisions to go ahead with small scale UG2 and Merensky open cast operations in the western Bushveld.

During the latter part of 1996, Amplats began open cast operations at Amandelbult (Figure 6.1) on the Merensky Reef. Metallurgical recoveries are reported to be in the region of 75%-80% from ore from this pit. Amandelbult has some 20 km of strike (Figure 6.1) in this area, so this pit could be quite a long-lived operation. Swartklip, which has some 13 km of strike, has also started a Merensky and a UG2 open cast pit, with metallurgical recoveries reported to be 70% from the Merensky. These too could be fairly long-lived operations. All of these pits will be mined down to a vertical depth of 25-30m. A UG2 pit has also commenced at Boschfontein, south of the Impala properties. However, it appears that this pit is suffering, not from problems related to weathering, but rather from problems of dilution of ore with waste. A further complication in this area is the existence of a railway line, limiting the vertical depth of the pit to only 18m. Amplats are currently investigating the Merensky in this area, and it is expected that it will

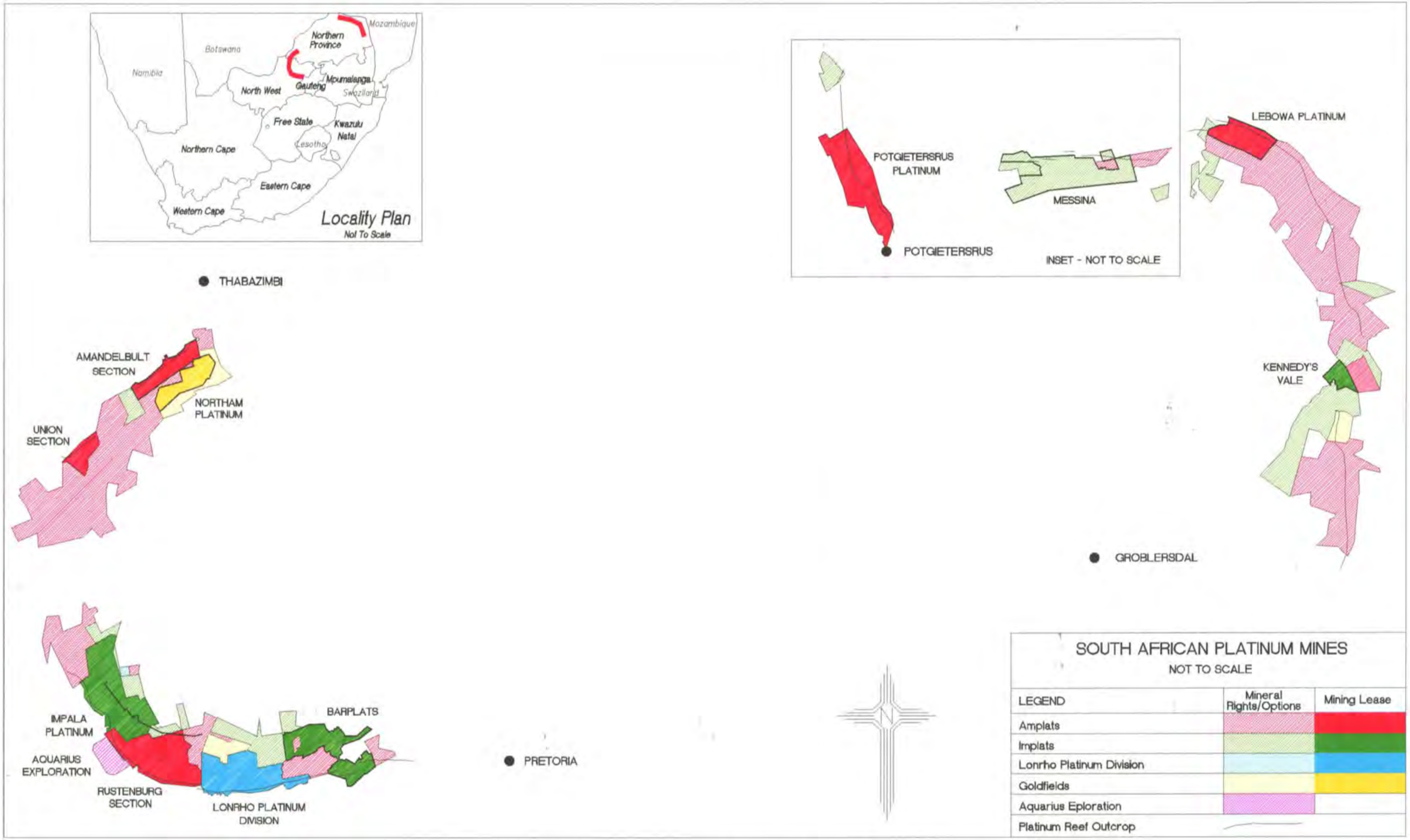


FIGURE 6 . 1 SOUTH AFRICAN PLATINUM MINES

come into production early in 1997. Union Section (Figure 6.1) is reported to be mining a UG2 open pit and investigating the possibility of a Merensky Pit (Diedricks, *pers. comm.*, 1996). Amplats have a block of ground, known as Boschkoppies, between Sun City and the Impala Properties. They also have quite a large block of ground, between EPL and CRM, known as the Pandora block, that has 4-5km of UG2 outcrop that could be mined by open cast methods. In addition to this, Amplats has another 3-4km of Merensky Reef outcrop at Brakspruit Mine, west of Marikana. It is not known if Amplats are examining the possibility of open cast mining in these areas, but the potential certainly exists.

The scale of these pits is difficult to estimate as, although Amplats holds all of the mineral rights in these areas, it is not known if they own the surface rights. Where they do not, it may or may not be possible to mine these resources, depending upon whether they would wish to negotiate with the land owners or not.

Impala is known to be examining the possibility of mining some shallow Merensky resources by open cast methods. As mentioned previously, Impala unfortunately positioned their concentrator and smelting complex on much of their UG2 outcrop. Impala has very little Merensky or UG2 strike length of either, exploitable by open cast methods (Figure 6.1). They possibly have 2-3 km of Merensky that they can open cast on the Impala property (Slabbert, *pers. comm.,* 1996). However, Impala does have exploitable open cast reserves at the Crocodile River Mine, and possibly in the eastern Bushveld, in the Northern Province. Unfortunately, CRM has limited surface rights and Impala would have to either negotiate a mutually acceptable deal with the surface rights holders or purchase the ground outright, either of which would possibly greatly reduce the profitability of open cast mining in this area.

Aquarius Exploration, an Australian company recently exercised its mineral rights options on a block of ground with virgin UG2 resources, near Kroondal, south of Rustenburg (Figure 6.1). With just over 6 km of strike, they are unlikely to become a major player in the South African PGM industry. Their novel approach to establishing an operating mine in the Bushveld on the UG2 is however, worthy of comment.

Aquarius plan to mine much of their shallow ore initially by open cast mining, thereafter taking a decline down from the central box-cut to facilitate underground mining. The open cast mining depth is planned at between 25 and 30m, at a mining cost of R40-R45/t of ore. The area has average PGM grades, and bulk testing has indicated that recoveries of between 55% and 60% can be expected from the open cast ore. One of the principal problems is that the UG2 leaders or markers closely approach the UG2 over much of the area. However, they are reported to carry higher than normal grade, and Aquarius is currently examining the possibility of mining the marker with the UG2, including the intervening hanging wall waste. To reduce the effect of this dilution, dense media separation, similar to that used on many of the small chromite mines in the region, is being tested. If this is successful, the stoping width will be sufficient to facilitate mechanized mining underground. The planned pit tonnage is 50 000t/month and thereafter 80 000t/month from underground (Dix, *pers. comm.*, 1996).

Like Russia, and the other major PGM producers of South Africa, BHP has become extremely secretive about its operations in the eastern Bushveld, specifically in Lebowa where they have had an active exploration programme for one or two years now. Depending on the local and regional dip of the Bushveld complex there, there is possibly potential for open cast mining once that project moves into a production phase.

6.2 Summary:

There is extensive interest at present, among all the major producers of PGM's in the Bushveld Complex, in small scale open cast mining of both the UG2 and of the Merensky. Potentially new operations, such as the Aquarius Mine, are examining the possibility of open cast mining of shallow reserves to improve their cash-flow position during shaft sinking and construction phases, which will, it is believed, considerably reduce their borrowings or long term capital requirements. It also helps to maximize the exploitation of a limited resource, and will provide a better return, sooner, to shareholders.

Established PGM producers are exploiting a previously ignored, and often much maligned, resource, usually to resolve short-term production problems, or spare capacity due to expansion. Some of these shallow resources are quite substantial (Amandelbult - 20 km strike; Swartklip - 13 km strike) and may be justifiable mining operations in their own right.

A number if technical issues can now be redefined and clarified. Firstly, deep and shallow ore need not be treated separately in the concentrator. In fact recent experience has shown that blending the two actually limits sliming in the concentrator, and does not seem to affect the recoveries which are on average 62.7%. Secondly, the dilution factor used (10%) was too high. A factor of 5%-7% is felt, in retrospect, to be sufficient. Thirdly, geological losses need careful consideration, based on experience and local knowledge. It appears as though the allowance for geological losses was a little too conservative, but judgment on this will be reserved until the current pit at EPL is complete. Geological losses will obviously vary greatly from one pit to another so this is a difficult factor to estimate, or for which to provide meaningful rules of thumb.

Due to current depressed PGM prices, and rising working costs and productivity problems being experienced by the industry in South Africa, there may be an increase in open cast mining simply because it is more profitable. However, this requires that other, underground, production not be curtailed. The reason for this is that this would be too expensive (mothballing shafts, retrenchments, etc.). Open cast mining of shallow UG2 or Merensky resources could be a viable option to defer expensive underground production expansion. It could also be utilized to replace worked out or dwindling underground resources in the short to medium term, depending on how much tonnage is available from such resources, rather than to replace those resources by underground development. Open cast operations can be operated on a fixed cost per ton basis, which is often a difficult goal to achieve in underground operations.

The exploitation of shallow open cast resources should be utilized to maximize profits during period of depressed metal prices, and be viewed as something more than short-term, opportunistic, stopgap operations. With the existing infrastructure that is in place, these pits can be extremely profitable.

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Appendices

APPENDIX 1

FORMULAE FOR FINANCIAL TEMPLATE

CELL ADDRESS				
AU 76	RESERVES (tons)	0-15 m	=$H$57	$H$57: ORE TONNAGE 0-15m
AU 77	GEOLOGICAL LOSS	10.0%	=(AU76*0.1)	
AU 78	MINEABLE TONS	(ORE)	=(AU76-AU77)	
AU 79	MINING DILUTION	10%	=AU78*0.1	
AU 80	% RECOVERY	pgeR	INPUT	Recovery % for shallow ore
AU 81	PGE RECOVERED	PGEkg	=((AU78*AU93*AU80)+(AU79*0.3*AU80))/1000	
AU 82				
AU 83	RESERVES (tons)	15-30 m	=$H$58	$H$57: ORE TONNAGE 15-30m
AU 84	GEOLOGICAL LOSS	10.0%	=(AU83*0.1)	
AU 85	MINEABLE TONS	(ORE)	=(AU83-AU84)	
AU 86	MINING DILUTION	10%	=AU85*0.1	
AU 87	% RECOVERY	pgeR	INPUT	Recovery % for deep ore
AU 88	PGE RECOVERED	PGEkg	=((AU85*AU93*AU87)+(AU86*0.3*AU87))/1000	
AU 89	REEF TONS	TOTAL	=AU76+AU83	
AU 90	MILLED TONS	T	=(AU78+AU79+AU85+AU86)	
AU 91	TONS MINED	M	=(AU89+$H$60)	
AU 92	PGE kg in conc.		=(AU81+AU88)	
AU 93	PGE GRADE	pgeG	=$H$61	Corrected grade from Figure
AU 94	PGE kg in matte	PGEkg	=AU92*0.99	
AU 95	MATTE VALUATION (R/k	pgeP	=$B$13	Rand/kg as at April 1996
AU 96	ON-MINE COSTS			
AU 97	MINING/t		=$B$21	Mining Cost / t of Ore
AU 98	MILLING/t		INPUT	Milling Costs / t of Ore
AU 99	SMELTING COST/t		=$B$16	Smelting Cost / t of Ore
AU 100	TRANSPORT @42c/t/km		INPUT	Transport Costs @ 42c/km
AU 101	TOTAL COSTS/t OF ORE		=(AU103/AU90)+(AU97+AU98+AU99+AU100)	
AU 102	REVENUE/t		=((AU94*AU95))/AU90	
AU 103	SUNDRY COSTS		INPUT	Total Sundry Costs
AU 104	GROSS REVENUE		=AU102*AU90	
AU 105	GROSS COSTS		=(AU97*AU90)+(AU98*AU90)+(AU99*AU90)+(AU100*AU90)+AU103	
AU 106	GROSS PROFIT/t		=(AU104-AU105)/AU90	
AU 107	GROSS PROFIT (TOTAL)		=AU106*AU90	
AU 108				
AU 109	COST/kg		=AU105/AU94	
AU 110				
AU 111	PROFIT/kg		=AU107/AU94	
AU 112				
AU 113	MARGIN %		=AU107/AU104	

References
&
Bibliography

References & Bibliography

Anon, 1993: Platinum Metals. *Metals and Minerals Annual Review - 1993*, 32-36.

Anon, 1992: Platinum Metals. *Metals and Minerals Annual Review - 1992*, 29-31.

Anon, 1992: *Aide-Mémoire for the Preperation of Environmental Management Programme Reports for Prospecting and Mining.* Department of Mineral and Energy affairs, Government Printers of South Africa, Pretoria. 34pp.

Anon, 1991: *Minerals Act, 1991: Act No. 50 of 1991.* Government Printers of South Africa, Pretoria.

Anon, 1987: *The successful development of an industrial process for the recovery of platinum-group metals from the UG2 Reef.* Application Report No. 1, MINTEK, Johannesburg, 10 pp.

Barry, S.D., 1992: Platinum - Substitution in Autocatalyst Applications in *Minerals Bureau Bulletin Nov/Dec 1992.* Department of Mineral and Energy Affairs, Minerals Bureau, Johannesburg.

Barry, S.D., 1992: Platinum Group Metals, Nickel, Copper and Cobalt-Reserves within the Bushveld Complex in *Minerals Bureau Bulletin Nov/Dec 1992.* Department of Mineral and Energy Affairs, Minerals Bureau, Johannesburg.

Barry, S.D., 1993: Platinum Group Metals in *South Africa's Mineral Industry 1992/93.* Department of Mineral and Energy Affairs, Minerals Bureau, Johannesburg.

Barry, S.D., 1994: Platinum Group Metals in *South Africa's Mineral Industry 1993/94.* Department of Mineral and Energy Affairs, Minerals Bureau, Johannesburg.

Carr, H.W., Groves, D.I. & Cawthorn, R.G., 1994: Magma loading and synmagmatic deformation: a new concept to explain regional and mine-scale discontinuities, including Merensky Reef potholes in the Bushveld Complex. *Econ.Geol.*, **89**, 1389-1410.

Cameron, E.N. & Desborough, G.A., 1964: Origin of Certain Magnetite-Bearing Pegmatites in the Eastern Part of the Bushveld Complex, South Africa. *Econ.Geol.* **59**,197-225

Cameron, E.N., 1978: The Lower Zone of the Eastern Bushveld Complex in the Olifants River Trough. *J.Petrol.* **19(3)**, 437-462

Christian, J.M., 1996: Platinum Group Metals: Hard Lesson to be Learned. *E & MJ*, **197,** no. 3, 21-27.

Clendinin, C.W., Charlesworth, E.G. & Maske, S., 1988: An early Proterozoic three-stage rift system, Kaapvaal Craton, South Africa. *Tectonophysics*, **145**,73-86

Davey, S.R., 1992: Lateral variations within the upper Critical Zone of the Bushveld Complex on the farm Rooikoppies 297 JQ, Marikana, *South Africa. South African J. Geol.*, **95,** 141-149.

Deetlefs, J.P., (ed.) 1986: *The Planning and Operation of Open-Pit and Strip Mines.* SAIMM. Johannesburg. 497pp.

de Klerk, W.J., 1991: Petrogenesis of the Upper Critical Zone in the Western Bushveld Complex with emphasis on the UG1 Footwall and the Bastard Units. *PhD Thesis*, Rhodes University (unpublished) 294pp.

Eales, H.V., de Klerk, W.J., Teigler, B. & Maier, W.D., 1994: Nature and origin of orthopyroxenites in the western Bushveld Complex, in the light of compositional data. *S.Afr.J.Geol.*, **97(4)**, 399-407.

Eales, H.V., Botha, W.J., Hattingh, P.J., de Klerk, W.J., Maier, W.D. & Odgers, A.T.R., 1993: The Mafic rocks of the Bushveld Complex; A review of emplacement and crystallization history, and mineralization, in the light of recent data. *Jour.Afr.Earth Sci.*, **16(1-2)**, 121-142.

Eales, H.V., de Klerk, W.J. & Teigler, B., 1990: Evidence of magma mixing processes within the Critical and Lower Critical Zones of the Northwestern Bushveld Complex, South Africa. *Chemical Geology*, **88**, 261-278.

Farquhar, J., 1986: The Western Platinum Mine. in Anhausser, C.R. & Maske, S. (eds.), *Mineral Deposits of Southern Africa*: Johannesburg, Geological Society of South Africa, v.2, 1135-1142.

Forrest, S. 1996: *Platinum Group Metals Review.* SMK Securities Investment Report, SMK, Johannesburg, 99pp.

Fuggle, R.F. & Rabie, M.A., (eds.) 1992: *Environmental Management in South Africa.* Juta & Co., Ltd. Cape Town, South Africa. 823pp.

Gain, S.B., 1980: The geology and PGE distribution in the upper group chromitite layers at Maadagshoek 254 KT, eastern Bushveld Complex: *Univ. Pretoria, Inst. Geol. Research on the Bushveld Complex. Research Report No. 22*, 24pp.

Gain, S.B., 1985: The Geologic Setting of the Platiniferous UG2 Chromitite Layer on the Farm Maadagshoek, Eastern Bushveld Complex. *Econ.Geol.*, **80**, 925-943.

Gilmour, B., 1994: Platinum Metals. *Metals and Minerals Annual Review - 1994*, 26-28.

Gilmour, B., 1995: Platinum Metals. *Metals and Minerals Annual Review - 1995*, 23-26.

Grimbeek, J.C., 1996: Mineralogical Investigation of Oxidized Merensky Ore from Karee Mine. *Mintek External Investigation Report No.4089601 (50) C*, Mintek, Johannesburg. 11pp.

Hall, A.L. & Humphrey, W.A., 1908: On the Chromite Deposits along the Southern and Eastern Margins of the Bushveld Plutonic Complex, *Trans. Geol. Soc. South Africa*, **11**, 69-77.

Hatton, C.J. & Sharpe, M.R., 1983: Significance and origin of boninite-like rocks associated with the Bushveld Complex. in Crawford, A.J. (ed.) *Boninites and Related Rocks*, Unwin Hyman, London (1983). 465pp

Houseman, G.A., 1990: Thermal structure of mantle plumes: axisymmetric or triple junction? *Geophys.Jour.Int.*, **102**,15-24.

Johnson Matthey, 1996: *Platinum 1996; Annual Review.* Johnson Matthey, London. 52pp.

Johnson Matthey, 1996: *Platinum 1996; Interim Review.* Johnson Matthey, London. 28pp.

Johnson Matthey, 1995: *Platinum 1995; Annual Review.* Johnson Matthey, London. 52pp.

Johnson Matthey, 1995: *Platinum 1995; Interim Review.* Johnson Matthey, London. 28pp.

Johnson Matthey, 1994: *Platinum 1994; Annual Review.* Johnson Matthey, London. 52pp.

Johnson Matthey, 1994: *Platinum 1994; Interim Review.* Johnson Matthey, London. 26pp.

Johnson Matthey, 1993: *Platinum 1993; Annual Review.* Johnson Matthey, London. 52pp.

Johnson Matthey, 1993: *Platinum 1993; Interim Review.* Johnson Matthey, London. 28pp.

Johnson Matthey, 1992: *Platinum 1992; Annual Review.* Johnson Matthey, London. 56pp.

Johnson Matthey, 1992: *Platinum 1992; Interim Review*. Johnson Matthey, London. 24pp.

Kartun, K., 1996: Delta Gold - A World-Class Platinum Exposure. *BZW Company Review*, Sydney, Australia. 16pp.

Kruger, F.J., 1990: The stratigraphy of the Bushveld Complex; a re-appraisal and the relocation of the Main Zone boundaries. *S.Afr.J.Geol.*, **93(2)**, 376-381

Lee, C.A. & Butcher, A.R., 1990: Cyclicity in the Sr-isotope stratigraphy through the Merensky and Bastard Reef Units, Atok Section, Eastern Bushveld Complex. *Econ.Geol.*, **85**, 877-883.

Leeb-du Toit, A., 1986: The Impala Platinum Mines. in Anhausser, C.R. & Maske, S. (eds.), *Mineral Deposits of Southern Africa*: Johannesburg, Geological Society of South Africa, 1091-1106.

Maier, W.D. & Walters, B.M., 1994: The UG2 - Merensky Reef interval at Amandelbult Section, Rustenburg Platinum Mines: Pattern of lateral variation in lithology. *S.Afr.J.Geol.*, **97(1)**, 45-51.

Maier, W.D. & Eales, H.V.., 1994: Plagioclase inclusions in orthopyroxene and olivine of the UG2 - Merensky Reef interval: regional trends in the western Bushveld Complex. *S.Afr.J.Geol.*, **97(4)**, 408-414.

Mallinson, C, 1995: *An Introduction to the Financial Evaluation of Mineral Deposits.Course Notes.* Rhodes University MSc in Exploration Geology.

Mallinson, C, 1995: *An Introduction to the Financial Evaluation of Mineral Deposits. Notes, Practicals and Programmes.* Rhodes University MSc in Exploration Geology.

Marais, P. & Nisbett, A.R., 1989: Flotation of Oxidized UG2 outcrop for Western Platinum Ltd. *Mintek Confidential Report No. C 1350M*, Mintek, Johannesburg. 40pp.

McLaren, C.H. & De Villiers, J.P., 1982: The Platinum-Group Chemistry and Mineralogy of the UG2 Chromitite Layer of the Bushveld Complex. *Econ.Geol.*, **77**, 1348-1366.

Mooney, D.G., 1996: 2# East Open Pit: Background Data and Feasibility Study. *LPD Internal Report, EPLGEO-102.* 112pp.

Mooney, D.G., 1995: Pegmatoids, Pipes and Potholes: A Discussion and Review. *LPD Bulletin No. 48, 31pp.*

Naldrett, A.J., 1989: Stratiform PGE Deposits in Layered Intrusions: *Reviews in Economic Geology*, **4**, 135-166.

Naudé, M.L., 1996: Karee Open Cast Test Run: Preliminary Report. *LPD Internal Report PROC-01.* 27pp.

Odendal, N.J., 1992: Platinum Group Metals in *South Africa's Mineral Industry 1991/92.* Department of Mineral and Energy Affairs, Minerals Bureau, Johannesburg.

Patterson, A., 1982: Big idea gets a small mine running sweetly. *Coal, Gold & Base Minerals of Southern Africa*, **30, no. 3,** 41-51.

Pleming, D., 1996: Lonrho Platinum Division. *Barnard, Jacobs, Mellet & Co (Pty.) Limited: Stock Market Report,* Johannesburg. 11pp.

Sharpe, M.R. & Hulbert, L.J., 1985: Ultramafic Sills beneath the Eastern Bushveld Complex: Mobilized Suspensions of Early Lower Zone Cumulates in Parental Magma with Boninitic Affinities. *Econ. Geol.*, **80**, 849-871

Sharpe, M.R. & Snyman, J.A., 1980: A model for the emplacement of the eastern compartment of the Bushveld Complex. *Tectonophysics*, **65**, 85-110.

Streczynski, A. 1990: Laboratory Rate Flotation of Platinum Group Metals and Gold from Eastern Platinum UG2 Reef and Pegmatoid Footwall of the Open Pit. *Mintek Confidential Report No. C 1482M,* Mintek, Johannesburg. 32pp.

Teigler, B., Eales, H.V. & Scoon, R., 1992: The Cumulate Succession in the Critical Zone of the Rustenburg Layered Suite at Brits, Western Bushveld Complex. *S.Afr.J.Geol.,* **95(1-2)**, 17-28.

van der Westhuizen, C., 1996: Alteration of the Modderspruit: Proposed New UG2 No. 2 Incline Shaft Opencast Mining Project. Eastern Platinum Ltd.. *Consulting Hydrogeologist Report.* 33pp.

van der Westhuizen, C., 1996: Eastern Platinum Ltd.: Drilling and Testing Monitoring Boreholes. *Consulting Hydrogeologist Report.* 12pp.

van Rensburg, P.W.J., (ed.) 1970: *Planning Open Pit Mines.* SAIMM. Johannesburg. 388pp.

Vermaak, C.F., 1976: The Merensky Reef - thoughts on its environment and genesis. *Econ. Geol.*, **71**, 1270-1298.

Vermaak, C.F., 1995: *The Platinum Group Metals; A Global perspective.* Mintek, Johannesburg. 247pp.

Vermaak, C.F. & Von Gruenewaldt, G., 1986: Introduction to the Bushveld Complex. in Anhaeusser, C.R. & Maske, S. (eds.) Mineral Deposits of Southern Africa. Geol Soc. S.Africa, Johannesburg. 1021-1030.

Viljoen, M.J. & Hieber, R., 1986: The Rustenburg section of the Rustenburg Platinum Mines Ltd., with reference to the Merensky Reef. in Anhausser, C.R. & Maske, S. (eds.), *Mineral Deposits of Southern Africa*: Johannesburg, Geological Society of South Africa, 1107-1134.

Viljoen, M.J., de Klerk, W.J., Coetzer, P.M., Hatch, N.P., Kinloch, E. & Peyerl, W., 1986: The Union Section of Rustenburg Platinum Mines Limited, with reference to the Merensky Reef. in Anhausser, C.R. & Maske, S. (eds.), *Mineral Deposits of Southern Africa*: Johannesburg, Geological Society of South Africa, 1061-1090.

Viljoen, M.J., Theron, J., Underwood, B., Walters, B.M., Weaver, J. & Peyerl, W.,1986: The Amandelbult Section of the Rustenburg Platinum Mines Limited, with reference to the Merensky Reef. in Anhausser, C.R. & Maske, S. (eds.), *Mineral Deposits of Southern Africa*: Johannesburg, Geological Society of South Africa, 1041-1060.

Von Gruenewaldt, G., Sharpe, M.R. & Hatton, C.J., 1985: The Bushveld Complex; Introduction and Review. *Econ.Geol.*, **80**, 803-812.

Von Gruenewaldt, G., 1979: A review of some recent concepts of the Bushveld Complex, with particular reference to sulfide mineralization. *Canadian Mineralogist*, **17**, 233-256.

Wagner, P.A., 1929: *Platinum Deposits and Mines of South Africa.* Oliver & Boyd, London, 326pp

Wollacott, L.C. & Eric, R.H. (eds.) 1994: *Mineral and Metal Extraction: An Overview.* SAIMM Monograph Series M8, Johannesburg. 412pp.