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(Report of Foreign Issuer)

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FORM 6-K
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

Report of Foreign Private Issuer
Pursuant to Rule 13a-16 or 15d-16
of the Securities Exchange Act of 1934

For: **January 22 to February 2, 2007**

Platinum Group Metals Ltd.

(SEC File No. 0-30306)

Suite 328 – 550 Burrard Street, Vancouver BC, V6C 2B5, CANADA
Address of Principal Executive Office

The registrant files annual reports under cover: Form 20-F [X] Form 40-F [ ]

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Indicate by check mark whether by furnishing the information contained in this Form, the registrant is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934:
Yes [ ] No [X]

If "Yes" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b): 82-_____

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

EXHIBIT LIST

99.1 Technical Report

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

Date: **February 9, 2007**

"Frank Hallam"
FRANK R. HALLAM
DIRECTOR & CFO

PLATINUM GROUP METALS (RSA) (Pty) LTD
REPUBLIC OF SOUTH AFRICA REGISTERED COMPANY
REGISTRATION NUMBER: 2000/025984/07

A WHOLLY-OWNED SUBSIDIARY OF

PLATINUM GROUP METALS LTD
TORONTO LISTED COMPANY
TSX – PTM; OTCBB: PTMQF

TECHNICAL REPORT

Western Bushveld Joint Venture

PROJECT 1

(ELANDSFONTEIN AND FRISCHGEWAAGD)

A REPORT ON THE REVISED RESOURCE ESTIMATION AND PRE-FEASIBILITY STUDY FOR A PORTION OF THE **WESTERN BUSHVELD JOINT VENTURE** FORMING PART OF A NOTARIALLY EXECUTED JOINT VENTURE PROJECT AGREED ON BETWEEN PLATINUM GROUP METALS (RSA) (PTY) LTD, PLATINUM GROUP METALS LTD, RUSTENBURG PLATINUM MINES LTD AND AFRICA WIDE MINERAL PROSPECTING AND EXPLORATION (PTY) LTD

TURNBERRY PROJECTS (PTY) LTD
FERNDALE, RANDBURG, REPUBLIC OF SOUTH AFRICA

GLOBAL GEO SERVICES (PTY) LTD
CJ MULLER (SACNAPS 400201/04) OF
GLOBAL GEO SERVICES (PTY) LTD, RANT-EN-DAL, GAUTENG, REPUBLIC OF SOUTH AFRICA

15 January 2007

IMPORTANT NOTICE

This report includes updated results for resources announced by Platinum Group Metals Ltd on 21 September 2006 (news release filed with SEDAR). The report communicates revised Inferred, Indicated and Measured Resources calculated using the updated results of 120 boreholes. The independent resource calculation confirms the initial declaration of Measured and shows an increase in Indicated 4E – platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au) – Resources for the project. The reader is warned that mineral resources that are not mineral reserves are not regarded as demonstrably viable.

Inferred, Indicated and Measured Resources have been reported. The US Securities and Exchange Commission does not recognise the reporting of Inferred Resources. These resources are reported under Canadian National Instrument 43-101, but there is a great deal of uncertainty as to their existence and economic and legal feasibility and investors are warned against the risk of assuming that all or any part of Inferred Resources will ever be upgraded to a higher category. Under Canadian rules estimates of Inferred Mineral Resources may not form the sole basis of feasibility studies or Pre-feasibility studies. INVESTORS IN THE USA AND ELSEWHERE ARE CAUTIONED AGAINST ASSUMING THAT PART OR ALL OF AN INFERRED RESOURCE EXISTS, OR IS ECONOMICALLY OR LEGALLY MINEABLE.

We further advise US investors and all other investors that while the terms "Measured Resources" and "Indicated Resources" are recognised and required by Canadian regulations, the US Securities and Exchange Commission does not recognise these either. US INVESTORS ARE CAUTIONED NOT TO ASSUME THAT ANY PART OF OR ALL OF MINERAL DEPOSITS IN THESE CATEGORIES WILL EVER BE CONVERTED INTO RESERVES.

The United States Securities and Exchange Commission permits US mining companies, in their filings with the SEC, to disclose only those mineral deposits that a company can economically and legally extract or produce. This report and other corporate releases contain information about adjacent properties on which the Company has no right to explore or mine. We advise US and all investors that SEC mining guidelines strictly prohibit information of this type in documents filed with the SEC. US investors are warned that mineral deposits on adjacent properties are not indicative of mineral deposits on the Company's properties.

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ITEM 3: SUMMARY

The property and terms of reference

The Western Bushveld Joint Venture (WBJV) is owned 37% by Platinum Group Metals RSA (Pty) Ltd, (PTM) – a wholly-owned subsidiary of Platinum Group Metals Ltd (Canada), (PTML) – 37% by Rustenburg Platinum Mines Ltd, (RPM) – a subsidiary of Anglo Platinum Ltd, (AP) – and 26% by Africa Wide Mineral Prospecting and Exploration (Pty) Ltd, (AW). AW is a company founded on Black Economic Empowerment principles as required under the Mineral and Petroleum Resources Development Act, 2002. The joint venture is a notorial contract and managed by a committee representing all partners. PTM is the operator of the joint venture.

This Technical Report complies with the Canadian National Instrument 43-101 (Standards of Disclosure for Mineral Projects) and the resource classifications in the SAMREC code.

The joint venture relates to properties on Elandsfontein 102JQ, Onderstepoort 98JQ, Frischgewaagd 96JQ and Koedoesfontein 94JQ covering some 67 square kilometres.

The Qualified Person (QP) for this Technical Report is Mr GI Cunningham (Turnberry Projects (Pty) Ltd) who relied on input from other suitably qualified experts such as Mr CJ Muller (Global Geo Services (Pty) Ltd) and Mr TV Spindler (Turnberry Projects (Pty) Ltd).

The principle QP and other qualified experts have visited the WBJV Project 1 site and on several occasions throughout the year of 2006 and detailed discussions were held with PTML and PTM technical personnel at the PTM and Turnberry offices in Johannesburg.

Location

The WBJV property is located in the western limb of the Bushveld Igneous Complex (BIC), 110 kilometres west-northwest of Pretoria and 120 kilometres from Johannesburg. The resources of the WBJV Project 1 are located approximately 1km from the active Merensky Reef mining face at the operating Bafokeng Rasimone Platinum Mine (BRPM) along strike. BRPM completed opencast mining on the UG2 Reef within 100m of the WBJV property boundary.

Ownership

The government of South Africa holds the mineral rights to the project properties under the new act, No. 28 of 2002: Mineral and Petroleum Resources Development Act, 2002. The mineral rights are a combination of new order prospecting permits under the Mineral and Petroleum Resources Development Act and old order permits

under previous legislation accompanied by filed applications for conversion. All applications for conversion have been accepted and execution of new order permits are either in place or are approved and in process.

Geology

The WBJV property is partly situated in a layered igneous complex known as the Bushveld Igneous Complex (BIC) and its surrounding sedimentary footwall rocks. The BIC is unique and well known for its layering and continuity of economic horizons mined for platinum, palladium and other platinum-group elements, chrome and vanadium. To the north the property extends into a younger Pilanesberg Igneous Complex that truncates the target BIC rocks.

Mineralisation

The potential economic horizons in the WBJV Project 1 are the Merensky Reef and UG2 Reef situated in the Critical Zone of the Rustenburg Layered Suite (RLS) of the BIC; these horizons are known for their continuity. The Merensky Reef in this project area is the main exploitation target; the UG2 Reef has lesser economic potential and will be exploited after the Merensky Reef during a later stage of the proposed mine life. The Merensky and UG2 Reefs generally form part of a well-known layered sequence, which is mined at the BRPM adjoining the WBJV property as well as on other contiguous platinum-mine properties. In general the layered package dips at about 19 degrees to the northeast and local variations in the reef attitude have been modelled.

Exploration concept

The Merensky Reef has been considered for extraction over a diluted mining width of 1.26m and the UG2 Reef diluted mining width is 1.53m. The grade content – centimetre gram per ton (cmg/t) – was used as a resource cut-off. Indicated resources total 5.676 million ounces for 4E (platinum, palladium, rhodium and gold) for Project 1. In addition the resource calculation includes a Measured Resource of 0.744 million ounces 4E. This brings the total updated Indicated and Measured Resource base to an estimated 6.420 million ounces 4E. The updated Inferred Resource estimate is 1.863 million ounces, which represents future opportunity and if implemented could enhance the mining profile. This resource is located from near surface to a depth in excess of 600m below surface, which is comparable to other active mining operations in South Africa on the same reefs.

Resource estimates are shown in the following tables.

Estimated Measured Resource base:

MR FPP = Merensky Reef pegmatoidal feldspathic pyroxenite; MR CR = Merensky Reef contact reef; and UG2 = Upper Group No. 2 chromitite seam; PGM = Platinum-group metals.
The cut-offs for Indicated and Inferred Resources have been established by a qualified person after a review of potential operating costs and other factors.

Measured Resource							
	Cut-off (cmg/t) 4E	**Million tons**	**Grade (g/t) 4E**	**Mining width (cm)**		**Tons PGM (4E)**	**Million ounces PGMs (4E)**
MR FPP	100	2.187	7.11	1.24		15.554	0.500
UG2	100	2.266	3.35	1.47		7.599	0.244
Total Measured		4.453	5.20			23.153	0.744

Prill Splits	**Pt**	**Pt (g/t)**	**Pd**	**Pd (g/t)**	**Rh**	**Rh (g/t)**	**Au**	**Au (g/t)**
MR FPP	62%	4.42	26%	1.85	5%	0.36	7%	0.48
UG2	64%	2.15	24%	0.80	10%	0.35	1%	0.05

Estimated Indicated Resource base:

MR FPP = Merensky Reef pegmatoidal feldspathic pyroxenite; MR CR = Merensky Reef contact reef; and UG2 = Upper Group No. 2 chromitite seam; PGM = Platinum-group metals.
The cut-offs for Indicated and Inferred Resources have been established by a qualified person after a review of potential operating costs and other factors.

Indicated Resource							
	Cut-off (cmg/t) 4E	**Million tons**	**Grade (g/t) 4E**	**Mining width (cm)**		**Tons PGM (4E)**	**Million ounces PGMs (4E)**
MR FPP	100	15.575	6.46	1.26		100.630	3.235
MR CR	300	0.183	5.68	1.01		1.040	0.033
UG2	100	25.168	2.98	1.50		74.891	2.408
Total Indicated		40.926	4.31			176.561	5.676

Prill Splits	**Pt**	**Pt (g/t)**	**Pd**	**Pd (g/t)**	**Rh**	**Rh (g/t)**	**Au**	**Au (g/t)**
MR FPP	62%	4.02	26%	1.68	5%	0.33	7%	0.43
MR CR	62%	3.53	26%	1.48	5%	0.29	7%	0.38
UG2	64%	1.91	24%	0.72	10%	0.31	1%	0.04

Independently estimated Inferred Resource base:

MR FPP = Merensky Reef pegmatoidal feldspathic pyroxenite; MR CR = Merensky Reef contact reef; and UG2 = Upper Group No. 2 chromitite seam; PGM = Platinum-group metals.
The cut-offs for Indicated and Inferred Resources have been established by a qualified person after a review of potential operating costs and other factors.

Inferred Resource							
	Cut-off (cmg/t) 4E	**Million tons**	**Grade (g/t) 4E**	**Mining width (cm)**		**Tons PGM (4E)**	**Million ounces PGMs (4E)**
MR FPP	100	2.570	6.56	1.22		16.958	0.545
MR CR	300	0.001	3.50	1.00		0.004	0.0002
UG2	100	11.792	3.48	1.50		40.991	1.318
Total Inferred		14.363	4.03			57.953	1.8632

Prill Splits	**Pt**	**Pt (g/t)**	**Pd**	**Pd (g/t)**	**Rh**	**Rh (g/t)**	**Au**	**Au (g/t)**
MR FPP	62%	4.08	26%	1.70	5%	0.34	7%	0.44
MR CR	62%	2.18	26%	0.91	5%	0.18	7%	0.23
UG2	64%	2.23	24%	0.84	10%	0.36	1%	0.05

Status of exploration

PTM has completed approximately 58,559m of BQ-size core drilling (diameter 36.2mm) from borehole WBJV001 to WBJV120. Subsequent to the borehole cut-off for the resources declared in this report boreholes up to WBJV156 have been completed. Resource estimation is done according to SAMREC specifications by the kriging method. The drill spacing on the Indicated Resource is approximately 250m or in some instances as close as 125 metres. In keeping with best practice in resource estimation allowance is made for known and expected geological losses. The losses are estimated at 19% (8% may be ascribed to faults, 4% to dykes and 7% to iron-replacements) for the project resource area, and this has been considered in the resource estimate. The resulting resource model has been selected to be available for mining over a mineable cut.

Potential development considerations

A project team of ten specialists in mining engineering and related disciplines have spent the past ten months considering, designing and costing the optimum-potential extraction plan for the resources on WBJV Project 1. An effort was made to correspond with Anglo Platinum’s engineers of similar disciplines to consider the practical experience of Anglo Platinum’s team in the design factors.

This report is at a Pre-feasibility level and does not provide an assurance that the resources are legally or commercially viable. None of the resources is considered to be reserves and there is no assurance that they will ever be converted to reserves. See the Notice of this Technical Report. At this stage the costs are estimated at roughly 25% confidence with a contingency to reduce the risk of an overrun. The following outlines the mining engineering aspects of this Technical Report.

Mining method and approach

The mining approach utilizes well-known traditional mining methods applied extensively by the South African mining industry for extracting tabular and relatively thin deposits. The proposed mine design aims at extracting higher-grade Merensky Reef but the design also makes provision for the extraction of UG2 Reef. Rock mechanics are taken into consideration to facilitate, where possible, the extraction both reefs from the same infrastructure. The mine design implements experience gained on the adjacent BRPM property and makes use of the same mining method proven successful elsewhere in the platinum mining industry in the BIC.

This selected mining method is a conventional breast-panel, scattered-stilting layout largely utilising handheld equipment. Mechanised methods attempted at other platinum mines in the BIC, including at the adjacent BPRM, have been in many cases less than successful and these methods were discarded early on in the process of selecting a suitable mining method for Project 1.

As a conservative factor to reduce the risk, not only are geological losses considered at the resource level, but it is also assumed that one in every ten panels will not be mined. The three dimensional mine design scheduling was done using Surpac Vision and a scheduler – MineSched – was then used to schedule the mining by applying practical constraints in a real time plan calendar. Among other factors, rates of underground development were determined considering actual experiences at platinum mines in the vicinity.

Flow-of-ore factors

The following important factors were applied to the resource grades in the planning of the flow of ore to the millhead:

Source	**Factor**
Gully Dilution	4.4% of tons milled
Reef Development Dilution	2.8% of tons milled
Shortfall (Dilution)	5.0% of tons milled
Mine Call Factor (MCF)	97.5%

Concentrator	
Merensky Reef	87.5%
UG2 Reef	82.5%

These factors are benchmarked against the industry norms on tabular platinum mines in South Africa.

Mining selection

Turnberry and PTM developed preliminary mining plans on the Merensky and UG2 Reef horizons. During the Pre-feasibility study for Project 1 a number of access options were considered as well as a number of tonnage rate profiles. The scenarios considered consisted of the following options:

- Processing 140,000 tons per month - vertical shaft system with a decline
- Processing 140,000 tons per month - vertical shaft system
- Processing 120,000 tons per month - vertical shaft system with a decline
- Processing 120,000 tons per month - vertical shaft system
- Processing 160,000 tons per month - vertical shaft system with a decline
- Processing 160,000 tons per month - vertical shaft system
- Processing 200,000 tons per month - vertical shaft system with a decline
- Processing 200,000 tons per month - vertical shaft system

The practical aspects of accessing a reasonably steady supply of reef over the mine life and the economic returns applicable to these approaches were considered in a preliminary scope basis. The higher the processing rate, the shorter the mine life and the greater the challenge in providing enough access points to the deposit for the mining to achieve and sustain these rates. The ability to sustain a production rate at a modelled target is a function of the shape of the deposit as well as the mining and access methods used. In order for manpower and capital equipment to be utilised practically, it was desirable to have a steady state production of more than 5 years in the planning and evaluation process.

By defining the geological model and developing the underlying geological and structural interpretation, it became clear that the viably conservative, practical and economical tonnage rate is 140,000 tons per month. The scenarios processing 160,000 and 180,000 tons per month were considered in detail, however these options involve larger shaft diameters and higher capital costs; although the initial returns were higher on an Internal Rate of Return (IRR) basis the risks and practical challenges of sustaining the tonnage rates against the marginal increase in returns directed the focus to the more conservative 140,000 tons per month option. In order to optimise the selected tonnage rate and access method, consequently reducing the risks and providing solid returns, the PTM and Turnberry Projects teams focused their efforts on matching the mine plan to the associated geological and structural model.

Reaching a Bankable Feasibility stage, both a vertical shaft and a decline with vertical shaft takeover options will be investigated. The potential tonnage rates that could be supplied by initial declines is less than the 140,000 tons per month target tonnage rate profile, and economic analysis indicates that in the long term the decline option provides a diminished IRR when compared to a vertical shaft only. Considering current spot prices the decline and shaft options deliver identical IRRs.

Taking all the above into consideration as well as the Net Present Value (NPV) models and the establishment of an earlier mining production profile, both options are expected to be considered at the Bankable Feasibility stage – market price and exchange rate assumptions are to be revised at that time.

Conclusions – Qualified Person

Based on the long-term metal prices (as indicated in the subsequent list) as well as the resultant IRR basis value, the Turnberry Projects team selected a vertical shaft from surface to 712 meters below surface with seven working levels spaced at 60m intervals as the best option for consideration for the WBJV Project 1. However, as explained earlier, the near-term production advantages of a decline approach are potentially significant in that it would add to the existing information regarding geology, structure and metallurgy at the beginning stages of the project with possible reduction in the project risk profile. Therefore, both the vertical shaft alone and a decline with vertical shaft takeover models are recommended for further investigation.

Considering the vertical option, the resulting mine will have a planned production life of 17 years with steady state production at 140,000 tons per annum for 12 years and peak production at 250,000 ounces 4E – platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au) – per year in concentrate. The project financial evaluation indicates a potentially viable mine with a pre-tax Internal Rate of Return of 17.7% and Net Present Value of R2,423 million at a discount rate of 5% with a post-tax IRR of 13.4%. The life of mine capital expenditure is expected to be R2,457 million with peak funding at R2,060 million. The life of mine operating cost is expected to be R352 per ton milled or R95,942 per kilogram recovered.

It is important to note that the planned mine life could be affected either positively or negatively by fluctuations in market conditions.

The following long-term metal prices (not including discounts to be applied at smelter payment for concentrate) and exchange rate were used to determine a recommended maximum value option:

Platinum	US$	900 per ounce
Palladium	US$	330 per ounce
Rhodium	US$	2,000 per ounce
Gold	US$	500 per ounce
Ruthenium	US$	100 per ounce
Iridium	US$	250 per ounce
Copper	US$	1.31 per pound
Nickel	US$	4.65 per pound
Exchange Rate: R/US$		7.50

The construction of the concentrator plant will take between 18 and 24 months. In the capital expenditure programme in the financial model for the vertical shaft alone, this construction has been delayed from the project start since shaft sinking and underground development takes the longest lead-time. This delay may be modified if the decline option is selected; plant construction could then begin at the start of the project. Long lead-time items include mill plant delivery items estimated at approximately 78 weeks and supply of electric power from the Eskom grid at approximately 24 months for completion.

At long-term metal prices the decline option reduces the value of the project. Even though the pre-tax IRR is 17% this approach is a potentially viable option providing access to the shallow portions of the deposit.

If the current metal prices, as indicated below, are used in the financial model there is no financial difference between the vertical shaft and decline with vertical shaft takeover options; both having a pre-tax IRR of between 28.9 and 29.0%. The spot metal prices (not including discounts to be applied at smelter payment for concentrate) as at mid November 2006 were as follows:

Platinum	US$	1,204 per ounce
Palladium	US$	322 per ounce
Rhodium	US$	4,800 per ounce
Gold	US$	627 per ounce
Ruthenium	US$	100 per ounce
Iridium	US$	250 per ounce
Copper	US$	3.36 per pound
Nickel	US$	14.88 per pound
Exchange Rate: R/US$		7.25

Taking current metal prices as well as the 25% level of accuracy of the costing in this Technical Report into account, a single option cannot be selected and both are recommended to be detailed further in any Bankable Feasibility study.

Recommendations – Qualified Person

After considering a number of access scenarios, the QP and other experts involved recommend the vertical shaft option for Project 1 based on the best IRR and NPV at long-term metal prices.

The following degree of accuracy can be assumed for certain factors considered in this study:

- Capital Cost Estimate – ±25%
- Operating Cost Estimate – ±25%
- Project Timing Estimate – ±20%
- Project Output Estimate – ±20%

It is the recommendation of the QP and other qualified experts that a Bankable Feasibility study (BFS) be commissioned to improve the accuracy of the above estimates and progress the project to the next phase.

To the understanding of Turnberry Projects, the detailed scope of work and budget for the BFS would be developed in consultation with a committee of the WBJV, formed specifically for the development of a final Feasibility study as specified in the agreement. This committee will consider the risks and opportunities of the Pre-feasibility study and the objectives of the partners including, but not limited to the

- timing;
- profile of production;
- return hurdles; and
- partners' own short- and long-term planning.

Turnberry Projects welcomes the opportunity to assist in developing this scope of work.

SALIENT FEATURES PRE-FEASIBILITY STUDY REVIEW

RESOURCES

	Cut-off (cm g/t)	Mt	g/t	Average Mining Width (m)	Tons PGE (4E)	Moz
Measured	100	4.453	5.20	1.33	23.153	0.774
Indicated	~100	40.926	4.31	1.43	176.561	5.676
Inferred	~100	14.363	4.03	1.45	57.953	1.863

METAL PRICES & EXCHANGE RATE

		Case A Forward Prices	Case B Current Prices
Platinum	(US$/oz)	900	1204
Paladium	(US$/oz)	330	322
Rhodium	(US$/oz)	2000	4800
Gold	(US$/oz)	500	627
Copper	(US$/lb)	1.31	3.36
Nickle	(US$/lb)	4.65	14.88
R/US$		7.50	7.25

FINANCIAL PARAMETERS AND INDICATORS

Basket Prices	ZAR/kg Case A (Forward)	US$/oz Case A (Forward)	ZAR/kg Case B (Current)	US$/oz Case B (Current)
Basket Price Merensky Reef	185 800	771	250 415	1 074
Basket Price UG2 Reef	208 800	866	311 458	1 336

Smelter and Refinery Discount	ZAR/kg V# Only	US$/oz V# Only	ZAR/kg V# & Decline	US$/oz V# & Decline
Reduction in Basket Price for MR	28 870	120	40 812	175
Reduction in Basket Price for UG2	36 356	151	51 222	220
Received Basket Price for MR	156 930	651	209 603	899
Received Basket Price for UG2	172 444	715	260 236	1 116

Taxes		
Government Royalty Payment: PGM	(% of Rev.)	3%
Government Royalty Payment: Base Metals	(% of Rev.)	1%
Company Dividend Tax	(% of Pay.)	12.50%
Company Tax	(% of Profit)	29%

Capital	ZAR (Billion)	US$ (Million)
Capital: Vertical Shaft Option Only (Total)	2.457	328
Capital: Vertical Shaft Option Only (Peak Funding)	2.143	286
Capital: Vertical Shaft and Decline Option (Total)	2.663	355
Capital: Vertical Shaft and Decline Option (Peak Funding)	2.040	272
Contingency included in the capital cost	0.242	32

Working Costs (R/ton and US$/ton)	ZAR/ton	US$/ton
Vertical Shaft Only		
Opex Cost for Vertical Shaft Only	352	47
Decline and Vertical Shafts		
Opex Cost for Decline Only *(thereafter Vertical Shaft Value Applies)*	312	42
Opex Cost of Decline and Vertical Shafts	352	47

ITEM 4: INTRODUCTION

Item 4(a): Terms of reference

This report is compiled for PTML in terms of the Canadian National Instrument 43-101 *Technical Report* (Form F1) and the National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (Companion Policy). The information and status of the project is disclosed in the prescribed manner.

Item 4(b): Purpose of the report

The intentions of the report are to

- inform investors and shareholders of the progress of the project
- make public, update and detail the resource calculations and mine designs for the project.

Item 4(c): Sources of information

The independent author and Qualified Person (QP) of this report has used the data provided by the representative and internal experts of PTM. This data is derived from historical records for the area as well as information currently compiled by the operating company, which is PTM. The PTM-generated information is under the control and care of Mr WJ Visser SACNSP 400279/04, who is an employee of PTM and is not independent. The AP data pertaining to the deposit and earlier resource calculations has been under the control and custody of Anglo Platinum. An independent qualified person, Mr CJ Muller, has visited the property of the WBJV since the previous Canadian National Instrument 43-101 (NI 43-101) was released on 28 March 2006 by PTM and has undertaken a due diligence with respect to the data.

Item 4(d): Involvement of the Qualified Person: personal inspection

The listed independent QP has no financial or preferential relationships with PTM. The QP has a purely business-related relationship with the operating company and provides technical and scientific assistance when required and requested by the company. The QP has other significant client lists and has no financial interest in PTM.

ITEM 5: RELIANCE ON OTHER EXPERTS

In preparing this report the author relied upon

- land title information for Elandsfontein 102JQ and Frischgewaagd 96JQ as provided by PTM;
- geological and assay information supplied by PTM and made available by AP;
- borehole analytical and survey data compiled by PTM and verified by an additional external auditor (Mr N Williams);
- all other applicable information;
- data supplied or obtained from sources outside of the company; and
- assumptions and conclusions of other experts as set out in this report.

The sources were subjected to a reasonable level of appropriate inquiry and review. The author has access to all information and visited the property during September 2006 to review the core. The author's conclusion, based on diligence and investigation, is that the information is representative, accurate and forms a valid basis from which to proceed to a feasibility study.

This report was prepared in the format of the Canadian National Instrument 43-101 Technical Report by the QP, Mr GI Cunningham,. The QP has the appropriate background and relied on, among others, an independent expert with a geological and geostatistical background involved in the evaluation of precious metal deposits for over 17 years. The QP has reported and made conclusions within this report with the sole purpose of providing information for PTM's use subject to the terms and conditions of the contract between the QP and PTM. The contract permits PTM to file this report, or excerpts thereof, as a Technical Report with the Canadian Securities Regulatory Authorities or other regulators pursuant to provincial securities legislation, or other legislation, with the prior approval of the QP. Except for the purposes legislated for under provincial security laws or any other security laws, other use of this report by any third party is at that party's sole risk and the QP bears no responsibility.

Specific areas of responsibility

The QP accepts overall responsibility for the entire report. The QP and other experts involved is reliant with due diligence on the information provided by Mr WJ Visser, the internal and not independent expert. The qualified experts have also relied upon the input of the PTM geological personnel in compiling this filing.

ITEM 6: PROPERTY DESCRIPTION AND LOCATION

Item 6(a) and Item 6(b): Area and Location of project

The WBJV project is located on the southwestern limb of the BIC (see Diagram 1) some 35km northwest of the town of Rustenburg, North West Province. The property adjoins Anglo Platinum's Bafokeng Rasimone

Platinum Mine (BRPM) and the Styldrift project to the southeast and east respectively (see Diagram 2). The Project 1 area of interest consists of the farms Elandsfontein 102JQ and Frischgewaagd 96JQ (see Diagram 3) situated in the southeastern corner of the larger joint-venture area.

The total joint-venture area includes portions of PTM's properties Elandsfontein 102JQ and Onderstepoort 98JQ, and also certain portions of Elandsfontein 102JQ, Onderstepoort 98JQ, Frischgewaagd 96JQ and the whole of Koedoesfontein 94JQ contributed by RPM, a wholly-owned subsidiary of Anglo Platinum (see Item 6(c) below for detail). These properties are centred on Longitude 27 ^{o} 00' 00'' (E) and Latitude 25 ^{o} 20' 00'' (S) and the mineral rights cover approximately 67km ^{2} or 6,700ha.

Item 6(c): Licences

The areas discussed in this report have been subdivided into several smaller portions as each area has its own stand-alone licence and Environmental Management Programme (EMP). Within the WBJV property there are nine separate licences and they are specifically listed in the manner below for cross-referencing to the licence specifications. The licences over the WBJV area are as follows:

1. Elandsfontein (PTM)
2. Elandsfontein (RPM)
3. Onderstepoort (PTM) 4, 5 and 6
4. Onderstepoort (PTM) 3 and 8
5. Onderstepoort (PTM) 14 and 15
6. Onderstepoort (RPM)
7. Frischgewaagd (RPM)
8. Frischgewaagd (PTM)
9. Koedoesfontein (RPM)

Applications have been made in a timely fashion for conversion to the new Mineral and Petroleum Resources Development Act, 2002. Prospecting is continuing during the conversions in progress.

1. Prospecting on **Elandsfontein (PTM)** , viz.
    - Elandsfontein 102JQ Portion 12 (a portion of Portion 3) (a total area of 213.4714ha)
    - Elandsfontein 102JQ Portion 14 (a total area of 83.4968ha) and
    - Remaining Extent of Elandsfontein 102JQ Portion 1 (a total area of 67.6675ha)

was originally carried out under the now expired prospecting permit (no. PP269/2002 reference RDNW (KL) 5/2/2/4477). A new prospecting permit application was submitted by PTM on 12 October 2003. The prospecting right documentation was notarially executed under protocol no. 467/2005 and the Minister of Minerals and Energy duly granted a new-order prospecting right to PTM as the holder of such prospecting right in terms of the provisions of Section 17 of the Mineral and Petroleum Resources Development Act, 2002

on 17 August 2005. The prospecting right will endure for a period of 3 (three) years with effect from 17 August 2005 to 15 September 2008. The prospecting right has been lodged for registration at the Mineral and Petroleum Titles Registration Office in Pretoria.

2. Prospecting on **Elandsfontein (RPM)** , viz.
   - Elandsfontein 102JQ Portion 8 (a portion of Portion 1) (a total area of 35.3705ha) and
   - Elandsfontein 102JQ Portion RE9 (a total area of 403.9876ha).

A prospecting permit was issued on 23 March 2004 and expired on 24 March 2006. The prospecting permit (no. PP50/1996) was issued on 11 March 2004 (reference RDNW (KL) 5/2/2/2305) and was valid until 10 March 2006. The second prospecting permit no. is PP73/2002 (reference RDNW (KL) 5/2/2/4361). This permit covers Mineral Area 2 (a portion of Mineral Area 1) (total area of 343.5627ha) of the farm Elandsfontein 102JQ. A conversion to a new-order prospecting right was approved.

3. The prospecting permit over **Onderstepoort (PTM) Portions 4, 5 and 6** was awarded on 30 April 2004 (reference no. RDNW (KL) 5/2/24716, PP No.48/2004) and was valid until 30 April 2006. The relevant entities are
   - Onderstepoort 98JQ Portion 4 (a portion of Portion 2) (a total area of 79.8273ha)
   - Onderstepoort 98JQ Portion 5 (a portion of Portion 2) (a total area of 51.7124ha) and
   - Onderstepoort 98JQ Portion 6 (a portion of Portion 2) (a total area of 63.6567ha).

An application for the conversion of the prospecting permit was lodged on 19 April 2006 and duly accepted. The converted prospecting right documentation was notarially executed under protocol no. 879/2006, and the Minister of Minerals and Energy duly granted a convertion to a new-order prospecting right to PTM as the holder of such converted prospecting right in terms of the provisions of Item 6 of Schedule II of the Mineral and Petroleum Resources Development Act, 2002 on 5 October 2006. The converted prospecting right will endure for a period of 3 (three) years with effect from 5 October 2006 to 4 October 2009. The converted prospecting right has been lodged for registration at the Mineral and Petroleum Titles Registration Office in Pretoria.

4. A prospecting permit application over **Onderstepoort (PTM) 3 and 8** was issued on 24 March 2004, (permit no. PP26/2004 reference RDNW (KL) 5/2/2/4717) and was valid until 23 April 2006. The applicable entities are:
   - Onderstepoort 98JQ Remaining Extent of Portion 3 (a total area of 274.3291ha) and
   - Onderstepoort 98JQ Portion 8 (a portion of Portion 1) (a total area of 177.8467ha).

An application for the conversion of the prospecting permit was lodged on 19 April 2006 and accepted. The converted prospecting right documentation was notarially executed under protocol no. 881/2006 and the Minister of Minerals and Energy duly granted a convertion to a new-order prospecting right to PTM as the holder of such converted prospecting right in terms of the provisions of Item 6 of Schedule II of the Mineral and Petroleum Resources Development Act, 2002 on 5 October 2006. The converted prospecting right will

endure for a period of 3 (three) years with effect from 5 October 2006 to 4 October 2009. The converted prospecting right has been lodged for registration at the Mineral and Petroleum Titles Registration Office in Pretoria.

5. A new-order prospecting right for **Onderstepoort (PTM) 14 and 15** , viz.
   - Onderstepoort 98JQ now consolidated under Mimosa 81JQ Portion 14 (a portion of Portion 4) (total area of 245.2880ha) and
   - Onderstepoort 98JQ now consolidated under Mimosa 81JQ Portion 15 (a portion of Portion 5) (a total area of 183.6175ha)

was granted to PTM on 25 April 2005. The new prospecting right was notarially executed under protocol no. 7 and is in force for a period of 3 (three) years terminating on 24 April 2008. The new prospecting right has been lodged for registration at the Mineral and Petroleum Titles Registration Office in Pretoria.

6. A new-order prospecting right for **Onderstepoort (RPM)** (Onderstepoort previous Portion 9) (a portion of Portion 3) (127.2794ha) has been applied for. A new-order prospecting right has also been applied for over Mineral Area 1 (total area of 29.0101ha) of Ruston 97JQ that was consolidated under Mimosa 81JQ. A permit has also been applied for over Mineral Area 2 (total area of 38.6147ha) of the farm Ruston 97JQ which is also consolidated under Mimosa 81JQ. Both applications are awaiting Government approval.

7. A prospecting permit was issued to RPM over **Frischgewaagd (RPM)** covering a 23/24th share of the undivided mineral rights (permit no. PP294/2002 reference RDNW (KL) 5/2/2/4414) relating to the following portions:
   - Frischgewaagd 96JQ Portion RE4 (286.8951ha)
   - Frischgewaagd 96JQ Portion 3 (made up of Portion RE and Portion 13) (466.7884ha)
   - Frischgewaagd 96JQ Portion 2 (made of up Portion RE2 and Portion 7 (a portion of Portion 2)) (616.3842 + 300.7757ha)
   - Frischgewaagd 96JQ Portion 15 (78.7091ha)
   - Frischgewaagd 96JQ Portion 16 (22.2698ha) and
   - Frischgewaagd 96JQ Portion 18 (45.0343ha).

The permit was valid until 16 October 2004. A conversion to a new-order prospecting right was approved.

8. On 16 November 2005 PTM submitted a new-order prospecting right application over **Frischgewaagd (PTM)** (Frischgewaagd 96JQ) for the remaining undivided mineral rights. The application covers the same area of interest as that of permit no. PP294/2002 (reference RDNW (KL) 5/2/2/4414) issued to RPM (see above paragraph):
   - Frischgewaagd 96JQ Portion RE4 (286.8951ha)
   - Frischgewaagd 96JQ Portion 3 (made up of Portion RE and Portion 13) (466.7884ha)
   - Frischgewaagd 96JQ Portion 2 (made of up Portion RE2 and Portion 7 (a portion of Portion 2)) (616.3842 + 300.7757ha)
   - Frischgewaagd 96JQ Portion 15 (78.7091ha)
   - Frischgewaagd 96JQ Portion 16 (22.2698ha) and
   - Frischgewaagd 96JQ Portion 18 (45.0343ha).

The Deputy Director-General (Mineral Regulation) advised PTM in writing on 25 October 2006 that a new-order prospecting right would be notarially executed shortly at the Regional Manager's Office of the Department of Minerals and Energy (DME) in Klerksdorp. The new prospecting right would thereafter be registered in the Mineral and Petroleum Titles Registration Office in Pretoria.

9. A prospecting permit was issued to RPM over **Koedoesfontein (RPM)** 94JQ (2795.1294ha) on 19 March 2004 under prospecting permit no. PP70/2002 (reference 5/2/2/4311) and was valid until 18 March 2006. A notarially executed new-order prospecting right was approved.

Item 6(d): Rights to surface, minerals and agreements

Regarding *Elandsfontein (PTM),* the purchase agreement was settled by way of an Agreement of Settlement, which was signed on 26 April 2005. Party to this agreement was a Sale Agreement. The Agreement of Settlement has entitled PTM to the rights to the minerals as well as the freehold. PTM has taken possession of the property.

Option agreements in respect of *Onderstepoort (PTM)* have been signed with the owners of the mineral rights on Portions *Onderstepoort 4, 5 and 6* ; *Onderstepoort 3 and 8* ; and *Onderstepoort 14 and 15* . The option agreement over Portions 3 and 8 requires a payment of C$1,000 after signing, C$1,000 after the granting of the prospecting permit and C$1,000 on each anniversary of the agreement. The option agreement for Portions 4, 5 and 6 requires a payment of R5,014 after signing, R3,500 on the first anniversary, R4,000 on the second anniversary and R4,500 on the third anniversary. The option agreement for Portions 4, 5, 6, 14 and 15 requires a payment of R117,000 after signing and payments of R234,000 and R390,000 within 10 days of the effective date. All payments are current and up to date.

WBJV terms

The detailed terms of the WBJV – relating to *Elandsfontein (PTM), Elandsfontein (RPM), Onderstepoort (PTM), Onderstepoort (RPM), Frischgewaagd (PTM), Frischgewaagd (RPM)* and *Koedoesfontein (PRM)* – were announced on 27 October 2004. The WBJV will immediately provide for a 26% Black Economic Empowerment interest in satisfaction of the 10-year target set by the Mining Charter and newly enacted Mineral and Petroleum Resources Development Act 28, 2002. PTM and RPM will each own an initial 37% working interest in the farms and mineral rights contributed to the joint venture, while AW will own an initial 26% working interest. AW will work with local community groups in order to facilitate their inclusion in the economic benefits of the joint venture, primarily in areas such as equity; the work will also involve training, job creation and procurement in respect of historically disadvantaged South Africans (HDSAs).

The WBJV structure and business plan complies with South Africa's recently enacted minerals legislation. Platinum exploration and development on the combined mineral rights of the WBJV will be pursued.

PTM, as the operator of the WBJV, undertook a due diligence on the data provided by RPM. PTM undertook to incur exploration costs in the amount of R35 million over a five-year period starting with the first three years at R5 million and increasing to R10 million a year for the last two, with the option to review yearly. The expenditure to date is in excess of PTM's obligations to the joint-venture agreement.

The Government of South Africa has proposed a 3% Gross Royalty on platinum production.

Item 6(e): Survey

Elandsfontein (PTM) and Elandsfontein (RPM) are registered with the Deeds Office (RSA) under Elandsfontein 102JQ, North West Province and measures 364.6357ha. The farm can be located on Government 1:50,000 Topo-cadastral sheet 2527AC Sun City (4th Edition 1996) which is published by the Chief Directorate, Surveys and Mapping (Private Bag X10, Mowbray 7705, RSA, Phone: +27 21 658 4300, Fax: +27 21 689 1351 or e-mail: cdsm@sli.wcape.gov.za). The approximate coordinates (WGS84) are 27 ^{o} 05' 00'' (E) and 25 ^{o} 26' 00'' (S).

Onderstepoort (PTM) and *Onderstepoort (RPM)* are registered with the Deeds Office (RSA) under Onderstepoort 98JQ, Northern Province and measures 1,085.2700ha. The farm can be located on Government 1:50,000 Topo-cadastral sheet 2527AC Sun City (4th Edition 1996) which is published by the Chief Directorate, Surveys and Mapping. The approximate coordinates (WGS84) are 27 ^{o} 02' 00'' (E) and 25 ^{o} 07' 00'' (S).

Frischgewaagd (PTM), Frischgewaagd (RPM) and *Koedoesfontein (RPM)* : Frischgewaagd is registered with the Deeds Office (RSA) under Frischgewaagd 96JQ, Northern Province and measures 1,836.8574ha and Koedoesfontein is registered with the Deeds Office (RSA) under Koedoesfontein 94JQ, Northern Province and measures 2,795.1294ha. Both farms can be located on Government 1:50,000 Topo-cadastral sheet 2527AC Sun City (4th Edition 1996) which is published by the Chief Directorate, Surveys and Mapping. The approximate coordinates (WGS84) are 27 ^{o} 02' 00'' (E) and 25 ^{o} 07' 00'' (S).

Item 6(f): Mineralised zones

The BIC in general is well known for containing a large share of the world's platinum and palladium resources. There are two very prominent economic deposits within the BIC. Firstly, the Merensky Reef (MR) and the Upper Group 2 (UG2) chromitite, which together can be traced on surface for 300km in two separate areas. Secondly, the Northern Limb (Platreef), which extends for over 120km in the area north of Mokopane.

In the past the Bushveld's platinum- and palladium-bearing reefs have been estimated at about 770 and 480 million ounces respectively (down to a depth of 2,000 metres). These estimates do not distinguish between the categories of Proven and Probable Reserves and Inferred Resource. Recent calculations suggest about 204 and

116 million ounces of Proven and Probable Reserves of platinum and palladium respectively, and 939 and 711 million ounces of Inferred Resources. Mining is already taking place at 2km depth in the BIC. Inferred and ultimately mineable ore resources can almost certainly be regarded as far greater than the calculations suggest. These figures represent about 75% and 50% of the world's platinum and palladium resources respectively. Reserve figures for the Proven and Probable categories alone in the BIC appear to be sufficient for mining during the next 40 years at the current rate of production. However, estimated world resources are such as to permit extraction at a rate increasing by 6% per annum over the next 50 years. Expected extraction efficiency is less for palladium. Thereafter, down-dip extensions of existing BIC mines, as well as lower-grade areas of the Platreef and the Middle Group chromitite layers, may become payable. Demand, and hence price, will be the determining factor in such mining activities rather than availability of ore.

Exploration drilling to date on the WBJV area has shown that both economic reefs (Merensky and UG2) are present and economically exploitable on the WBJV properties. The separation between these reefs tends to increase from the subcrop environment (less than five metres apart) to depths exceeding 650 metres (up to 50 metres apart) towards the northeast. The subcrops of both reefs generally strike southeast to northwest and dip on average 14 degrees to the northeast. The reefs locally exhibit dips from 4 to 42 degrees (average 14 degrees) as observed from borehole information.

The most pronounced Platinum-group metal (PGM) mineralisation along the western limb of the BIC occurs within the Merensky Reef and is generally associated with a 0.1–1.2m-thick pegmatoidal feldspathic pyroxenite unit. The second important mineralised unit is the UG2 chromitite layer, which is on average 0.6–2.0m thick.

Item 6(g): Liabilities and payments

All payments and liabilities are recorded under Item 6(d).

Item 6(h) and Item 6(i): Environmental liabilities and Prospecting permits

There are no known environmental issues relating to the PTM or WBJV properties.

Mining and exploration companies in South Africa operate with respect to environmental management regulations in Section 39 of the Minerals Act (1991) as amended. Each prospecting area or mining site is subject to conditions such as that

- environmental management shall conform to the EMP as approved by the DME;
- prospecting activities shall conform to all relevant legislations, especially the National Water Act (1998) and such other conditions as may be imposed by the director of Minerals Development;

- surfaces disturbed by prospecting activities will be rehabilitated according to the standard laid down in the approved EMPs;
- financial provision will be made in the form of a rehabilitation trust and/or financial guarantee;
- a performance assessment, monitoring and evaluation report will be submitted annually.

Prospecting permits are issued subject to the approval of the EMP, which in turn is subject to provision of a financial guarantee.

On *Elandsfontein (PTM)* the operator conducted exploration under an EMP approved for a prospecting permit granted to Royal Mineral Services on 14 November 2002 (now expired). A new application for a prospecting permit and an EMP has been lodged with the DME in the name of PTM and has been approved. A follow-up EMP was requested by the DME and was compiled by an independent consultant (Geovicon CC, Mike Bate) and filed on 23 August 2004. The updated EMP was accepted by the DME on 20 October 2004. The EMP financial guarantee with respect to this application is held by Standard Bank of South Africa (guarantee no. M410986) in the amount of R10,000. In terms of the notarial prospecting agreement (Clause 10) the Minister or authorised person has the right to inspect the performance of the company with regard to environmental matters.

With regard to the *Onderstepoort (PTM)* area that was contributed to the WBJV by PTM, all EMPs were lodged with the DME and approved on 30 April 2004 and 24 April 2004 for *Onderstepoort 4,5 and 6* and *Onderstepoort 3 and 8* respectively. Financial provision of R10,000 for each of the optioned areas have been lodged with Standard Bank (guarantee no. TRN M421362 for *Onderstepoort 4, 5 and 6* ; guarantee no. TRN M421363 for *Onderstepoort 3 and 8* ; and M421364 for *Onderstepoort 14 and 15*).

Regarding *Onderstepoort 14 and 15* , a follow-up EMP was requested by the DME and was compiled by an independent consultant (Geovicon CC, Mike Bate) and filed on 23 August 2004. The updated EMP was accepted by the DME on 20 October 2004. The financial guarantee of R10,000 in respect of this application is held by Standard Bank of South Africa (guarantee no. M410986). In terms of the notarial prospecting agreement (Clause 10) the Minister or authorised person has the right to inspect the performance of the company with respect to environmental matters.

In the areas of the WBJV that were originally owned by RPM, PTM will take responsibility for the EMPs that originated from RPM in respect of Elandsfontein, Onderstepoort, Frischgewaagd and Koedoesfontein. PTM as operator of the joint venture will be the custodian and will be responsible for all aspects of the Environmental Management Programmes and for all specifics as set out in all the various allocated and approved EMPs for properties that form part of the WBJV.

With respect to *Elandsfontein (RPM)* (Portions 8 and 9 of Elandsfontein 102JQ) there is an EMP dated 26 February 2004. There is also an EMP dated 11 March 2004 for portions of Mineral Area 2 (a portion of Mineral Area 1) of the farm Elandsfontein 102JQ.

Regarding *Frischgewaagd (RPM)* – Remaining Extent of Portion 4, Portion 3 (a portion of Portion 1), Portions 15, 16, 18, 2 and 17 (a portion of Portion 10) – an EMP dated 22 September 2002 exists.

The EMP for *Onderstepoort (RPM)* was submitted together with the prospecting permit application.

The EMP for *Koedoesfontein (RPM)* was received by the DME on 22 September 2002.

ITEM 7: PHYSIOGRAPHY, ACCESSIBILITY AND LOCAL RESOURCES

Item 7(a): Topography, elevation and vegetation

Topography

Topographically, the WBJV area is located on a central plateau characterised by extensive savannah with vegetation consisting of grasses and shrub with few trees. The toal elevation relief is greater as prominent hills occur in the northern most portions, but variations in topographical relief are minor and limited to low, gently sloped hills.

The *Elandsfontein* and *Frischgewaagd* properties gently dip in a northeasterly direction towards a tributary of the Elands River. Elevations range from 1,080 metres above mean sea level (AMSL) towards the Elands River in the north to 1,156m AMSL towards Onderstepoort in the southwest, with an average of 1,100m AMSL. On the *Onderstepoort* property to the west of the project area, the site elevation is approximately 1,050m AMSL with the highest point at 1.105m AMSL. The project area is bounded on the north by the Elands River, a perennial stream draining to the northeast. Minor drainage into the Elands River is from south to north on the area of concern.

Groundwater

GCS Groundwater Consulting Services undertook a preliminary groundwater investigation. The key aspects of the study are summarised as follows.

Groundwater flow system

The groundwater level elevation correlates well with surface elevation. The general groundwater flow direction is north and northeast towards the surface runoff channels. The groundwater flow gradient varies between 0.02 and 0.0004.

Aquifer type

Groundwater within the project area generally occurs in secondary aquifers created by weathered and fractured geological processes. Beneath the project area the aquifer is unconfined to semi-confined, and is classified as a minor aquifer. The average depth of the groundwater table is 26m below surface. The depth of the water level varies between three metres to more than 60 metres. The deep water level is mainly attributed to the large-scale extraction for irrigation.

Groundwater quality

The majority of water boreholes in the area have a depth of 40–80m. No water strike data is available for the project area, but the available borehole information from investigations in the adjacent regions shows that water-strikes occur at 23m.

Chemical analyses of five water samples taken during this study show that the dominant water type is Mg-Ca-HCO_3. The groundwater chemical data indicates that magnesium, calcium and total dissolved solids (TDS) are generally present in concentrations exceeding the Department of Water Affairs and Forestry (DWAF) Domestic Water Quality Guidelines. This can be attributed to the geology in the project area. Two boreholes located within villages show elevated nitrate concentrations attributed to human activities.

Groundwater users

Groundwater usage in the area is primarily for domestic purposes, livestock watering as well as irrigation. The total groundwater extraction from the study area is 30,000 litres per day.

Surface water

Forming the northern boundary of the projet area, the Elands River is the major source of surface water in close proximity to the proposed mining site. This watercourse flows directly into the Vaalkop Dam, which is the main source of water in the area.

Catchment boundaries

Stream drainage is directed towards the northeast and feeds into the Elands River, which forms the northern boundary of the area. The project area lies in the quaternary sub-catchment A22F, which forms part of the Elands River sub-catchment of the Limpopo drainage region.

Surface water use

The water from the Vaalkop dam is treated at the Vaalkop purification facility before it is distributed to the end-users for domestic purposes and livestock watering. Downstream users rely upon this dam as a consistent source of water.

Surface water authority

The surface water authority is the Department of Water Affairs and forestry (DWAF) and the water service providers for the area are Rand Water (RW) and Magalies Water (MWB) – currently forming part of RW.

Air quality

The ambient air quality is good as the activities in the area area mainly agriculture and grazing. The main impact on the air quality is vehicle emissions. Concerning the regional air quality, it is heavily impacted by SO_4 emissions from smelter operations in the area.

Soils

The soils are moderate to deep, black and red clay, with thin sandy loam soils to the east. The agricultural potential of North West Province soils is generally limited with a topsoil of 0–300mm thick. The erodibility index is 5 (high) and the average sub-catchment sediment yield is 83 x $10m^3$ tons per annum.

Land use

The main land use on the project area is mining, agriculture and grazing. The area comprises mostly land suitable for grazing and arable land for certain crops only. Typical animal life of the Bushveld has largely disappeared from the area owing to farming activities. Efforts are being made by the Norht West Parks Board to reintroduce the natural animal populations in parks such as Pilanesberg and Madikwe. Individual farmers also are moving from traditional cattle farming to game farming, and organised hunting is becoming a popular means of generating income.

Fauna

The project area consists of natural habitats with operational ecosystems despite areas of disturbance within these habitats. No habitat of exceptional sensitivity or concern exists.

Birds

Approximately one third (328 species) of the roughly 900 bird species of South Africa occur in the Rustenburg/Pilanesberg area. The most characteristic of these include lilac-breasted rollers, African hoopoes and owls. The Red Data bird species that occur (*) or could potentially occur (**) in the study area are listed in the following table:

Species occurring in the area*	**Habitat**
Martial Eagle (V)	Tolerates a wide range of vegetation types found in open grassland, shrub, Karoo and woodland.
Species potentially occurring in the area**	
African Whitebacked Vulture (V)	Nests in large trees, transmission and reticulation power lines.
Tawny eagle (V)	Occurs mainly in woodlands as well as lightly wooded areas.
Blue Crane (V)	Dry short grassland. Not very dependent on wetlands habitat for breeding. Preferred nesting sites are secluded open grasslands as well as agricultural fields.
Grass Owl (V)	Breeding in permanent and seasonal vleis. Vacates while hunting or post breeding.

Red Data status = V = vulnerable.

Herpetofauna

In total 143 species of herpetofauna occur in the North West Province. This is considered high as it accounts for roughly one third of the total occurring in South Africa. Monitor lizards and certain snake and gecko species are found in the project area. The table below shows Red Data species in the North West Province.

Scientific name	**English name**	**Conservation Status**
Python natalensis	Southern African Python	Vulnerable
Homoroselaps dorsalis	Striped Harlequin Snake	Rare
Dalophia pistillum	Blunt-tailed worm lizard	Data Deficient
Crocodylus niloticus	Nile crocodile	Vulnerable
Pyxicephalus adspersus	Giant Bullfrog	Near Threatened

Habitats for all the above-named species, excluding the Nile crocodile, occur on the project area with the wetland patches along the stream potentially suitable habitats for Giant Bullfrog.

Mammals

The Southern Greater Kudu found in North West Province are among the biggest in the country. On the project area it is expected that larger antelope such as gemsbok, Cape eland, common waterbuck, impala, and red hartebeest may be kept on the farms, while smaller cats, viveriids, honey badgers, and vervet monkeys should occur as free-roaming game. The project area could potentially be a habitat for the following Red Data species.

Scientific Name	**English Name**	**Conservation Status**
Atelerix frontalis	South African hedgehog	Rare
Proteles cristatus	Aardwolf	Rare
Hyaena brunnea	Brown hyena	Rare

Panthera pardus	Leopard	Rare
Mellivora capensis	Honey badger	Vulnerable

Flora

This general area's vegetation is classified as Mixed Bushveld. Where the soil is mostly coarse, sandy and shallow, and overlies granite, quartzite, sandstone or shale, the vegetation varies from a dense, short bushveld to a fairly open tree savannah. On shallow soils Red Bushwillow Combretum apiculatum dominates the vegetation. Other trees and shrubs include Common Hook-thorn Acacia caffra, Sicklebush Dichrostachys cinerea, Live-long, Lannea discolor, Sclerocarya birrea and various Grewia species. Here the grazing is sweet, and grasses such as Fingergrass Digitaria eriantha, Kalahari Sand Quick Schmidtia pappophoroides, Wool Grass Anthephora pubescens, Stipagrostis uniplumis, and various Aristida and Eragrostis species dominate the herbaceous layer. On deeper and more sandy soils, Silver Clusterleaf Terminalia sericea becomes dominant, with Peeling Plane Ochna pulchra, Wild Raisin Grewia flava, Peltophorum africanum and Burkea africana often prominent woody species, while Broom Grass Eragrostis pallens and Purple Spike Cat's tail Perotis patens are characteristically present in the scanty grass sward.

Specifically, the project area is located in the Clay Thorn Bushveld – Bredenkamp and Van Rooyen (1996) – vegetation type in the Savannah Biome – Rutherford and Westfall (1994). The vegetation of the eastern section of Elandsfontein is dominated by and closed Acacia tortilis vegetation, which is typical of Clay Thorn Bushveld, with other species such as Rhus lancea, Ziziphus mucronata and Rhus pyroides adding to the species richness. The closed woodland areas occur along the main road where cattle kraals are located as well as along the drainage line. Some fallow lands occur in this area where a good grass layer dominated by species such as Themeda triandra, Cymbopogon contortis, Botriochloa bladhii and Sorghum versicolor has re-established as well as a sparse tree layer. The areas on the western section of Elandsfontein consist of a fenced game reserve as well as a natural area further to the north near the Elands River. The tree and herbaceous layer is more diverse in this area where the tree layer is dominated by Ziziphus mucronata, Acacia tortilis and the shrub Grewia flava. The following table shows flora species in this vegetation type (* exotic species).

Forbs	**Grasses**	**Trees&Shrubs**
Tephrosia capensis	*Themeda triandra*	*Euclea divinorum*
Commelina erecta	*Eragrostis superba*	*Diospyros lycioides*
Crotolaria eremicola	*Cymbopogon plurinodus*	*Ziziphus mucronata*
Chamaecrista comosa	*Sorghum versicolor D*	*Vangueria infausta*
Ruellia patula	*Urelytrum agropyroides*	*Rhus lancea*
Clematis brachiata	*Aristida bipartita*	*Grewia flava*
Eriosema cordatum	*Pennisetum thunbergii*	*Acacia tortilis*
*Tithonia rotund i'folia**	*Heteropogon contortis*	*Carissa bispinosa*

*Tagetes minuta**	*Melinis repens*	*Olea europaea*
*Datura stramonium**	*Panicum deustum*	*Asparagus laricinus*
*Schkuria pinnata**	*Digitaria eriantha*	*Rhus pyroides*
Vernonia oligocephala	*Eragrostis rigidior*	*Acacia karroo*
Sebaea grandis	*Choris virgata*	*Gymnosporia buxifolia*
Crabbea angustifolia	*Urochloa mosambicensis*	
Rhynchosia minima	*Setaria sphacelata*	
Tephrosia capensis	*Eragrostis capensis*	
Hybiscus trionum	*Cymbopogon excavatus*	
Commelina erecta	*Cymbopogon sp.*	
*Tithonia rotundi'folia**	*Botriochloa insculpta*	
	Urelytrum agropyroides	
	Cymbopogon excavatus D	*Acacia karroo*
	Botriochloa insculpta D	*Ziziphus mucronata*
	Botriochloa bladhii	*Rhus lancea*
	Themeda triandra	*Rhus pyroides*
	Sorghum versicolor	*Dombeya rotundifolia*
		Grewia flava
Berkheya radula	*Echinochloa colona D*	*Euclea divinorum*
Pavonia burchelii	*Brachiaria brizantha D*	*Diospyros lycioides*
Crinum bulbispermum	*Paspalum urvillei*	*Ziziphus mucronata*
		Grewia flava
		Rhus lancea

Item 7 (b): Means of access to the property

South Africa has a large and well-developed mining industry in the area where the project is located. This, among other factors, means that the infrastructure is well established, with well-maintained highways and roads as well as electricity distribution networks and telephone systems.

The project area is located on the southwestern limb of the BIC, some 35km northwest of the North West Province town of Rustenburg. The town of Boshoek is situated 10km to the south along the tar road that links Rustenburg with Sun City and crosses the project area. The WBJV adjoins the AP-managed BRPM to the southeast. A railway line linking BRPM to the national network passes the project area immediately to the east with a railway siding at Boshoek.

The WBJV properties are readily accessible from Johannesburg by travelling 120km northwest on Regional Road 24 to the town of Rustenburg and then a further 35km. The resort of Sun City is located approximately

10km north of Project 1 (see Diagram 2). Both BRPM to the southeast of the project area and Styldrift – a joint venture between the Royal Bafokeng Nation and Anglo Platinum, which lies directly to the east of the property – has modern access roads and services. Numerous gravel roads crossing the WBJV properties provide easy access to all portions.

Item 7(c): Population centres and modes of transport

The major population centre is the town of Rustenburg about 35km to the southeast of the project. Pretoria lies approximately 100km to the east and Johannesburg about 120km to the southeast. A popular and unusually large hotel and entertainment centre, Sun City, lies about 10km to the north of the project area. The Sundown Ranch Hotel lies in close proximity to the project area and offers rooms and chalets as accommodation. The WBJV properties fall under the jurisdiction of the Moses Kotane Municipality. A paved provincial road crosses the property. Access across most of the property can be achieved by truck without the need for significant road building.

Noise

The area has a rural residential character and the main sources of noise are local traffic, community-related activities and natural sounds. Despite the fact that there are existing mining activities in the are, ambient or background noise levels are rather low.

Item 7(d): Climate

With low rainfall and high summer temperature, the area is typical of the Highveld Climatic Zone. Temperatures in this climatic zone are generally mild, with mean annual maximum temperatures of 26.4°C, and mean monthly maximum temperatures of more than 30°C in summer. In winter low mean annual minimum temperatures of 10.9°C are experienced, with mean monthly minimums as low as 2.8° C.

Temperature

In summer (November to April) the days are warm to hot and generally sunny in the morning, with afternoon showers or thunderstorms; temperatures average 26ºC (79ºF) and can rise to 38ºC (100ºF); and night temperatures drop to around 15ºC (60ºF). During winter months (May to October) days are dry and sunny with moderate to cool temperatures, while evening temperature drop sharply. Temperatures by day generally reach 20ºC (68ºF) and can drop to below 0ºC with frost occurring in the early morning. The hottest months are generally December and January with June and July being the coldest.

Monthly and annual rainfall

The area is considered semi arid with an annual rainfall of 520mm, which is below the average rainfall in South Africa. The rainy season is in the summer months from October to April with the highest rainfall in December and January.

MONTHLY AVERAGES		J	F	M	A	M	J	J	A	S	O	N	D
TEMP	(℃)	24.1	23.2	22.0	18.4	15.0	11.7	12.0	14.8	18.8	21.3	22.6	23.7
SUNSHINE	(HRS)	259	237	246	218	268	261	290	306	298	276	250	274
RAINFALL	(mm)	117	83	74	57	14	5	3	5	13	37	64	67

The table below is a guide to monthly averages for temperature, sunshine and rainfall for the region. (Reported within the submitted EMP which was lodged in conjunction with the prospecting permit application: investigation conducted by DWA, a contractor trading under the name of Digby Wells and Associates, Environmental Solutions Provider).

Wind

Analysis of the meteorological data showed that the prevailing winds are from the east and southeast and wind speed averages 2.5m/s.

Evaporation

Potential A-span evaporation figures for the area exceed the rainfall. This gives an indication of water deficiency in the area. The average monthly evaporation figures are shown in the following table:

Month	J	F	M	A	M	J	J	A	S	O	N	D
Avg	224.7	180.4	173.4	132.5	119.4	96.7	109.0	152.8	193.6	224.3	215.2	232.6

Extreme weather conditions

The area is prone to drought conditions resulting in limited availability of groundwater, and frost is not uncommon during the winter period. Rainfall occurs in the form of showers and thunderstorms with which strong gustly winds are associated.

Exploration is conducted year-round and is unaffected by the climate.

Item 7(e): Infrastructure with respect to mining

As this report details the exploration programme, it suffices to note that all areas are close to major towns and informal settlements as a potential source of labour with paved roads being the norm. Power lines cross both project areas and water is as a rule drawn from boreholes. As several platinum mines are located adjacent to

and within 50km of the property there is excellent access to materials and skilled labour. One of the smelter complexes of AP is located within 60km of the property.

Surface rights as to 365ha on Elandsfontein have been purchased in the area near the resource and this may be of some use for potential operations. Further surface rights will be required.

ITEM 8: HISTORY

Item 8(a): Prior ownership

Elandsfontein (PTM) , Onderstepoort (Portions 4, 5 and 6) , Onderstepoort (Portions 3 and 8) and *Onderstepoort (Portions 14 and 15)* were all privately owned. Previous work done on these properties has not been fully researched and is largely unpublished. Such academic work as has been done by the Council for Geoscience (government agency) is generally not of an economic nature.

Elandsfontein (RPM) , Frischgewaagd , Onderstepoort (RPM) and *Koedoesfontein* have generally been in the hands of major mining groups resident in the Republic of South Africa. Portions of Frischgewaagd previously held by Impala Platinum Mines Limited were acquired by Johannesburg Consolidated Investment Company Limited, which in turn has since been acquired by AP through RPM.

Item 8(b): Work done by previous owners

Previous geological exploration and resource estimation assessments were done by Anglo Platinum as the original owner of some of the mineral rights. AP managed the exploration drilling programme for the Elandsfontein and Frischgewaagd borehole series in the area of interest. Geological and sampling logs and an assay database are available.

Prior to the establishment of the WBJV and commencement of drilling for the Pre-feasibility study, PTM had drilled 36 boreholes on the Elandsfontein property, of which the geological and sampling logs and assay databases are available.

Existing gravity and ground magnetic survey data were helpful in the interpretation of the regional and local geological setting of the reefs. A distinct increase in gravity values occurs from the southwest to the northwest, most probably reflecting the thickening of the Bushveld sequence in that direction. Low gravity trends in a southeastern to northwestern direction. The magnetic survey reflects the magnetite-rich Main Zone and some fault displacements and late-stage intrusives in the area.

The previous declarations filed with SEDAR on 13 April 2006 may be accessed on the SEDAR website and specifically by reference to *Independent Preliminary Assessment Scoping Study Report and Resource Update Western Bushveld Joint Venture Elandsfontein Project (Project 1).*

Item 8(c): Historical reserves and resources

Previous reserves and resources quoted for the area are those published in the AP 2004 annual report including 7.8Mt grading 5.88g/t 4E (1.47 million ounces 4E) on the Merensky Reef and 4.8Mt grading 4.52g/t 4E (0.70 million ounces 4E) on the UG2 Reef. This is reported for AP's 37% interest (equal to PTM's as the WBJV was completed at that time). In terms of a 100% interest in the property the estimate would be 21.08Mt grading 5.88 g/t 4E (3.99 million ounces 4E) on the Merensky Reef and 13.00Mt grading 4.52g/t 4E (1.89 million ounces 4E) on the UG2 Reef. The resources of AP as reported are subject to a satisfactory independent audit. The prill splits for these estimates are not available but the estimates are seen as relevant, reliable and in compliance with SAMREC reporting best practice.

An independent expert subsequently provided an updated estimated Inferred Resource of 15.41Mt grading 7.92g/t 4E (3.93 million ounces 4E) on the Merensky Reef and 10.05Mt grading 2.52g/t 4E (0.82 million ounces 4E) on the UG2 Reef, as announced in the news release dated 7 March 2005 (SEDAR-filed 22 April 2005).

PTM then announced on 12 December 2005 (SEDAR-filed 13 January 2006) an estimated Indicated Resource of 6.92Mt grading 5.89g/t 4E (1.31 million ounces 4E) and an Inferred Resource of 20.28Mt grading 5.98g/t 4E (3.90 million ounces 4E).

On 2 March 2006 an increase in Indicated Resource to 20.45Mt grading 3.91g/t 4E (2.57 million ounces 4E) and in Inferred Resource to 30.99Mt grading 5.16g/t 4E (5.14 million ounces 4E) was published (SEDAR-filed 13 April 2006).

All of the SEDAR-filed communications listed above are in accordance with SAMREC categories and were reliable at the time of the estimate.

Item 8(d): Production from the property

There has been no previous production from any of the WBJV properties.

ITEM 9: GEOLOGICAL SETTING

Regional geology

The stable Kaapvaal and Zimbabwe Cratons in southern Africa are characterised by the presence of large mafic-ultramafic layered complexes. These include the Great Dyke of Zimbabwe, the Molopo Farms Complex in Botswana and the well-known BIC. The BIC was intruded about 2,060 million years ago into rocks of the Transvaal Supergroup along an unconformity between the Magaliesberg quartzites (Pretoria Group) and the overlying Rooiberg felsites (a dominantly felsic volcanic precursor). The BIC is by far the most economically important of these deposits as well as the largest in terms of preserved lateral extent, covering an area of over 66,000km ^{2}. It has a maximum thickness of 8km, and is matched in size only by the Windimurra intrusion in Western Australia and the Stillwater intrusion in the USA (Cawthorn, 1996).

The mafic component of the Complex hosts layers rich in PGEs, nickel, copper, chromium and vanadium. The BIC is reported to contain about 75% and 50% of the world's platinum and palladium resources respectively (Vermaak, 1995). The mafic component of the BIC is subdivided into several generally arcuate segments/limbs, each associated with a pronounced gravity anomaly. These include the western, eastern, northern/Potgietersrus, far western/Nietverdient and southeastern/Bethal limbs.

The mafic rocks are collectively termed the *Rustenburg Layered Suite (RLS)* and are subdivided into the following five zones:

- *Marginal Zone* comprising finer-grained gabbroic rocks with abundant country-rock xenoliths.
- *Lower Zone* – the overlying Lower Zone is dominated by orthopyroxenite with associated olivine-rich cumulates (harzburgite, dunite).
- *Critical Zone* – its commencement is marked by first appearance of well-defined cumulus chromitite layers. Seven Lower Group chromitite layers have been identified within the lower Critical Zone. Two further chromitite layers – Middle Group (MG) – mark the top of the pyroxenite-dominated lower Critical Zone. From this stratigraphic position upwards, plagioclase becomes the dominant cumulus phase and noritic rocks predominate. The MG3 and MG4 chromitite layers occur at the base of the upper Critical Zone, which is characterised from here upwards by a number of cyclical units. The cycles commence in general with narrow pyroxenitic horizons (with or without olivine and chromitite layers); these invariably pass up into norites, which in turn pass into leuconorites and anorthosites. The UG1 – first of the two Upper Group chromitite layers – is a cyclical unit consisting of chromitite layers with overlying footwall units that are supported by an underlying anorthosite. The overlying UG2 chromitite layer is of considerable importance because of its economic concentrations of PGEs. The two uppermost cycles of the Critical Zone include the Merensky and Bastard cycles. The Merensky Reef (MR) is found at the base of the Merensky cycle, which consists of a pyroxenite and pegmatoidal feldspathic pyroxenite assemblage with associated thin chromitite

layers that rarely exceed one metre in thickness. The top contact of the Critical Zone is defined by a giant mottled anorthosite that forms the top of the Bastard cyclic unit.

- *Main Zone* – consists of norites grading upwards into gabbronorites. It includes several mottled anorthosite units towards the base and a distinctive pyroxenite, the Pyroxenite Marker, two thirds of the way up. This marker-unit does not occur in the project area, but is evident in the adjacent BRPM. The middle to upper part of the Main Zone is very resistant to erosion and gives rise to distinctive hills, which are currently being mined for dimension stone (black granite).
- *Upper Zone* – the base is defined by the appearance of cumulus magnetite above the Pyroxenite Marker. The Upper Zone is divided into Subzone A at the base; Subzone B, where cumulus iron-rich olivine appears; and Subzone C, where apatite appears as an additional cumulus phase.

Local geology

Exposures of the BIC located on the western limb include the stratigraphic units of the RLS. The sequence comprises mostly gabbros, norites, anorthosites and pyroxenites. There are two potentially economically viable platinum-bearing horizons in this area: (1) the Merensky Reef – occurring as either a pegmatoidal feldspathic pyroxenite, a harzburgite, or a coarse-grained pyroxenite – and (2) the UG2 Reef as a chromite seam/s.

The Merensky Reef subcrops, as does the UG2 Reef, beneath a relatively thick (± 2–5m) overburden of red Hutton to darker Swartland soil forms. The sequence strikes northwest to southeast and dips at between 4 and 42 degrees with an average of 14 degrees (in this area specifically). The top 32m of rock formation below the soil column is characterised by a highly weathered rock profile (regolith) consisting mostly of gabbro within the Main Zone succession. Thickness of this profile increases near intrusive dykes traversing the area, suggesting possible targets for well drilling.

The sequence of the BIC within the WBJV area is confined to the lower part of the Main Zone (Porphyritic Gabbro Marker) and the Critical Zone (HW5–1 and Bastard Reef to UG1 footwall sequence). The rock sequence thins towards the southwest (subcrop) including the marker horizons with concomitant middling of the economic reefs or total elimination thereof. The UG2 Reef and, more often, the UG1 Reef are not developed in some areas owing to the irregular and elevated palaeo-floor of the Transvaal sediments.

Stratigraphy

The detailed stratigraphy of the western BIC is depicted in Diagram 4. The identifiable units within the WBJV area are, from top to bottom:

1. the base of the noritic Main Zone
2. the anorthositic hanging wall sequence (HW5–1)
3. the Bastard Reef pyroxenite
4. the Mid3–1 units
5. the Merensky Reef pyroxenite
6. FW1–5
7. the anorthositic footwall (FW6–12)
8. the UG2 unit
9. the underlying medium-grained norite (FW13)
10. the multiple UG1 chromitite seams
11. the underlaying medium-grained mottled anorthositic FW16
12. the Transvaal basement sediments.

Drilling below the UG1 indicated the general absence of the basal-chilled alteration zone in contact with the Transvaal Supergroup sediments in the Project 1 area.

The Main and Critical Zone sequences of the BIC as seen in the WBJV boreholes (see Tables 1(a) and 1(b) and Diagram 5) consist of norites and gabbronorites within the Main Zone (less than 60m thick) at the top of the sequence. Spotted and mottled anorthositic hanging wall units (HW5–1) are less than 20m thick close to subcrop and less than 130m thick away from the subcrop; these overlie the Bastard pyroxenite (less than 2m thick), which is followed by norite to mottled anorthosite. The Mid3–1 units (ranging in thickness from 6–30m from shallow to deeper environments) overlie the Merensky Reef pyroxenite (less than 2m thick). The Merensky Reef varies at this point from pegmatoidal feldspathic pyroxenite less than 10cm thick and/or a millimetre-thick chromitite layer, a contact only, to a thicker (more than 100cm) type of reef consisting of harzburgite and/or pegmatoidal pyroxenite units. Some of the norite footwall units (FW1–5) at the immediate footwall of the reef are not always developed and the total noritic footwall sequence is much thinner (less than 13m) than at the adjacent BRPM operation. The mottled anorthosite footwall unit (FW6) has a chromitite layer (Lone-chrome) which, although mere millimetres thick within the pegmatoidal feldspathic pyroxenite-reef-type area, is generally developed in this area and constitutes a critical marker horizon. Footwall units FW7–11 (mostly norite) are also not always developed and are much thinner (less than 25m) than at BRPM. The mottled anorthosite footwall unit, FW12, is generally well developed (less than 2m) and overlies a very thin UG2 chromitite/pyroxenite reef in the southern part of the property. The UG2 chromitite layer is in most cases disrupted and is either very thin or occurs as a pyroxenite in this area of the WBJV project. Further

northeast towards the Frischgewaagd area, the UG2 Reef seems to thicken, especially in geological environments where the palaeo-floor to the Bushveld Complex tends to have lower slope gradients.

Thickening of the stratigraphic units as described above, trends more or less from the southwest to the northeast. This may have resulted from a general thickening of the entire BIC towards the central part of the Complex, away from the steeper near-surface contact with the Transvaal Supergroup. Some localities were identified in the central part of the WBJV project area, where thinning of lithologies is may be due to palaeo-high environments within the footwall below the BIC.

Correlation and lateral continuity of the reefs

The lower noritic portion of the Main Zone could be identified and correlated with a high degree of confidence. A transgressive contact exists between the Main Zone and the anorthositic hanging wall sequence. The HW5–1 sequence is taken as a marker horizon; it thins out significantly from northeast to southwest across and along the dip direction. Because of thinning of the Critical Zone, only the primary mineralised reefs (Merensky and UG2), the Bastard Reef, Merensky pyroxenite above the Merensky Reef, FW6 and FW12 have been positively identified. The sequence was affected by iron-replacement, especially the pyroxenites towards the western part of the property. Evidence of iron-replacement also occurs along lithological boundaries within the Main Zone and the HW5 environment of the Critical Zone and in a down-dip direction towards the deeper sections of the property.

The Merensky Reef and UG2 Reef are positively identified in new intersections. The intersection depths are summarised in Table 1(a) and 1(b) – Appendix A. Only the reef intersections that had no faulting or disruptions/discontinuities were used in the resource estimate. The UG1, traditionally classified as a secondary reef typically with multiple chromitite seams, has been intersected in some boreholes; although in many cases strongly disrupted, it showed surprisingly attractive grades.

Resource estimation is not possible within 50m from surface owing to core loss resulting from near-surface weathering (weathered rock profile), joint set interference, reef identification/correlation problems and thinning of the reefs towards the west.

Merensky Reef is poorly developed in the Elandsfontein property area, from the subcrop position to as far as 100m down-dip and as far as 800m along strike. This was evident in marginal grades, and is no doubt due to the presence of a palaeo-high in the Transvaal sediment floor rocks below the BIC. The area is locally referred to as the Abutment.

With respect to the UG2 Reef in the project area, relative to the Abutment's effect, a smaller area extending from subcrop position to as deep as 400m down-dip with strike length 420m of UG2 Reef was characterised by a relatively low grade.

Structural discontinuities

Viljoen (1999) originally proposed a structural interpretation based on geological and geophysical data for the western lobe of the BIC. This study included gravity and vibroseis seismic data for the southwestern portion of the RLS northwest of Rustenburg (including the Boshoek section). It was concluded that the Merensky Reef is present within much of this lobe, including the part further to the east below the Nebo granite sheet. The position of the Merensky Reef is fairly closely defined by seismic reflectors associated with the cyclic units of the upper Critical Zone. The seismic data also portrayed an essentially sub-horizontal disposition of the layering within the BIC mafic rocks below the Nebo granite sheet.

The gravity data indicates a gravity-high axis extending throughout the western lobe following the upper contact of the mafic rocks with the overlying granitic rocks. A number of pronounced gravity highs occur on this axis. A gravity anomaly with a strike length of 9km is situated northeast of Rustenburg towards the east of the Boshoek section. The gravity highs have been interpreted as representing a thickening of the mafic rocks, reflecting feeder sites for the mafic magma of the western BIC (Viljoen, 1999).

The western lobe is interpreted by Viljoen as having two main arcuate feeder dykes which closer to surface have given rise to arcuate, coalescing, boat-shaped keels containing saucer-shaped, inward-dipping layers, analogous to the Great Dyke of Zimbabwe.

In the Boshoek section north of Rustenburg, the variable palaeo-topography of the Bushveld floor represented by the Transvaal Supergroup contact forms a natural unconformity with the overlying Bushveld layered sequence. Discontinuities due to structural interference of faults, sills and dykes are pronounced in the area and are ascribed to the presence of the Pilanesberg Alkaline Complex intrusion to the north of the property. The possibility exists that pothole edges may be associated with the Contact Reef. Duplicated reef intersections (isolated cases) could also represent pothole edge effects (goose-necking). Furthermore, pseudo-reefs along the pothole edges associated with goose-necking may be interpreted within the project area as evidence of potholes.

Faulting

A structural model was developed from data provided by the magnetic survey results and geological logs of drilled cores. At least three generations of faults were identified on the property with the dominant structures oriented at 345 degrees and 315 degrees north respectively. The first fault set appears to be the most

prominent, with the largest displacement component of more than 20m. The majority of the faults are normal faults dipping in a westerly direction, decreasing in their dip downwards and displaying typical listric fault system behaviour.

Dykes and sills

Several dolerite intrusives, mainly steep-dipping dykes and bedding-parallel sills, were intersected in boreholes. These range in thickness from 0.5–30m and most appear to be of a chilled nature; some are associated with faulted contacts. Evident on the magnetic image is an east-west-trending dyke which was intersected in borehole WBJV005 and appears to be of Pilanesberg-intrusion age. This dyke has a buffer effect on structural continuity as faulting and earlier stage intrusives are difficult to correlate on either side; and more work is required to understand the mechanics.

Shear zones

A shear zone has been identified along the footwall contact Alteration Zone. This structure appears to be confined to the extreme southern section of the Elandsfontein shallow reef environment and progressively eliminates stratigraphy from the UG2 horizon to the Main Zone from east to west. The elimination effect of the shear zone is confined to the first 200m from surface.

Replacement pegmatites

Pseudo-form replacement bodies exist within the Elandsfontein property and seem to be concentrated in the lower part of the Main Zone and HW5 of the Critical Zone. Reef packages to the south in the Elandsfontein (PTM) area are marginally affected (Siepker and Muller, 2004). This should be taken into consideration in the resource estimation and geological loss figures for both Merensky and UG2 reefs. Because of the pseudo-form nature of these bodies, it is difficult to assess their interference with the reef horizon are difficult.

Depth of oxidation and overburden

Evidence from boreholes to date shows that the regolith thickness in the WBJV area varies from 21–32m (it is for this reason that all boreholes are cased up to a depth of at least 40m). The depth of oxidation coincides with depth of weathering and affects the reef horizons along the subcrop environment and along the 1,015 AMSL reef contour line.

Geological and rock-engineering-related losses

The resource model allows for a geological loss of 19% due to faults (8%), dykes (4%) and iron-replacement (7%). Refer to Tale 12 – Appendix A.

Structural model

A structural model was deduced from geophysical information and borehole intersections. In general, three phases of deformation are recognised in the area. The oldest event appears to be associated with dykes and sills trending at 305 degrees and is of post-BIC age. A second phase represented by younger fault features is trending in two directions at 345 degrees and 315 degrees northwards respectively and appears to have consistent down-throws towards the west. A third and final phase of deformation may be related to a regional east-west-striking dyke system causing discontinuity on adjacent structures.

Site-specific geology

The general stratigraphy of the upper Critical Zone proximal to the primary economic reefs – Merensky and UG2 – is outlined as follows (see Diagram 4):

- Most of the boreholes drilled on the property have collared in the lower part of the ***Main Zone (MZN)*** sequence and typically in gabbronorites. The thickness of these gabbronorites and in particular the Porphyritic Gabbro Marker seems to increase from 10m in the southeast to 80m in the northwest. In this marker-unit, pyroxene porphyries tend to increase in size towards the base. At least three mottled anorthositic units (poikilitic lithological phases) were intersected in boreholes within the MZN norites below the Porphyritic Gabbro Marker with thicknesses of between 4 and 25 metres.
- The contact at the base of the MZN cycle is transgressional towards the underlying ***HW5*** cycles, which are medium mottled to spotted anorthosite, to large mottled anorthosite. No known mineralisation occurs in these units.
- The gradational contact between HW5 and ***HW4*** transgresses from mottled anorthosite into a vari-textured (leopard-spotted) anorthosite.
- The *HW4–3* interface is a transitional contact. The ***HW3*** is typically a large mottled anorthosite with no apparent mineralisation and is an easily recognisable marker horizon.
- The ***HW2*** unit is classified as a cycle of leuconorite, norite and medium-grained pyroxenites. The ***HW1*** pyroxenitic norite is normally relatively thin in the project area, usually measuring no more than 0.3m.
- The ***Bastard pyroxenite (Bpyx)*** commonly underlies the HW1 unit with a transitional contact. Pyroxenes and feldspars are commonly medium-grained; sulphide-accumulation occurs towards the bottom of the unit.
- The ***Bastard Reef (BR)*** is characterised by a coarse-grained pyroxenite. The unit is relatively thin and sulphide-enriched with nominal mineralisation towards the base. If the unit is well-developed, a thin chromitite stringer occurs at the base with generally no increase in the intensity of mineralisation. This reef horizon is currently not perceived as an economically viable unit in the project area.

- The ***Mid3*** lithological unit underlies the Bastard Reef as an abrupt contact and is characterised as a large mottled anorthosite similar to that of HW3. No mineralisation occurs in this unit but it is a defined marker horizon.
- The ***Mid2*** unit is a leuconorite phase in the cycle between Mid3–1 and is normally less than 3m thick.
- The ***Mid1*** norites are usually less than a metre thick and have a gradational contact with the underlying Merensky pyroxenite. The light grey medium-grained norites consist of equal-granular cumulate pyroxenes with intercumulus feldspar. The lithological sequence from the sharp contact below the Bastard Reef to a gradational contact at the base of the Mid1 norite unit varies in thickness from 2–7m.
- The ***Merensky pyroxenite (Mpyx)*** forms the hanging wall of the Merensky Reef and has a thickness ranging from 0.2–5.0m. It consists of cumulate pyroxenes with interstitial feldspar. The subhedral pyroxenes are medium- to coarse-grained and tend to become coarser-grained towards the upper contact with the Merensky Reef Upper Chromitite (MRUCr) stringer. The Merensky pyroxenite contains interstitial sulphide (2–4%) towards the bottom contact and just above the MRUCr. The main sulphides are represented by pyrrhotite and pentlandite with minor pyrite.
- The ***Merensky Reef Upper Chromitite (MRUCr)*** exists as a chromitite stringer roughly 1–10mm thick or as disseminated chromitite lenses. It forms the base of a new cycle of differentiation considered responsible for thermal reconstitution of the underlying pyroxenite which formed the pegmatoidal Merensky Reef. It is this cycle which introduced much of the PGE and base metal sulphide mineralisation of the Merensky Reef (Viljoen, 1999).
- The ***Merensky Reef pegmatoidal feldspathic pyroxenite (FPP)*** ranges in thickness from 0–0.75m and is bounded by the MRUCr and the Merensky Reef Bottom Chromitite (MRBCr). The unit consists of cumulus pegmatoidal pyroxene and intercumulus plagioclase. The plagioclase is an interstitial phase which encloses the orthopyroxene and clinopyroxene in a poikilitic texture. The FPP contains disseminated and cumulate sulphides (3–5%) represented by pyrrhotite, pentlandite and minor pyrite. In the presence of the MRUCr the feldspathic pyroxenite (Mpyx) grades into a well-developed FPP with strong reconstitution of sulphides within the proximal footwall units.
- The ***Merensky Bottom Chromitite (MRBCr)*** ranges in thickness from a couple of millimetres to 0.07 metres. At normal reef elevation, it represents a more or less conformable base of an existing differentiation cycle. Where anorthosite underlies the Merensky Reef, the downward settling of immiscible sulphides was arrested and became concentrated in the narrow pegmatoidal reef. Viljoen (1999) has suggested that this is due to the unreactive nature of the anorthosite. Where norite underlies the bottom chromitite (MRBCr), the thermal front penetrated into the footwall and resulted in the blotchy, thermal reconstitution of the fairly reactive footwall norite or leuconorite.
- The immediate footwall of the Merensky Reef generally consists of norite (FW1, FW2 or FW3) that is often mineralised up to one metre below the Merensky contact. **FW1** and **FW2** is not present in the

project area and has not been intersected by any of the holes drilled to date. The **FW3** unit has a leuconorite – with a poikilitic anorthositic pseudo-form – mottled texture and is an unconformity of the Merensky Reef in the Abutment- and mid terrace regions. The Merensky Reef seems to be overlying unconformably to FW6 anorthosites in the deep terrace regions and no evidence was gained for the existence of **FW4** and **FW5** norites on the project area.

- The **FW6** mottled anorthosite/norite cycle is common in the mid- and deep terrace geo-zones and range in thickness from 1–4m. The lowermost sublayer FW6(d) is a mottled anorthosite. A single chromitite stringer also known as the *Lone-chrome* (2–10mm thick) is present at the base of the FW6(d) sublayer and is a distinct marker with respect to rock recognition. The FW6 (c) is also a mottled anorthosite but seldom developed. FW6(b) is a leuconorite 2–3cm thick. FW6(a), the uppermost sublayer, is a mottled anorthosite. The geotechnical competency of this lithological sequence is perceived as stable with respect to potential tunnelling activities.
- The **FW7** unit is a distinct olivine-rich norite 3–7m thick that occurs as an abrupt underlay to the Lone-chrome of FW6. The texture of this unit is peculiar and unique, with the pyroxenes partly replaced by olivine to provide a corona-texture appearance and a greenish background throughout the unit.
- The **FW8** unit is a leuconorite approximately 1–2m thick with occasional anorthosite sublayering. This unit seems to be eliminated from the succession in the Abutment region of the project area.
- The **FW9** unit is a mottled anorthosite and mostly thin (1–2m thick); it has a rose-pink background colour and is not always developed. It has a gradational contact with the overlying FW8 and transitional contact with the underlying FW10.
- The **FW10** unit is sporadically developed and seems to be more prominent in the deep terrace region. As a porphyritic norite it seldom exceeds 0.7m and normally occurs as a 0.25–0.30m lithological unit. Its dark green appearance is due to the alteration and presence of olivine recrystallisation.
- The **FW11** unit ranges in thickness from 6–10m and is a leuconorite with numerous thin anorthosite banding (1–3cm) and occasional mottled anorthosite bands (approximately 15cm) proximal to the base.
- The **FW12** is a large (3–5cm) poikilitic anorthosite which has an abrupt contact with the underlying feldspathic pyroxenite that overlies the UG2 and varies in thickness between five and eight metres.
- An **upper feldspathic pyroxenite** about 3m thick above the UG2 main seam contains cumulate pyroxene and intercumulus feldspar and hosts three Leader seams of variable thickness. These seams are generally situated 0.2–3m above the UG2 main seam. These three Leaders are not always present and Leaders 2 (UG2L2) and 3 (UG2L3) seem to be vacant on the slope environments where the Transvaal Basement is elevated. The thickness of the Leaders has been logged as 10–20cm for UG2L1 and UG2L2. The UG2L3 is normally a chromitite seam a few millimetres thick.

- The lower contact between the **UG2 main seam** and the underlying feldspathic unit is usually irregular. Poikilitic bronzite crystals give the UG2 chromitite seam a spotted appearance and appear to be confined to the main seam. This unit is often massive chromitite but in places occurs as numerous seams due to the presence of interstitial pyroxenite. The thickness of the UG2 layer seems to increase in depth from the subcrop, starting as a very disrupted thin seam (5–20cm) on the Abutment environment and becoming a pronounced thick deposit (more than 2m) in the deeper eastern section at the WBJV boundary.
- The **UG2 Reef** is characterised by 0.3m-thick coarse-grained lower feldspathic pyroxenite developed below its base. A 5–30cm harzburgitic leuconorite unit is developed below the feldspathic pyroxenite, mostly at the property's mid- and deep terrace regions. If pegmatoidal, the feldspathic pyroxenite contains disseminated chromite and chromitite stringers. An abrupt contact normally occurs between the harzburgite unit and the underlying FW13 norites.
- The **FW13** unit is a cyclic sublayered unit of leuconorite and spotted anorthosite, and varies in thickness from 2 metres in the Abutment region to 30 metres where layering has low slope gradients.
- The **UG1** (lower-set units of the Upper Group chromitite layers) is a cyclical unit consisting of three to five thin chromitite layers of varying (1–25cm) thickness. Intermittent norite may be associated with this unit. Mineralisation is confined to the chromitite seams and this unit may be considered a potential target. Placement of footwall development in the mine design for exploitation of the UG2 will take cognisance of the presence of the UG1 below.
- The **FW16** underlies the UG1 chromitite cycle as a sequence of mottled anorthosite grading to a leuconorite towards the base and has an irregular, sharp contact with the Transvaal sediments.

ITEM 10: DEPOSIT TYPE

The project area is situated on the western limb of the BIC. PGM mineralisation is hosted within the Merensky Reef and the UG2 Reef located within the upper Critical Zone of the RLS of the BIC. The property adjoins BRPM to the southeast, which is currently mining the Merensky Reef. The geology of BRPM is relatively well understood and is regarded in certain aspects as representative of the WBJV area.

The Merensky Reef is a well-developed seam along the central part and towards the northeastern boundary of the property. Islands of thin reefs and relatively low-level mineralisation are present. The better-developed reef package, in which the intensity of chromitite is generally combined with pegmatoidal feldspathic pyroxenite development, occurs as larger island domains along a wide central strip in a north-south orientation from subcrop to the deeper portions.

The UG2 Reef is well developed towards the northeast of the Elandsfontein project area but deteriorates towards the southwest. Within the latter area, the reef is present as a thin discontinuous or disrupted chromitite/pyroxenite layer; it also appears to be disrupted by the shear zone along the footwall alteration zone. Towards the northwest on Frischgewaagd the reef is generally well-developed and occurs as a single prominent chromitite layer varying in thickness from a few centimetres to roughly two metres.

The isopach thickness between the Merensky and the UG2 Reefs in the Project 1 area increases from approximately 10 metres to as much as 80 metres in a southwest-northeast direction. The dip of the reefs is shallow near the surface but the angle increases in places to 42 degrees; the average dip is 14 degrees. A similar situation exists in the north of the project area but with isopach thicknesses ranging from 6–25m at depths of 200m below surface. In general, the isopach thicknesses appear to increase in a northeasterly direction sub-parallel to the strike of the BIC layered lithologies.

Geological model: Boshoek section of the Western Bushveld Complex (from Schürmann, 1993)

The Boshoek Section is located in the mafic part of the southwestern BIC. It lies between the Magaliesberg Formation quartzites in the south and west, the gabbro of the Upper Zone in the east and the Pilanesberg Alkaline Complex in the north. The BRPM lease area is situated in the Boshoek section north of Rustenburg.

Rocks of the Rustenburg Layered Suite are poorly exposed in the southwestern BIC. The RLS is subdivided into the Boshoek, Rustenburg and Marikana sections by marked undulations within the sedimentary floor rocks. These undulations appear to be responsible for lateral variations in thickness of the different units of the lower Critical Zone. In the Boshoek section, only the Marginal, Critical and Main Zones are developed within the RLS. The lower Critical Zone is conformable and above the Marginal Zone lithological sequence.

The Marginal Zone is mainly represented by norite, the lower Critical Zone by harzburgite and pyroxenite, the upper Critical Zone by anorthosite, norite and porphyritic pyroxenites, the Main Zone by gabbros and the Upper Zone by ferrogabbro.

Leeb-Du Toit (1986) described the succession from the UG1 to the top of the Bastard Reef in the Impala Lease area and introduced a model whereby characteristic rock layers are numbered sequentially from the Merensky Reef footwall layers downwards and upwards from the Reef hanging wall layers.

Structure

Floor rocks in the southwestern BIC display increasingly varied degrees of deformation towards the contact with the RLS. Structure within the floor rocks is dominated by the north-northwest-trending post-Bushveld Rustenburg fault. This normal fault with down-throw to the east extends northwards towards the west of the

Pilanesberg Alkaline Complex. A second set of smaller faults and joints, striking 70 degrees and dipping very steeply south-southeast or north-northwest, is related to the Rustenburg fault system. These structures were reactivated during intrusion of the Pilanesberg Alkaline Complex. Dykes associated with this Complex intruded along these faults and joints.

Two stages of folding have been recognised within the area. The earliest folds are mainly confined to the Magaliesberg Quartzite Formation. The fold axes are parallel to the contact between the RLS and the Magaliesberg Formation. Quartzite xenoliths are present close to the contact with the RLS and the sedimentary floor. Examples of folding within the floor rocks are the Boekenhoutfontein, Rietvlei and Olifantsnek anticlines. The folding was initiated by compressional stresses generated by isostatic subsidence of the Transvaal Supergroup during sedimentation and the emplacement of the pre-Bushveld sills.

The presence of an undulating contact between the floor rocks and the RLS, and in this instance the resultant formation of large-scale folds, substantiates a second stage of deformation. The fold axes trend at approximately orthogonal angles to the first folding event. Deformation during emplacement of the BIC was largely ductile and led to the formation of basins by sagging and folding of the floor rocks. This exerted a strong influence on the subsequent evolution of the Lower and Critical Zones and associated chromitite layers.

The structural events that influenced the floor rocks played a major role during emplacement of the BIC. There is a distinct thinning of rocks from east to west as the BIC onlaps onto the Transvaal floor rocks, even to the extent that some of the normal stratigraphic units have been eliminated. The Merensky and UG2 isopach decreases from 60m to 2m at subcrop position as clearly illustrated by the section in Diagram 6. There is also a subcrop of the Critical Zone against the main zone rocks.

Stratigraphy of the upper Critical Zone

The upper Critical Zone of the RLS comprises mostly norites, leuconorites and anorthosites. Leeb-Du Toit (1986) assigned numbers to the various lithological units according to their position in relation to the Merensky unit. The footwall layers range from FW14 below the UG1 chromitite to FW1 directly below the Merensky Reef. The hanging wall layers are those above the Bastard Reef and range from HW1 to HW5. The different layers within the Merensky unit are the Merensky feldspathic pyroxenite at the base, followed by a leuconorite (Middling 2) and a mottled anorthosite (Middling 3). The feldspathic pyroxenite layers (pyroxene cumulates) are named according to the reef hosted by them. These include (from the base upwards) the UG1, the UG2 (upper and lower), the Merensky and the Bastard pyroxenite.

Schürmann (1993) subdivided the upper Critical Zone in the Boshoek section into six units based on lithological features and geochemical trends. These are the Bastard, the Merensky, the Merensky footwall, the Intermediate, the UG2 and the UG1 units. The Intermediate and Merensky footwall units were further

subdivided based on modal-mineral proportions and whole-rock geochemical trends. The following is a detailed description of the subdivision of the upper Critical Zone in the Boshoek section (from Schürmann, 1993):

Bastard unit

The Bastard unit consists of a basal pyroxenite some 3m thick with a thin chromitite developed on the lower contact. This chromitite is the uppermost chromitite layer in the Critical Zone. A 6.5m-thick norite layer (HW1) overlies the pyroxenite. HW1 is separated from HW2 by two thin mottled anorthosite layers. HW3 is a 10m-thick mottled anorthosite and constitutes the base of the Giant Mottled Anorthosite. The mottled anorthosites of HW4 and HW5 are about 2m and 37m thick respectively. Distinction between HW3, 4 and 5 is based on the size of the mottles of the respective layers.

Merensky unit

The Merensky unit, with the Merensky Reef at its base, is the most consistent unit within the Critical Zone (see Item 9).

Merensky footwall unit

This unit contains the succession between the FW7/FW6 and the FW1/MR contacts. Leeb-Du Toit (1986) indicated that where the FW6 layer is thicker than 3m, it usually consists of four well-defined rock types. The lowermost sublayer, FW6(d), is a mottled anorthosite with mottles of between 30mm and 40mm in diameter. It is characterised by the presence of nodules or “boulders” and is commonly referred to as the Boulder Bed. The nodules are described as muffin-shaped, 5–25cm in diameter, with convex lower contacts and consisting of cumulus olivine and orthopyroxene with intercumulus plagioclase. A single 2–10mm chromitite stringer is present at the base of the FW6(d) sublayer. FW6(c) is also a mottled anorthosite but not always developed. FW6(b) is a leuconorite containing pyroxene oikocrysts 10–20mm in diameter. Two layers (both 2–3cm thick) consisting of fine-grained orthopyroxene and minor olivine define the upper and lower contacts. FW6(a), the uppermost sublayer, is also a mottled anorthosite.

FW6 is overlain by a uniform norite (FW5), with a thickness of 4.1m. It appears to thin towards the north to about one metre. FW4 is a mottled anorthosite 40cm thick, with distinct layering at its base. FW3 is an 11m-thick uniform leuconorite. FW2 is subdivided into three sublayers. FW2(b) is a 76cm-thick leuconorite and is overlain by a 33cm-thick layer of mottled anorthosite – FW2(a). Where FW2 attains a maximum thickness of 2m, a third layer in the form of a 1–2cm-thick pyroxenite or pegmatitic pyroxenite, FW2(c), is developed at the base. FW2(c) is absent in the Boshoek section area (Schürmann, 1993). FW1 is a norite layer about 7m thick. Schürmann further subdivided the Merensky footwall unit into four subunits. The lowermost subunit consists of sublayers FW6(d) and FW6(b). Subunit 2, which overlies subunit 1, commences with FW6(a) at the base and grades upwards into FW5. The FW5/FW4 contact is sharp and divides subunits 2 and 3. Subunit

3 consists of FW4, FW3 and sublayer FW2(b). Subunit 4 consists of FW2(a) and FW1 and forms the uppermost subunit of the Merensky footwall unit.

Intermediate unit

The Intermediate unit overlies the upper pyroxenite of the UG2 unit and extends to the FW7/FW6 contact. The lowermost unit is the 10m-thick mottled anorthosite of FW12 which overlies the UG2 upper pyroxenite with a sharp contact. FW11, a roughly one-metre-thick leuconorite, has gradational contacts with the under- and overlying layers. FW10 consists of a leuconorite layer of about 10m. Subdivision between these two units is based on the texture and subtle differences in the modal composition of the individual layers. Leeb-Du Toit (1986) termed FW11 a spotted anorthosite and FW10 an anorthositic norite. FW12, 11 and 10 constitute the first Intermediate subunit as identified by Schürmann (1993).

The second Intermediate subunit consists of FW9, 8 and 7. The 2m-thick FW9 mottled anorthosite overlies the FW10 leuconorite with a sharp contact. The FW8 leuconorite and FW7 norite are respectively 3m and 37m thick. The FW9/FW8 and FW8/FW7 contacts are gradational but distinct. A 1.5m-thick highly contorted mottled anorthosite “flame bed” is present 15m above the FW8/FW7 contact.

UG2 unit

The UG2 unit commences with a feldspathic pyroxenite (about 4m thick) at its base and is overlain by an orthopyroxene pegmatoidal layer (0.2–2m thick) with a sharp contact. Disseminated chromite and chromitite stringers are present within the pegmatoid. This unit in turn is overlain by the UG2 chromitite (0.5–0.8m thick) on an irregular contact. Poikilitic bronzite grains give the chromitite layer a spotted appearance. A 9m feldspathic pyroxenite overlies the UG2 chromitite. The upper and lower UG2 pyroxenites have sharp contacts with FW12 and FW13. The upper UG2 pyroxenite hosts the UG2 Leader seams, which occur between 0.2m and 3m above the main UG2 chromitite.

UG1 unit

The UG1 chromitite layer is approximately one metre thick and forms the base of this unit. It is underlain by the 10m-thick FW14 mottled anorthosite. The UG1 chromitite layer bifurcates and forms two or more layers within the footwall mottled anorthosite, while lenses of anorthosite also occur within the chromitite layers. The overlying pyroxenite consists of cumulus orthopyroxene, oikocrysts of clinopyroxene and intercumulus plagioclase. The UG1 pyroxenite is separated from the overlying FW13 leuconorite (about 8m thick) by a thin chromitite layer (1–10cm) with sharp top and bottom contacts.

ITEM 11: MINERALISATION

Mineralisation styles and distribution

Merensky Reef

The most pronounced PGM mineralisation along the western limb of the BIC occurs within the Merensky Reef and is generally associated with a 0.1–1.2m-thick pegmatoidal feldspathic pyroxenite unit. The Merensky Reef is generally also associated with thin chromitite layers on either/both the top and bottom contacts of the pegmatoidal feldspathic pyroxentite. The second important mineralised unit is the UG2 chromitite layer, which is on average 0.6–2.0m thick and occurs within the project area (Elandsfontein and Frischgewaagd).

The Merensky Reef at the adjacent BRPM mining operation consists of different reef types (or facies types) described as either contact-, pyroxenite-, pegmatoidal pyroxenite- or harzburgite-type reef. Some of these facies are also recognised on WBJV project areas.

From logging and sampling information of holes on the WBJV property it is evident that the footwall mineralisation of Merensky Reef below the main chromitite layer occurs in reconstituted norite, which is the result of a high thermal gradient at the base of the mineralising Merensky cyclic unit. The upper chromitite seam may form an upper thermal unconformity. Footwall control with respect to mineralisation is in many cases more dominant than the actual facies (e.g. the presence of leucocratic footwall units) or a chromitite (often with some pegmatoidal pyroxenite).

Within the project area, the emplacement of the Merensky Reef is firstly controlled by the presence or absence of chromitite seams and secondly by footwall stratigraphic units. The Merensky Reef may be present immediately above either the FW3 or FW6 unit. This has given rise to the terms Abutment terrace (FW3 thermal erosional level), mid terrace (FW3 or FW6 thermal erosional levels) and deep terrace (FW6 thermal erosional level).

Within, and not necessarily confined to, each of the terraces, the morphology of the Merensky Reef can change. Merensky Reef has been classified as Type A, Type B, Type C or Type D (see Diagram 7) according to certain characteristics:

Type A Merensky Reef facies relates to the interface between the normal hanging wall of the Merensky Reef and the footwall of the Merensky Reef. There is no obvious chromite contact or any development of the normal pegmatoidal feldspathic pyroxenite. This may well be classified as hanging wall on footwall, but normally has a PGM value within the pyroxenite.

Type B Merensky Reef facies is typified by the presence of a chromite seam which separates the hanging wall pyroxenite from the footwall which could be the FW3 or FW6 unit.

Type C Merensky Reef facies can be found on any of the three terraces and has a characteristic top chromite seam overlying a pegmatoidal feldspathic pyroxenite. This facies has NO bottom chromite seam.

Type D Merensky Reef facies is traditionally known throughout the BIC as Normal Merensky Reef and has top and bottom chromite seams straddling the pegmatoidal feldspathic pyroxenite.

UG2 Reef

The facies model for the UG2 Reef has been developed mainly from borehole exposure data in the northeast of the property. The integrity of the UG2 deteriorates towards the southwest of the project area, where it occurs as a thin chromite layer and/or pyroxenitic unit. It is thus unsuitable for the development of a reliable geological facies model.

In the northeast of the project area the UG2 is relatively well-developed and usually has three thin chromite seams (Leaders) developed above the main seam.

The UG2 Reef facies can also be explained in terms of four distinct facies types (see Diagram 8). Several factors appear to control the development of the UG2 package. Of these the digital terrain model (DTM) of the Transvaal Basement is likely to have the most significant impact. The distinct variance in the various facies is seen as directly related to the increasing isopach distance between the UG2 and Merensky Reef. In this regard, the facies-types for the UG2 have been subdivided into the Abutment terrace facies, mid-slope terrace facies and the deep-slope terrace facies. They are described as follows:

- The Abutment terrace facies was identified in the area where the basement floor was elevated, perhaps as a result of footwall upliftment or an original palaeo-high. In this area it appears that there was insufficient remaining volume for the crystallisation and mineralisation of PGEs. A reduced lithological sequence and thinning-out of layering is evident in the facies domain/s. In this environment there is an irregular and relatively thin (5–20cm) UG2 main seam developed with no evidence suggesting the presence of harzburgitic footwall. No Leaders are present and there is a distinct absence of the normal overlying FW8–12 sequence.
- The intermediate area between the Abutment terrace facies and the mid-slope terrace facies has no UG2 development. The footwall is usually a thin feldspathic pyroxenite transgressing downwards to a medium-grained FW13 norite. The hanging wall generally occurs as either/both the FW7 and FW8 norites.

- The mid- and deep-slope terrace facies environments that form the central and northern boundaries of the project area are characterised by a thicker to well-developed UG2 main seam of about 0.5 metres to more than three metres respectively. Here, as with the Abutment terrace facies, the development of a robust UG2 is dependent on the Merensky/UG2 isopach. This facies is characterised by the fact that all Leaders are exposed at all times and Leader 3 (UG2L3) occurs as a pencil-line chromite seam. A prominent development of a harzburgite FW unit (5–30cm) is often present in this facies type.

ITEM 12: EXPLORATION

Item 12(a): Survey (field observation) results, procedures and parameters

Fieldwork in the form of soil sampling and surface mapping was initially done on the farm Onderstepoort, where various aspects of the lower Critical Zone, intrusive ultramafic bodies and structural features were identified. Efforts were later extended southwards to the farms Frischgewaagd and Elandsfontein. The above work contributed directly to the economic feasibility of the overall project, directing the main focus in the project area towards delineation of the subcrop position of the actual Merensky and UG2 economic reef horizons.

Geophysical information obtained from AP was very useful during the identification and extrapolation of major structural features as well as the lithological layering of the BIC. The aeromagnetic data alone made it possible to delineate magnetic units in the Main Zone, to recognise the strata strike and to identify the dykes and iron-replacements

BW Green was contracted to do ground geophysical measurements. Ground gravity measurements of 120.2km have been completed on 500m line spacing perpendicular to the strike across the deposit, together with 65.5km magnetic. The ground gravity data played a significant role in determining the hinge line where the BIC rocks start thickening down-dip, and this raised the possibility of more economic mineralisation. At the same time the data shows where the Transvaal footwall causes the abutment or onlapping of the BIC rocks. Ground magnetic data helped to highlight faults and dykes as well as to delineate the IRUPs.

Gravity Survey

The objective of the gravity survey was twofold:

1. to determine the structure of the subcropping mafic sheet on the sedimentary floor. This mafic sheet has a positive density contrast of 0.3 gram per cubic centimetre (Smit et al,) with the sediments.
2. to determine the thinning (or abutment) to the west of the mafic rocks on the floor sediments.

The instruments used for this survey are:

1. Gravity meter – *Texas Instruments Worden Prospector Gravity Meter* – This is a temperature-compensated zero length quartz spring relative gravimeter with a claimed resolution of 0.01mgal and an accuracy of 0.05mgal.

2. Position – *Garmin GPS 12, Garmin GPS 72 and Magellan eXplorist 300* – These are 12- (Garmins) and 14-channel (Magellan) hand-held navigation GPSs; all with screens displaying the track, the ability to repeat and average each reading to a required level of accuracy and large internal memories. The GPSs were all set to the UTM projection (zone 35J) and WGS84 coordinate system. The X-Y positional accuracy was well within the specifications of this survey but the Z coordinate accuracy was inadequate.

3. Elevation – *American Paulin System Surveying Micro Altimeter M 1-6* – This is a survey-standard barometric altimeter with a resolution of 30cm commonly used in regional gravity surveys. Although it does not meet the requirements of micro-gravity surveys, it is well up to the requirements of this survey.

Field Procedure

The survey was completed in two phases – a reconnaissance survey followed by a second detailed phase completed in four steps. The initial phase consisted of a gravity survey along the major public roads of the project area. All kilometre posts (as erected by the Roads department) were tied in as base stations through multiple loops to a principal base station. Readings were taken at 100m-intervals between the base stations, re-occupying the stations at less than hourly intervals. The instrument was only removed from its padded transport case for readings. The readings were taken on the standard gravimeter base plate and then used to determine the positions.

At each station the gravimeter was read, the GPS X-Y position was taken until the claimed error was less than 5m and then stored along with the time on the instrument (All three GPSs were used alternately during the survey with a short period of overlap to check for instrument error). The elevation was then determined using the Paulin altimeter. This exercise covered 55 line kilometres.

The second phase involved taking readings at every 100m along lines 500m apart with a direction of 51 degrees true north. The GPSs played an important role in identifying gaps and ensuring that the lines being navigated were parallel to each other. Previously established base stations were re-occupied at least every hour. Where base stations were missing, additional stations were tied in with the original. This exercise covered 65 kilometres.

Post Processing

If drift on the altimeter and gravimeter were found to be excessive new readings were taken, otherwise drift corrections were applied to the readings. Using the gravimeters dial constant the raw readings were converted to raw gravity readings. The latitude, Bouguer and free-air corrections were then applied to the data. For the Bougeur correction a density of 2.67 gram per cubic centimetre (g/cc) was used. The terrain-effect was calculated for the observation points closest to the Pilanesberg and was found to be insignificant in relation to the gravitational variations observed.

The resultant xyz positions was then gridded on a 25m grid using a cubic spline gridding algorithm. Filters were applied to this grid and the various products used in an interpretation which included information about the varying thickness of the mafic sheet, the presence of faults and the extent of the IRUPs.

Magnetic Survey

The purpose of the ground magnetic survey was to trace faults and dykes, determine the sense and magnitude of movement of such features and to delineate the highly magnetic IRUPs. It was decided to be consistent with the gravity survey and to use lines of a similar direction and spacing. In practise, however, this was not always possible owing to the magnetic survey's susceptibility to interference from parallel fences, power lines and built-up areas in general. For these reasons as well as possible interference from gravity-related equipment, magnetic surveys are generally done after the gravity survey.

The instruments used for this survey are:

1. Magnetics – *Geometrics G 856* – This instrument is a proton-precession magnetometer used in this case as a total field instrument.
2. Position – *Garmin GPS 12, Garmin GPS 72 and Magellan eXplorist 300* – see gravity survey.

Field Procedure

The field procedure was similar to that of the second phase detailed gravity survey with the GPS used for guidance and covered 65 kilometres. With no equivalent to the gravity survey's first phase and no second magnetometer being used as a base station, a series of magnetic base stations also had to be tied in so that a base station was returned to every 30 minutes. Readings (including time) were taken at an average of 5m intervals. Position was determined by GPS every 100m and other positions interpolated through processing. Possible sources of interference such as fences and power lines were noted.

Post processing

All high-frequency signals associated with cultural effects were removed. The individual lines were then put through various filters and the results presented as stacked profiles and interpreted. Inversion modelling was

also performed on specific anomalies and the results included in the interpretation compilation, together with information on faults, dykes and IRUPs.

Item 12(b): Interpretation of survey (field observation) results

The structural features identified from the aeromagnetic data were interpreted in terms of a regional structural model shown in Diagrams 9(a) and 9(b). Major dyke features were easily recognised and these assisted in the compilation of a structural model for the WBJV project area. Exploration drilling later helped to identify a prominent east-west-trending linear feature as a south-dipping dyke. This dyke occurs along the northern boundary of the project area. A second dyke occurs along the northeastern boundary of the Elandsfontein and Frischgewaagd areas. Other major structural features include potential faults oriented at 345 degrees north in the deep environment of the Frischgewaagd south area.

Item 12(c): Survey (field observation) data collection and compilation

Anglo Platinum supplied the geophysical and satellite imagery data. Mr WJ Visser (PTM) and Mr BW Green were responsible for the interpretation and modelling of the information, with the assistance of AP. All other field data (mapping, soil sampling, XRF, petrography and ground magnetic and gravimetric surveys) were collected, collated and compiled by PTM (RSA) personnel under the guidance and supervision of Mr WJ Visser and are deemed to be reliable and accurate.

ITEM 13: DRILLING

Type and extent of drilling

The type of drilling being conducted on the WBJV is a diamond-drilling core-recovery technique involving a BQ-size solid core extraction. The drilling is placed on an unbiased 500m x 500m grid and detailed when necessary to a 250m x 250m grid. The grid has been extended for 4.5 km along strike to include the whole of the Project 1 area.

Procedures, summary and interpretation of results

The results of the drilling and the general geological interpretation are digitally captured in SABLE and a GIS software package named ARCVIEW. The exact borehole locations, together with the results of the economic evaluation, are plotted on plan. From the geographic location of the holes drilled, regularly spaced sections are drawn by hand and digitised. This information was useful for interpreting the sequence of the stratigraphy intersected as well as for verifying the borehole information.

Comment on true and apparent widths of the mineralised zones

The geometry of the deposit has been clearly defined in the sections drawn through the property. With the exception of three inclined boreholes, all holes were drilled vertically (minus 90 degrees) and the down hole surveys indicate very little deviation. A three-dimensional surface – digital terrain model (DTM) – was created used in the calculation of the average dip of 14 degrees. This dip has been factored into the calculations on which resource estimates are based.

Comment on the orientation of the mineralised zones

The mineralised zones within the project area include the Merensky Reef and the UG2 Reef, both of which are planar tabular ultramafic precipitants of a differentiated magma and therefore form a continuous sheet-like accumulate. The stratigraphic markers above and below the economic horizons have been recognised and facilitate recognition of the Merensky Reef and the UG2 Reef. There are a few exceptions to the quality of recognition of the stratigraphic sequences. These disruptions are generally of a structural nature and are to be expected within this type of deposit. In some boreholes no clear stratigraphic recognition was possible. These holes were excluded from resource calculations.

ITEM 14: SAMPLING METHOD AND APPROACH

Item 14(a): Sampling method, location, number, type and size of sampling

The first step in the sampling of the diamond-drilled core is to mark the core from the distance below collar in one-metre units and then for major stratigraphic units. Once the stratigraphic units are identified, the economic units – Merensky Reef and UG2 Reef – are marked. The top and bottom contacts of the reefs are clearly

marked on the core. Thereafter the core is rotated in such a manner that all lineations pertaining to stratification are aligned to produce a representative split. A centre cut line is then drawn lengthways for cutting. After cutting, the material is replaced in the core trays. The sample intervals are then marked as a line and a distance from collar. The sample intervals are typically 15–25cm in length. In areas where no economic zones are expected, the sampling interval could be as much as a metre. The sample intervals are allocated a sampling number, and this is written on the core for reference purposes. The half-core is then removed and placed into high-quality plastic bags together with a sampling tag containing the sampling number, which is entered onto a sample sheet. The start and end depths are marked on the core with a corresponding line. The duplicate tag stays as a permanent record in the sample booklet, which is secured on site. The responsible project geologist then seals the sampling bag. The sampling information is recorded on a specially designed sampling sheet that facilitates digital capture into the SABLE system (commercially available logging software). The sampling extends for about a metre into the hanging wall and footwall of the economic reefs.

A total of 58,559m has been drilled by PTM from borehole WBJV001 to WBJV120 across the Project 1 area, covering approximately 12,507,656m². Altogether 15,783 samples have been submitted for assaying: 13,282 field samples, 1,243 standards and 1,258 blanks.

Item 14(b): Drilling recovery performance

All reef intersections that are sampled require a 100% core recovery. If less than 100% is recovered, the drilling company will re-drill, using a wedge to achieve the desired recovery.

Item 14(c): Sample quality and sample bias

The sampling methodology accords with PTM protocol based on industry-accepted best practice. The quality of the sampling is monitored and supervised by a qualified geologist. The sampling is done in a manner that includes the entire economic unit together with hanging wall and footwall sampling. Sampling over-selection and sampling bias is eliminated by rotating the core so that the stratification is vertical and by inserting a cutline down the centre of the core and removing one side of the core only.

Item 14(d): Widths of mineralised zones – mining cuts

The methodology in determining the mining cuts is derived from the core intersections. Generally, the economic reefs are about 30cm thick. For both the Merensky Reef and UG2 Reef, the marker unit is the bottom reef contact, which is a chromite contact of less than a centimetre. The cut is taken from that chromite contact to 10cm below and extended vertically to accommodate most of the metal content. If this should result in a mining cut less than a metre up from the bottom reef contact, it is extended further to one metre. If the mining cut is thicker than the proposed metre, the last significant reported sample value above one metre is added to determine the top reef contact.

In view of footwall mineralisation within the Merensky Reef package, the first 25cm footwall sample is included in the mining cut. This ensures that the mining cuts are consistent and can be correlated across the deposit. In the case of the UG2 Reef, the triplets (if and where developed) are included in the mining cut. See Diagram 11 for an illustration of the Merenksy Reef mining cut model.

Item 14(e): Summary of sample composites with values and estimated true widths

Sample composites are shown in Table 1(a) and 1(b) – Appendix A.

ITEM 15: SAMPLE PREPARATION, ANALYSES AND SECURITY

Item 15(a): Persons involved in sample preparation

Drilled core is cleaned, de-greased and packed into metal core boxes by the drilling company. The core is collected from the drilling site on a daily basis by a PTM geologist and transported to the exploration office by PTM personnel. Before the core is taken off the drilling site, the depths are checked and entered on a daily drilling report, which is then signed off by PTM. The core yard manager is responsible for checking all drilled core pieces and recording the following information:

- Drillers' depth markers (discrepancies are recorded).
- Fitment and marking of core pieces.
- Core losses and core gains.
- Grinding of core.
- One-meter-interval markings on core for sample referencing.
- Re-checking of depth markings for accuracy.

Core logging is done by hand on a PTM pro-forma sheet by qualified geologists under supervision of the project geologist, who is responsible for timely delivery of the samples to the relevant laboratory. The supervising and project geologists ensure that samples are transported by PTM contractors.

Item 15(b): Sample preparation, laboratory standards and procedures

Samples are not removed from secured storage location without completion of a chain-of-custody document; this forms part of a continuous tracking system for the movement of the samples and persons responsible for their security. Ultimate responsibility for the secure and timely delivery of the samples to the chosen analytical facility rests with the project geologist and samples are not transported in any manner without the project geologist's permission.

When samples are prepared for shipment to the analytical facility the following steps are followed:

- Samples are sequenced within the secure storage area and the sample sequences examined to determine if any samples are out of order or missing.

- The sample sequences and numbers shipped are recorded both on the chain-of-custody form and on the analytical request form.
- The samples are placed according to sequence into large plastic bags. (The numbers of the samples are enclosed on the outside of the bag with the shipment, waybill or order number and the number of bags included in the shipment).
- The chain-of-custody form and analytical request sheet are completed, signed and dated by the project geologist before the samples are removed from secured storage. The project geologist keeps copies of the analytical request form and the chain-of-custody form on site.
- Once the above is completed and the sample shipping bags are sealed, the samples may be removed from the secured area. The method by which the sample shipment bags have been secured must be recorded on the chain-of-custody document so that the recipient can inspect for tampering of the shipment.

During the process of transportation between the project site and analytical facility the samples are inspected and signed for by each individual or company handling the samples. It is the mandate of both the supervising and project geologist to ensure secure transportation of the samples to the analytical facility. The original chain-of-custody document always accompanies the samples to their final destination.

The supervising geologist ensures that the analytical facility is aware of the PTM standards and requirements. It is the responsibility of the analytical facility to inspect for evidence of possible contamination of, or tampering with, the shipment received from PTM. A photocopy of the chain-of-custody document, signed and dated by an official of the analytical facility, is faxed to PTM's offices in Johannesburg upon receipt of the samples by the analytical facility and the original signed letter is returned to PTM along with the signed analytical certificate/s.

The analytical facility's instructions are that if they suspect the sample shipment has been tampered with, they will immediately contact the supervising geologist, who will arrange for someone in the employment of PTM to examine the sample shipment and confirm its integrity prior to the start of the analytical process.

If, upon inspection, the supervising geologist has any concerns whatsoever that the sample shipment may have been tampered with or otherwise compromised, the responsible geologist will immediately notify the PTM management in writing and will decide, with the input of management, how to proceed. In most cases analysis may still be completed although the data must be treated, until proven otherwise, as suspect and unsuitable as a basis for a news release until additional sampling, quality control checks and examination prove their validity.

Should there be evidence or suspicions of tampering or contamination of the sampling, PTM will immediately undertake a security review of the entire operating procedure. The investigation will be conducted by an independent third party, whose the report is to be delivered directly and solely to the directors of PTM, for their consideration and drafting of an action plan. All in-country exploration activities will be suspended until this review is complete and the findings have been conveyed to the directors of the company and acted upon.

The laboratories that have been used to date are Anglo American Analytical Laboratories, Genalysis (Perth, Western Australia), ALS Chemex (South Africa) and (currently) Set Point Laboratories (South Africa). Dr B Smee has accredited Set Point Laboratories.

Samples are received, sorted, verified and checked for moisture and dried if necessary. Each sample is weighed and the results are recorded. Rocks, rock chips or lumps are crushed using a jaw crusher to less than 10mm. The samples are then milled for 5 minutes in a Labtech Essa LM2 mill to achieve a fineness of 90% less than 106µm, which is the minimum requirement to ensure the best accuracy and precision during analysis.

Samples are analysed for Pt (ppb), Pd (ppb) Rh (ppb) and Au (ppb) by standard 25g lead fire-assay using silver as requested by a co-collector to facilitate easier handling of prills as well as to minimise losses during the cupellation process. Although collection of three elements (Pt, Pd and Au) is enhanced by this technique, the contrary is true for rhodium (Rh), which volatilises in the presence of silver during cupellation. Palladium is used as the co-collector for Rh analysis. The resulting prills are dissolved with aqua regia for ICP analysis.

After pre-concentration by fire assay and microwave dissolution, the resulting solutions are analysed for Au and PGMs by the technique of ICP-OES (inductively coupled plasma–optical emission spectrometry).

Item 15(c): Quality assurance and quality control (QA&QC) procedures and results

The PTM protocols for quality control are as follows:

1. The project geologist (Mr A du Plessis) oversees the sampling process.
2. The core yard manager (Mr P Pitjang) oversees the core quality control.
3. The exploration geologists (Ms B Kgetsi, Mr A Nyilika and Mr L Radebe) and the sample technicians (Mr I Ernst and Mr LJ Selaki) are responsible for the actual sampling process.
4. The project geologist oversees the chain of custody.
5. The internal QP (Mr W Visser) verifies both processes and receives the laboratory data.
6. The internal resource geologist (Mr T Botha) and the database manager (Mr M Rhantho) merge the data and produce the SABLE sampling log with assay values.
7. Together with the project geologist, the resource geologist determines the initial mining cut.
8. The external auditor (Mr N Williams) verifies the sampling process and signs off on the mining cut.

9. The second external database auditor (Mr A Deiss) verifies the SABLE database and highlights QA&QC failures.
10. Ms E Aling runs the QA&QC graphs (standards, blanks and duplicates) and reports anomalies and failures to the internal QP.
11. The internal QP requests re-assays.
12. Check samples are sent to a second laboratory to verify the validity of data received from the first laboratory.

An additional independent external auditor (Mr. N Williams) corroborated the full set of sampling data. This included examination of all core trays for correct number sequencing and labelling. Furthermore, the printed SABLE sampling log (including all reef intersections per borehole) is compared with the actual remaining borehole core left in the core boxes. The following checklist was used for verification:

1. Sampling procedure, contact plus 10cm, sample length 15–25cm.
2. Quality of core (core-loss) recorded.
3. Correct packing and orientation of core pieces.
4. Correct core sample numbering procedure.
5. Corresponding numbering procedure in sampling booklet.
6. Corresponding numbering procedure on printed SABLE log sheet.
7. Comparing SABLE log sheet with actual core markings.
8. Corresponding chain-of-custody forms completed correctly and signed off.
9. Corresponding sampling information in hardcopy borehole files and safe storage.
10. Assay certificates filed in borehole files.
11. Electronic data from laboratory checked with signed assay certificate
12. Sign off each reef intersection (bottom reef contact and mining cut).
13. Sign off completed borehole file.
14. Sign off on inclusion of mining cut into resource database.

Standards

Certified reference standards are inserted into the sampling sequence to assess the accuracy and possible bias of assay values for platinum, palladium, rhodium and gold (tabulated below) and to monitor potential bias of the analytical results.

Standard type	Pt	Pd	Rh	Au
CDN-PGMS-5	Yes	Yes	-	-
CDN-PGMS-5	Yes	Yes	-	Yes
CDN-PGMS-5	Yes	Yes	-	Yes
CDN-PGMS-5	Yes	Yes	-	Yes
CDN-PGMS-5	Yes	Yes	-	-
AMIS0005	Yes	Yes	Yes	-
AMIS0007	Yes	Yes	Yes	-
AMIS0010	Yes	Yes	-	-

Generally the standards are inserted in place of the tenth sample in the sample sequence. The standards are stored in sealed containers and considerable care is taken to ensure that they are not contaminated in any manner (e.g. through storage in a dusty environment, being placed in a less than pristine sample bag or being in any way contaminated in the core saw process).

Assay testing refers to Round Robin programmes involving collection and preparation of material of varying matrices and grades, to provide homogeneous material for developing reference materials (standards) necessary for monitoring assaying. Assay testing is also useful in ensuring that analytical methods are matched to the mineralogical characteristics of the mineralisation being explored. Samples are sent to a sufficient number of international testing laboratories to provide enough assay data to statistically determine a representative mean value and standard deviation necessary for setting acceptance/rejection tolerance limits.

Tolerance limits are set at two and three standard deviations from the Round Robin mean value of the reference material: a single analytical batch is rejected for accuracy when reference material assays are beyond three standard deviations from the certified mean, and any two consecutive standards within the same batch are rejected on the basis of bias when both reference material assays are beyond two standard deviations limit on the same side of the mean.

All 1243 standard sample values for boreholes WBJV001 to WBJV120 were plotted on a graph for each particular standard and element based on the actual Round Robin results. The mean, two standard deviations (Mean+2SDV and Mean-2SDV) and three standard deviations (Mean+3SD and Mean-3SD) were also plotted on these graphs (Appendix B – Graphs 1 to 11).

Reasons why standards failed may include database errors, selection of wrong standards in the field, sample mis-ordering errors and bias from the laboratory. A failed standard is considered to be cause for re-assay if it falls within a determined mining cut for either the Merensky or UG2 Reefs (MRMC and UG2MC). The table below represents these failures. The bulk of the economic value of the reefs is located within the combined value for Pt and Pd with Rh and Au comprising only 10% of the 4E value (refer to page 11 for the prill splits). As requested by a result, standards that failed for Rh and/or Au (Rh evaluated for AMIS0005, AMIS0007 and AMIS0010 standards; Au evaluated for CDN-PGMS-5, 6, 7 and 11) are not included in the final results as the influence is deemed as not of material economic value. Of the submitted 1,243 standard samples submitted, the total number of standards that failed for Pt and/or Pd based on 3 standard deviations is 116. As tabulated below, only 4 of these are deemed to be true failures (present within the mining cut) and were caused by laboratory problems, which account for a mere 0.3% failure rate.

Bhid	Defl	From	To	SampID	Batch_no.	Std_type	Pt	Pd	Reef	Reason
WBJV033	D2	339.31	339.31	P318	200/08/WBJV-013	CDN11	0.08		MRMC	True Lab Failure
WBJV043	D1	579.90	579.90	P527	2005/09/WBJV/P527	CDN11	0.08		UG2MC	True Lab Failure
WBJV099	D2	458.69	458.69	P15626	2006/06/WBJV-040	AMIS0005	0.02	0.03	UG2MC	Possible Blank
WBJV109	D1	533.94	533.94	P17520	2006/07/WBJV-045	AMIS0005	2.08	1.40	UG2MC	AMIS0007

Blanks

The insertion of blanks provides an important check on the laboratory practices, especially potential contamination or sample sequence mis-ordering. Blanks consist of a selection of Transvaal Quartzite pieces (devoid of platinum, palladium, copper and nickel mineralisation) of a mass similar to that of a normal core sample. The blank being used is always noted to track its behaviour and trace metal content. Typically the first blank is sample 5 in a given sampling sequence.

Assay values of 1258 blanks from PTM were plotted on graphs (Appendix B) for each particular element – Pt, Pd, Rh and Au. A warning limit is plotted on the graphs, which is equal to five times the blank background, above which the blank is considered a failure. All blank failures are tabulated below.

Bhid	Defl	From	To	SampID	Batch_no.	Pt	Pd	Rh	Au	Reef	Lab
WBJV007	D0	256.38	256.38	O3048	2005/05 WBJV 007 D0	3.750	1.940	0.340		UG2MC	SETP
WBJV008	D0	323.52	323.52	J2988	2005/04 WBJV008 DO	0.877	0.362				GEN
WBJV026	D0	64.03	64.03	N516	2005/08/WBJV-009				0.060		SETP
WBJV45	D1	563.19	563.19	P3832	2005/11/WBJV-023		0.130			MRMC	SETP
WBJV057	D0	180.50	180.50	P5423	2005/11/WBJV-025	0.540	0.140	0.100			SETP
WBJV099	D1	469.90	469.90	P15531	2006/05/WBJV-039	3.430	2.070	0.610			SETP
WBJV104	D2	535.50	535.50	P17420	2006/06/WBJV-043	2.550	1.560	0.240	0.130		SETP
WBJV113	D1	429.80	429.80	P17672	2006/07/WBJV-046		0.210	0.060		UG2MC	SETP

Of the 1258 blanks submitted, only eight failed, several of these failures being most likely due to data entry errors in the field. This constitutes a mere 0.64% failure rate. The following table summarises possible causes of the failures.

BHID	Defl	ID	Pt	Pd	Rh	Au	Reef	Lab	Reason for failure
WBJV007	D0	O3048	3.750	1.940	0.340		UG2MC	SETP	Possible AMIS0007 standard
WBJV008	D0	J2988	0.877	0.362				GEN	Possible data entry problem
WBJV026	D0	N516				0.060		SETP	True lab failure – possible contamination
WBJV45	D1	P3832		0.130			MRMC	SETP	Geological problem (not within the mining cut)
WBJV057	D0	P5423	0.540	0.140	0.100			SETP	P5424 is actually the blank (inserted incorrectly)
WBJV099	D1	P15531	3.430	2.070	0.610			SETP	Possible AMIS0005 standard
WBJV104	D2	P17420	2.550	1.560	0.240	0.130		SETP	Possible AMIS0007 standard
WBJV113	D1	P17672		0.210	0.060		UG2MC	SETP	True lab failure

Assay validation

Although samples are assayed with reference materials, an assay validation programme is being conducted to ensure that assays are repeatable within statistical limits for the styles of mineralisation being investigated. It should be noted that validation is different from verification; the latter implies 100% repeatability. The assay validation programme entails

- a re-assay programme conducted on standards that failed the tolerance limits set at two and three standard deviations from the Round Robin mean value of the reference material;
- ongoing blind pulp duplicate assays at Set Point Laboratory;
- check assays conducted at an independent assaying facility (Genalysis).

Re-assay

Re-assays have been conducted for two of the failed standards (the pulps) that were identified as failures during the previous analysis (which was done up to borehole WBJV093). This procedure entailed re-submission and re-assaying of failed standard #2 together with standard #1 submitted before and standard #3 submitted after the particular failed standard #2, as well as all submitted field samples (pulps) in between #1 and #3.

Both failed standards plus 12 samples above and 12 samples below in the same analytical batch (totalling 25 samples per failed standard) were sent to Set Point Laboratory for re-assaying using the original pulps as source – a total of 50 samples (pulps). See the table below for a comparison of the failed standards and the re-assayed values. The re-assayed data was examined to ensure that the quality control was acceptable and both failed standards passed on the re-assay.

Bhid	Defl	SampID	Std_type	Pt	Pd	Reef	Lab	Re-assay (Pt)	Comment
WBJV033	D2	P318	CDN11	0.08		MRMC	SETP	0.11	Pass
WBJV043	D1	P527	CDN11	0.08		UG2MC	SETP	0.12	Pass

The new data has been incorporated into the database.

Duplicates

The purpose of having field duplicates is to provide a check on possible sample over-selection. The field duplicate contains all levels of error – core or reverse-circulation cutting splitting, sample size reduction in the prep lab, sub-sampling at the pulp, and analytical error.

Field duplicates were, however, not used on this project by very significant reason of the assemblage of the core. Firstly, BQ core has an outer diameter of only 36.2mm. Secondly, it is friable and brittle owing to the chrome content: this makes it extremely difficult to quarter the core, which usually ends up in broken pieces and not a solid piece of core.

Because of this problem, the laboratory was asked to regularly assay split pulp samples as a duplicate sample to monitor analytical precision (Appendix B). As can clearly be seen on the graphs of the original analysis vs. the duplicate analysis, no irregular values are plotted. This indicates no sample mis-ordering or nugget effect.

The relationship between grade and precision is plotted using the method of Thompson and Howarth (1978) as the mean vs. the absolute difference between the duplicates for each element. The precision for both Pt and Pd is 5% at about 2g/t. No nugget effect is evident in the data, which indicates that the samples were correctly prepared. The precision for Au is near 15% at 2g/t, which overall reflects the low grade of Au in the intersections.

Check assays

Genalysis in Perth, Australia was utilised as the second laboratory for checks on the assay results from Set Point laboratory. A total of 1,056 samples were selected and as most of the check sampling sent to Genalysis was within the mining cuts, the lab was also requested to add osmium (Os), iridium (Ir) and ruthenium (Ru) to the assay process to determine values for these elements. In addition to the extra elements, the laboratory was also required to determine the specific gravity of each sample.

The above request (assaying for Os, Ir and Ru) made it necessary for the lab to use a different assay method to ascertain the values for the different elements. The check sampling was done using nickel-sulphide collection as against lead-collection. From the graphs in Appendix B it is evident that the two laboratories are producing equivalent analyses; this confirms the satisfactory performance of Set Point Laboratory on the standards.

Item 15(d): Adequacy of sampling procedures

The QA&QC practice of PTM is a process beginning with the actual placement of the borehole position (on the grid) and continuing through to the decision for the 3D economic intersection to be included in (passed into) the database. The values are also confirmed, as well as the correctness of correlation of reef/mining cut

so that populations used in the geostatistical modelling are not mixed; this makes for a high degree of reliability in estimates of resources/reserves.

The author of this report (the independent QP) relied on subordonate qualified persons for the following:

- correct sampling procedures (marking, cutting, labelling and packaging) were followed at the exploration office and accurate recording (sample sheets and digital recording in SABLE) and chain-of-custody procedures were followed;
- adequate sampling of the two economic horizons (Merensky and UG2 Reefs) was done;
- preparations by PTM field staff were done with a high degree of precision and no deliberate or inadvertent bias;
- correct procedures were adhered to at all points from field to database;
- PTM's QA&QC system meets or exceeds the requirements of NI 43-101 and mining best practice; and that
- the estimates provided for the Merensky and UG2 Reefs are a fair and valid representation of the actual in-situ value.

The QP's view is supported by Mr N Williams, who audited the whole process (from field to database), and by Mr A Deiss, who regularly audits the SABLE database for correct entry and integrity and also verifies the standards, blanks and duplicates within the database as a second check to the QA&QC graphs run by Ms E Aling.

ITEM 16: DATA VERIFICATION

Item 16(a): Quality control measures and data verification

All scientific information is manually captured and digitally recorded. The information derived from the core logging is manually recorded on A4-size logging sheets. After being captured manually, the data is electronically captured in a digital logging program (SABLE). For this exercise the program has very specific requirements and standards. Should the entered data not be in the set format the information is rejected. This is the first stage of the verification process.

After the information is transferred into SABLE, the same information is transferred into a modelling package (DATAMINE). Modelling packages are rigorous in their rejection of conflicting data, e.g. the input is aborted if there are any overlaps in distances or inconsistencies in stratigraphic or economic horizon nomenclature. This is the second stage of verification.

Once these stages of digital data verification are complete, a third stage is generated in the form of section construction and continuity through DATAMINE. The lateral continuity and the packages of hanging wall and footwall stratigraphic units must align or be in a format consistent with the general geometry. If this is not the case, the information is again aborted.

The final stage of verification is of a geostatistical nature, where population distributions, variance and spatial relationships are considered. Anomalies in grade, thickness, isopach or isocon trends are noted and questioned. Should inconsistencies and varying trends be un-explainable, the base data is again interrogated, and the process is repeated until a suitable explanation is obtained.

Item 16(b): Verification of data

The geological and economic base data has been verified by Mr A Deiss and has been found to be acceptable.

Item 16(c): Nature of the limitations of data verification process

As with all information, inherent bias and inaccuracies can and may be present. Given the verification process that has been carried out, however, should there be a bias or inconsistency in the data, the error would be of no material consequence in the interpretation of the model or evaluation.

The data is checked for errors and inconsistencies at each step of handling. The data is also rechecked at the stage where it is entered into the deposit-modelling software. In addition to ongoing data checks by project staff, the senior management and directors of PTM have completed spot audits of the data and processing procedures. Audits have also been done on the recording of borehole information, the assay interpretation and final compilation of the information.

The individuals in PTM's senior management and certain directors of the company who completed the tests and designed the processes are non-independent mining or geological experts.

Item 16(d): Possible reasons for not having completed a data verification process

There are no such reasons. All data has been verified before being statistically processed.

ITEM 17: ADJACENT PROPERTIES

Comment on public-domain information about adjacent properties

The adjacent property to the south of the WBJV is the Bafokeng Rasimone Platinum Mine (BRPM), which operates under a joint-venture agreement between Anglo Platinum and the Royal Bafokeng Nation. The operation lies directly to the south of the project area and operating stopes are within 1,500m of the WBJV

current drilling area. This is an operational mine and the additional information is published in Anglo Platinum's 2004 Annual Report, which can be found on the www.angloplats.com website.

The Royal Bafokeng Nation has itself made public disclosures and information with respect to the property and these can be found on www.rbr.co.za .

The AP website includes the following points (Investment Analysts Report 11 March 2005):

- Originally, the design was for 200,000 tons per month Merensky Reef operation from twin declines using a dip-mining method. The mine also completed an opencast Merensky Reef and UG2 Reef operation, and mechanised mining was started in the southern part of the mine.
- The planned steady state would be 220,000 tons per month, 80% from traditional breast mining. As a result of returning to traditional breast mining the development requirements are reduced.
- The mining plan reverted to single skilled operators.
- The mine mills about 2,400,000 tons per year with a built-up head grade of 4.30g/t 4E in 2005.
- Mill recovery in 2004 was 85.83%.
- For 2005 the production was 195,000 equivalent refined platinum ounces.
- Operating costs per ton milled in 2002, 2003, 2004 and 2005 were R284/t, R329/t, R372/t and R378/t respectively.

The adjacent property to the north of the WBJV is Wesizwe Platinum Limited **.** The Pilanesberg project of Wesizwe is situated on the farms Frischgewaagd 96 JQ, Ledig 909 JQ, Mimosa 81 JQ and Zandrivierpoort 210 JP. To date 50 boreholes have been drilled and an exploration programme is still actively being conducted.

Wesizwe's interim report for the six months ended 30 June 2006 published by Wesizwe included a resource declaration on the Merensky and the UG2 Reef horizons. The statement was prepared in accordance with Section 12 of listing requirements of the JSE and the South African Code for Reporting of Mineral Resources and Mineral Reserves (SAMREC code). The following table summarises the total estimated mineral resource for the Pilanesberg Project.

Reef	Category	Million tons (Mt)	4PGE Grade (gpt)	Total 4PGE (million ounces)
Merensky and UG2	Indicated	7,950	5.23	1.338
Merensky and UG2	Inferred	61,912	5.10	10,154

Down-dip to the east is AP's Styldrift project of which AP's attributable interest is 50% of the mineral resource and ore reserves. The declared 2005 resource for the project is as follows:

	Merensky Reef		UG2	
Category	**Resource (Mt)**	**Grade 4E (g/t)**	**Resource (Mt)**	**Grade 4E (g/t)**
Measured	-	-	1,7	5,2
Indicated	23,7	5,51	7,9	5,19
Inferred	61,7	6,37	97,1	4,86

Source of adjacent property information

The BRPM operations information is to be found on website www.angloplats.com and the Royal Bafokeng Nation's information on website www.rbr.co.za . Wesizwe Platinum Limited information is on website www.wesizwe.co.za and the Styldrift information on website www.angloplats.com .

Relevance of the adjacent property information

The WBJV deposit is a continuation of the orebody concerned in the BRPM operations and the Wesizwe project, and the information obtained from BRPM and Wesizwe is thus of major significance and appropriate in making decisions about the WBJV.

The technical information on adjoining properties has been provided by other qualified persons and has not been verified by the QP of this report. It may not be indicative of the subject of this report.

Application of the adjacent property information

The BRPM technical and operational information can be useful to the WBJV as far as planning statistics are concerned. However, the overall design and modus operandi of the WBJV is different from that of the BRPM operations and only certain aspects of the BRPM design can be used. The overall design recommendations for the WBJV are based on best-practice approaches in the industry.

ITEM 18: MINERAL PROCESSING AND METALLURGICAL TESTING

During May 2006 SGS Lakefield Research Africa (Pty) Ltd carried out a mineralogical investigation of PGM-bearing ore types (Merensky Reef) from the WBJV project in conjunction with metallurgical test work. SGS Lakefield also performed petrographic and mineralogical work towards the end of 2005 on samples received from the project area. This included XRD analysis (RIR method), optical microscopic examination (modal analysis) and QEM scans (Intellection Pty Ltd, Brisbane, Australia). Samples studied were mostly from the mineralised Merensky and UG2 Reefs.

Mineralogy

The objective of the work was to identify the various rock types and their mineralogical assemblages as well as the PGM, nickel and copper deportment of the reefs. The interpretation of the data may also facilitate early predictions about the metallurgical behaviour of the ore types. The data is presented in three reports received

from SGS Lakefield (MIN0306/015; MIN0805/64 and MIN0805/06). The following is a summary of the findings.

Alteration

Alteration of the silicates within the Merensky Reef is generally low to moderate and confined mainly to fractured zones, where orthopyroxene is altered to talc. Plagioclase is altered to chlorite/sericite. The alteration caused disaggregation of the sulphides into very fine clustered disseminations within the reef.

Sulphide assemblages

Estimates of proportions of sulphides present in the Merensky Reef were based on microscopic observations and geochemical analyses. Sulphide composition of the samples appeared to be variable, ranging in content from 1.8% to 0.3%. Sulphide compositions of composited samples are estimated as pentlandite (43%), pyrrhotite (35%), chalcopyrite (20%) and pyrite (2%). Polished thin-section petrography shows that the sulphides occur as

- sporadically-distributed, fine-grained clusters associated with interstitial silicates (e.g. phlogopite, quartz and amphibole) and also within the boundary confines of altered orthopyroxene and plagioclase; those present are mainly chalcopyrite with minor pyrrhotite and pyrite; particle size varies from 30µm to less than 1µm; or
- isolated, coarser composite particles and blebs consisting mainly of chalcopyrite and pentlandite.

PGM and gold deportment

Five groups of PGM speciation were identified for the Merensky Reef:

a) sulphides
b) arsenides
c) tellurium (Te)-, Antimony (Sb)- and Bismuth (Bi)-bearing
d) Au-bearing phases and
e) Fe-bearing PGMs.

The sulphides comprise about 71% of the PGMs observed, the arsenides 8%, the Te-, Sb- and Bi-bearing PGMs 13%, the Au-bearing phases 7% and the Fe-bearing PGMs about 1%. The major PGM phase encountered was cooperite which comprised 63% of the observed particles. Moncheite comprised 11%; electrum 6% and braggite 5%. Sperrylite is less common, comprising about 4% of the PGMs. Hollingworthite, isoferroplatinum and laurite each comprise about 1.5% in the observed particles.

Results of work on the mineral associations of the PGMs indicate that

- 77% of the PGM+Au phases observed are associated with sulphides (occluded mainly in and/or attached to chalcopyrite and pentlandite);
- 21% of the phases are occluded in silicates (usually in close proximity to sulphides);
- only 2% occur on the boundary between silicate minerals and chromite;
- PGMs occluded in silicates occur mainly in alteration silicates and in interstitial silicate phases (talc, chlorite, quartz, amphibole and phlogopite).

Grain-size distribution

Nearly 40% of the PGMs are sulphides that are larger than 1,000 μm^2 in size. Approximately 75% are larger than 100 μm^2. The Te-, Sb- and Bi-bearing PGMs are generally smaller than the sulphides.

Results of the QEM scan

The QEM scan study was based on five individual reef intersection samples, three from the Merensky and two from the UG2 Reefs. PGMs within the Merensky Reef FPP facies are relatively fine-grained and the liberation characteristics may look disappointing; on the other hand, in the binary and ternary associations these particles are largely exposed. With a finer grind, therefore, the bulk of the PGMs should lend themselves to recovery within the primary flotation circuit.

The pyroxenite contact reef facies studied proved to be low-grade and high in Cr content with partial alteration of the silicates to talc, chlorite, tremolite and quartz. The sulphides have been finely dispersed within the alteration silicates and are thus less amenable to flotation.

The two UG2 Reef intersections proved to be similar with respect to the mineralogy and deportment of the sulphides. One sample contained anomalous magnetite, which was probably due to iron-rich ultramafic pegmatite replacement (IRUP). This also affected the speciation of the PGMs and the sample contained mainly PtFe alloys as the dominant PGM phase. The PGMs are very fine-grained but with optimal grind size good recoveries are possible.

Metallurgical test work

Merenksy Reef

Eight core samples were available for the metallurgical test work. The bulk of the comminution and flotation work was carried out on a composite of the inner dog box core samples, and variability flotation tests were carried out on the individual inner dog box core samples. The data were received from SGS Lakefield in the form of a written report. The main results of the comminution test work are as follows:

- Bond abrasion tests on the inner dog box composite sample categorise the ore as having a medium abrasion tendency (0.2–0.5g).
- Bond rod mill work index (BRWI) tests on the composite sample classify the ore type as hard (15.60 kWh/t.).
- Bond ball mill work index (BBWI) tests classify the composite sample as hard (18.5 kWh/t.). The ratio of BBWI:BRWI is less than 1, which indicates that the ore breaks down relatively easily into a size that can be handled by a secondary grinding mill.

The main results of the flotation test work are as follows:

- Flotation tests using the standard flotation conditions to determine the effect of grind produced results showing that a finer grind (90% less than 75µm) increased the 3E recovery rate to 82% from that achieved by a coarser grind of (60% less than 75µm).
- Reagent optimisation flotation tests showed that an increased SIBX dosage of 70g/t, 50g/t Aero 5747 and 20g/t KU-47 reagent suite produced the highest 3E recoveries (94.73%). These were selected as the optimal flotation conditions.
- A two-stage cleaning process of the rougher concentrate produced a final 3E concentrate at 57.74g/t.
- A four-cycle locked-cycle test using the two-stage cleaning process flow sheet showed that it did not conclusively reach mass stabilisation and concentrate grades were lower than for the two-stage cleaning process. These factors could not be conclusively validated because of the limited sample mass available for the test work.

Variability flotation test work on the individual core samples showed that the deposit is variable as none of the samples produced results similar to that of the composite sample.

UG2 Reef

Of the Frischgewaagd UG2 material available for testwork, eight samples of low metal value were composited for comminution tests.

- The Bond Rod Mill Work Index was 12.1kWh/ton which is relatively hard.
- The Bond Ball Mill Work Index was 17.3kWh/ton which is relatively hard.

A further twelve bore core samples were subjected to milling and flotation response, including composites representing the Eastern area and Western area respectively. Rougher recoveries compare well with those obtained from test work performed on UG2 ore samples taken from an adjacent property.

A locked cycle test performed on a composite sample has been completed. The best-fit curve indicates that a final concentrate of 150g/t 4E corresponds to a recovery of around 83%. Equilibrium mass pull was 2.6%

Process Design

Using an MF2 type mill and flotation circuit, in which an initial rougher flotation follows primary grinding to 50% less than 75microns, followed by a secondary flotation of primary flotation tails after a regrind to 80% less than 75microns and individual cleaning of the rougher concentrates, laboratory results indicate that overall recoveries of PGMs + Au will exceed 80% in a low mass final concentrate of around 150g/t. The concentrator has been accepted as producing a concentrate with a grade of 150g/t 4E's for both Merensky and UG2 reefs with corresponding overall metal recoveries of 87.5 and 82.5% respectively.

Additional metallurgical samples have been submitted for further test work and will be reported on in the planned Feasibility study.

ITEM 19: MINERAL RESOURCE ESTIMATES

Item 19(a): Standard reserve and resource reporting system

The author has complied with the SAMREC Code for reporting mineral resources and mineral reserves. The code allows for a resource or reserve to be upgraded (or downgraded) if, amongst other things, economic, legal, environmental, permitting circumstances change. A set of geological and geostatistical rules have been applied for this mineral resource classification, which also relies on the structural and facies aspects of the geology. These rules are consistent with the Inferred, Indicated and Measured Resource classification as set out in the SAMREC code.

Item 19(b): Comment on reserves and resources subsets

This report deals primarily with the Inferred, Indicated and Measured Resources. The specific data distribution and geographic layout allows the Inferred Resource to qualify for upgrading to higher-confidence resource categories.

Item 19(c): Comment on Inferred Resource

The definition of the resource is as given in the SAMREC Code and the Inferred Resource is calculated and reported separately.

Item 19(d): Relationship of the QP to the issuer

Apart from having been contracted to compile this report the QP has no commercial or other relationship with PTM.

Item 19(e): Detailed mineral resource tabulation

From the interpolated block model a mineral resource calculation was made in respect of the pegmatoidal feldspathic pyroxenite (FPP) facies and contact reef (CR) of the Merensky Reef; and of the UG2 Reef. The FPP domain covers the pegmatoidal feldspathic pyroxenite and harzburgite facies of the Merensky Reef. Table 2a shows the tonnage and grade for each facies at specific cut-off grades for 4E (cmg/t.). The cut-off grade categories are based on content because the interpolation was done on content, as was the mechanism for the change of support or post-processing.

Table 2(a): Mineral resource for the Merensky and UG2 Reefs.

Cut-off (4E)	Tonnage	Tonnage (-12% geological loss)	Avg grade (4E)	Metal content (4E)	Mining width
cmg/t	tons (t)	tons (t)	g/t	g	cm
Merensky (CR facies) Indicated					
0	6,754,547	5,944,001	0.76	4,521,437	101
100	999,936	879,944	2.30	2,021,340	101
200	305,268	268,636	4.60	1,236,152	101
300	208,279	183,286	5.68	1,041,590	101
400	166,070	146,141	6.35	927,496	101
500	131,985	116,147	7.01	813,746	101
600	101,931	89,699	7.73	692,942	101
Merensky (CR facies) Inferred					
0	6,005,927	5,285,216	0.54	2,868,081	100
100	446,253	392,703	1.29	507,592	100
200	17,084	15,034	2.37	35,692	100
300	1,237	1,089	3.47	3,779	100
400	138	122	4.70	573	100
500	35	31	5.81	181	100
600	11	10	6.85	69	100
Merensky (FPP facies) Measured					
0	2,542,426	2,237,335	6.96	15,565,233	124
100	2,486,160	2,187,821	7.11	15,554,756	124
200	2,475,972	2,178,855	7.13	15,542,540	124
300	2,440,674	2,147,793	7.20	15,474,534	124
400	2,324,471	2,045,534	7.42	15,172,746	124
500	2,112,285	1,858,811	7.79	14,475,063	124
600	1,834,435	1,614,303	8.28	13,367,478	124

Merensky (FPP facies) Indicated					
0	18,000,000	15,840,000	6.37	100,976,246	126
100	17,700,000	15,576,000	6.46	100,630,851	126
200	17,400,000	15,312,000	6.57	100,615,673	126
300	16,400,000	14,432,000	6.85	98,829,167	126
400	14,700,000	12,936,000	7.30	94,455,335	126
500	12,600,000	11,088,000	7.89	87,486,571	126
600	10,600,000	9,328,000	8.57	79,944,355	126
Merensky (FPP facies) Inferred					
0	2,925,351	2,574,309	6.59	16,961,128	122
100	2,920,503	2,570,043	6.60	16,958,298	122
200	2,834,049	2,493,963	6.76	16,857,921	122
300	2,610,347	2,297,105	7.16	16,452,491	122
400	2,301,165	2,025,025	7.74	15,679,047	122
500	1,969,440	1,733,107	8.43	14,616,943	122
600	1,654,228	1,455,721	9.19	13,385,070	122
UG2 Measured					
0	2,858,561	2,515,534	3.08	7,736,393	147
100	2,575,224	2,266,197	3.35	7,599,914	147
200	2,306,184	2,029,442	3.62	7,342,415	147
300	2,124,755	1,869,784	3.77	7,046,834	147
400	1,824,169	1,605,269	3.96	6,357,745	147
500	1,305,480	1,148,822	4.30	4,937,222	147
600	752,977	662,620	4.77	3,157,735	147
UG2 Indicated					
0	35,200,000	30,976,000	2.51	77,873,261	150
100	28,600,000	25,168,000	2.98	74,891,109	150
200	21,100,000	18,568,000	3.70	68,743,471	150
300	17,800,000	15,664,000	4.09	63,988,897	150
400	14,700,000	12,936,000	4.42	57,238,074	150
500	11,200,000	9,856,000	4.83	47,640,730	150
600	7,823,528	6,884,705	5.29	36,451,502	150
0	15,000,000	13,200,000	3.15	41,539,978	150
UG2 Inferred					
100	13,400,000	11,792,000	3.48	40,991,657	150
200	10,600,000	9,328,000	4.12	38,438,561	150

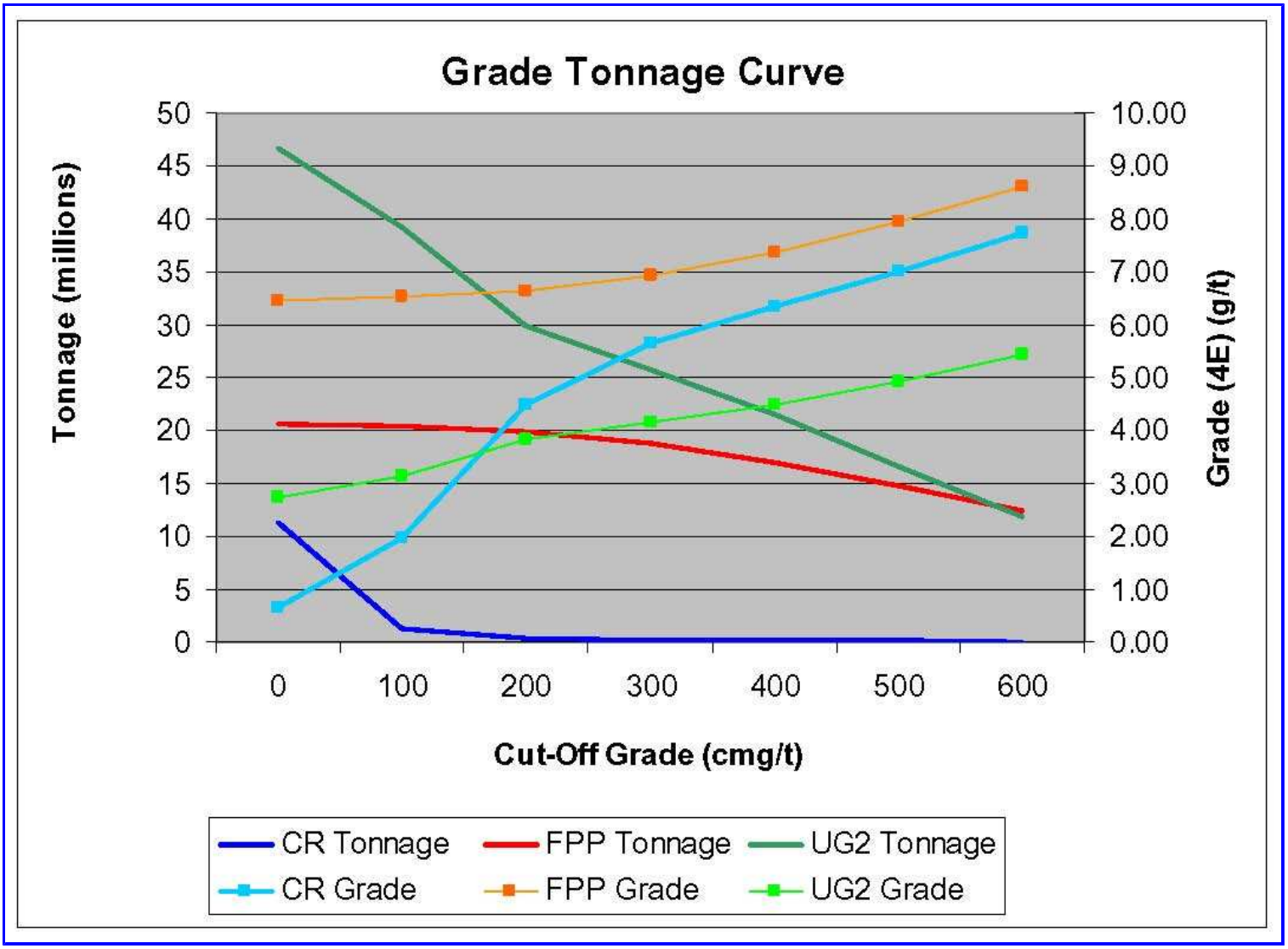
300	9,414,515	8,284,773	4.43	36,740,583	150
400	8,041,117	7,076,183	4.78	33,836,609	150
500	6,412,221	5,642,754	5.22	29,477,958	150

600	4,831,474	4,251,697	5.73	24,378,853	150

Table 2(b): Mineral resource including copper and nickel.

Cut-off (4E)	Tonnage	Tonnage (-12% geological loss)	Avg grade (4E)	Metal content (4E)		Mining width	Cu	Ni
cmg/t	tons (t)	tons (t)	g/t	g	Moz	cm	%	%
Merensky (CR facies) Indicated								
300	208,279	183,286	5.68	1,041,590	0.033	101	0.0318	0.0787
Merensky (CR facies) Inferred								
300	1,237	1,089	3.47	3,779	0.000	100	0.0243	0.0661
Merensky (FPP facies) Measured								
100	2,486,160	2,187,821	7.11	15,554,756	0.500	124	0.0934	0.2019
Merensky (FPP facies) Indicated								
100	17,700,000	15,576,000	6.46	100,630,851	3.235	126	0.0885	0.1922
Merensky (FPP facies) Inferred								
100	2,920,503	2,570,043	6.60	16,958,298	0.545	122	0.0946	0.2008
UG2 Measured								
100	2,575,224	2,266,197	3.35	7,599,914	0.244	147	0.0077	0.0859
UG2 Indicated								
100	28,600,000	25,168,000	2.98	74,891,109	2.408	150	0.0058	0.0692
UG2 Inferred								
100	13,400,000	11,792,000	3.48	40,991,657	1.318	150	0.0075	0.0772

Diagram 12 shows the grade tonnage curve for the different reefs and respective facies.


Grade Tonnage Curve
Tonnage (millions)
Grade (4E) (g/t)
50
45
40
35
30
25
20
15
10
5
0
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
0
100
200
300
400
500
600
Cut-Off Grade (cmg/t)
CR Tonnage
FPP Tonnage
UG2 Tonnage
CR Grade
FPP Grade
UG2 Grade

A cut-off grade of 100cmg/t was selected as a resource cut-off for the FPP facies and the UG2; for the CR facies the cut-off is 300cmg/t.

The resources include the upgrading to the Measured and Indicated mineral resource categories of a portion of the Merensky Reef and UG2 mineral resources. Drilling activity by PTM, the operator of the WBJV, has to date covered approximately 40% of the WBJV surface area and involved 58,559 metres of drilling. This update includes the results up to borehole 120, along with previous results from Anglo Platinum. The resources are estimated by the kriging method and the Indicated Resources have a drill spacing of 250 metres or less. In keeping with best practice in resource estimation, an allowance for known and expected geological losses is made. These account for approximately 19% of the area. This number was considered when the resource was estimated.

The prill split estimates of the platinum, palladium, rhodium and gold (4E) have been provided in compliance with Canadian National Policy 43-101. Caution must be exercised with respect to these estimates as they have been calculated by simple arithmetic means. While a rigorous statistical process of resource estimates has been completed on the combined 4E grades consistent with South African platinum industry best practice for estimation, the prill split has been calculated using the arithmetic mean of the assay information. A summary of the declared resources as described is given below:

Estimated Measured Resource base:

MR FPP = Merensky Reef pegmatoidal feldspathic pyroxenite; MR CR = Merensky Reef contact reef; and UG2 = Upper Group No. 2 chromitite seam; PGM = Platinum-group metals.
The cut-offs for Indicated and Inferred Resources have been established by a qualified person after a review of potential operating costs and other factors.

Measured Resource							
	Cut-off (cmg/t) 4E	**Million tons**	**Grade (g/t) 4E**	**Mining width (cm)**		**Tons PGM (4E)**	**Million ounces PGMs (4E)**
MR FPP	100	2.187	7.11	1.24		15.554	0.500
UG2	100	2.266	3.35	1.47		7.599	0.244
Total Measured		4.453	5.20			23.153	0.744

Prill Splits	**Pt**	**Pt (g/t)**	**Pd**	**Pd (g/t)**	**Rh**	**Rh (g/t)**	**Au**	**Au (g/t)**
MR FPP	62%	4.42	26%	1.85	5%	0.36	7%	0.48
UG2	64%	2.15	24%	0.80	10%	0.35	1%	0.05

Estimated Indicated Resource base:

MR FPP = Merensky Reef pegmatoidal feldspathic pyroxenite; MR CR = Merensky Reef contact reef; and UG2 = Upper Group No. 2 chromitite seam; PGM = Platinum-group metals.
The cut-offs for Indicated and Inferred Resources have been established by a qualified person after a review of potential operating costs and other factors.

Indicated Resource							
	Cut-off (cmg/t) 4E	**Million tons**	**Grade (g/t) 4E**	**Mining width (cm)**		**Tons PGM (4E)**	**Million ounces PGMs (4E)**
MR FPP	100	15.575	6.46	1.26		100.630	3.235
MR CR	300	0.183	5.68	1.01		1.040	0.033
UG2	100	25.168	2.98	1.50		74.891	2.408
Total Indicated		40.926	4.31			176.561	5.676

Prill Splits	**Pt**	**Pt (g/t)**	**Pd**	**Pd (g/t)**	**Rh**	**Rh (g/t)**	**Au**	**Au (g/t)**
MR FPP	62%	4.02	26%	1.68	5%	0.33	7%	0.43
MR CR	62%	3.53	26%	1.48	5%	0.29	7%	0.38
UG2	64%	1.91	24%	0.72	10%	0.31	1%	0.04

Independently estimated Inferred Resource base:

MR FPP = Merensky Reef pegmatoidal feldspathic pyroxenite; MR CR = Merensky Reef contact reef; and UG2 = Upper Group No. 2 chromitite seam; PGM = Platinum-group metals.
The cut-offs for Indicated and Inferred Resources have been established by a qualified person after a review of potential operating costs and other factors.

Inferred Resource							
	Cut-off (cmg/t) 4E	**Million tons**	**Grade (g/t) 4E**	**Mining width (cm)**		**Tons PGM (4E)**	**Million ounces PGMs (4E)**
MR FPP	100	2.570	6.56	1.22		16.958	0.545
MR CR	300	0.001	3.50	1.00		0.004	0.0002
UG2	100	11.792	3.48	1.50		40.991	1.318
Total Inferred		14.363	4.03			57.953	1.8632

Prill Splits	**Pt**	**Pt (g/t)**	**Pd**	**Pd (g/t)**	**Rh**	**Rh (g/t)**	**Au**	**Au (g/t)**
MR FPP	62%	4.08	26%	1.70	5%	0.34	7%	0.44
MR CR	62%	2.18	26%	0.91	5%	0.18	7%	0.23
UG2	64%	2.23	24%	0.84	10%	0.36	1%	0.05

Minor elements (Ru, Ir and Os)

Assaying for Ru, Ir and Os are expensive and time consuming and are therefore generally not done. Laboratories in South Africa are not accredited to assay for these elements and therefore samples need to be sent to Genalysis in Australia for reliable assaying.

PTM send a total of 1056 samples over the Merensky and UG2 mining horizons to Genalysis for assaying.

Regression

The known Ru, Ir and Os values were plotted against Pt and Pd to obtain the best correlation. Pt showed the best correlation and was used to estimate the absent Ru, Ir and Os values from a regressed formula. The samples were only taken over the mining cuts for the particular borereholes. A total of 146 Ru, 125 Ir and 117 Os values for the Merensky Reef mining cut and 450 Ru, 441 Ir and 434 Os values for the UG2 mining cut were used to calculate the regression formula. Graphs 12 and 13 – Appendix B – demonstrate the correlation between the different elements.

Overall evaluation of the project

These elements were used in the financial model, but caution should be taken of the following:

- The number of assayed samples in relation to the number of Pt/Pd samples is limited.
- The major portion of values is obtained from regressed values.
- The confidence in these elements is low and therefore has a greater risk than the other elements in the Indicated and Measured Resources.

The contribution of these elements to the total revenue is relatively low and will therefore not compromise the overall evaluation of the project.

Item 19(f): Key assumptions, parameters and methods of resource calculation

A total of 287 borehole intersections were utilised in the resource calculation (see Diagram 5) of which only 129 intersections could be used for Merensky Reef mineral resource estimation and 158 for UG2. A number of historical boreholes were found not to meet the quality assurance criteria and were not used in the evaluation of the project area.

An area towards the southwest has been identified where resource estimation was not possible for the Merensky Reef, owing to the diamond drilling information having intersected the reefs at less than 50m from surface resulting in excessive core loss due to the presence of the regolith for the first 40m. A further reason is that reef identification and correlation problems often occur due to incomplete core as a result of thinning of the reefs and/or stratigraphy.

Reef width for purposes of these resource estimates refers to a mining cut of one metre or more. The methodology in determining the mining cuts is derived from the core intersections. Generally, the economic reefs are less than one metre thick. The marker unit for both the Merensky and UG2 Reefs is the bottom reef contact – a chromite seam of less than one centimetre. The mining cut is taken from this chromite contact to 10cm below and extended vertically to include most of the metal content. If the resultant mining cut is less than one metre up from the chromitite contact, it is extended further to one metre in length. If thicker, the last significant reported sample value above one metre is added to determine the top reef contact. The first 25cm footwall sample is included in the mining cut because of footwall mineralisation within the Merensky Reef package. If, in the case of the UG2 Reef, the Triplets are developed they are included in the mining cut. This methodology ensures that the mining cuts are consistent and can be correlated across the deposit.

Borehole reef widths and 4E grades used in the resource estimation exercises are depicted in Tables 1(a) and (b).

The available borehole data is derived from previous drilling by AP and the recently drilled PTM holes. The AP borehole PGM values consisted of Pt, Pd, Rh and Au. For some of the later boreholes, Rh values was not assayed for by AP and these were inferred from existing relationships of Pt and Rh values (see Diagrams 13 and 14).

Diagram 13: Scatter plot of Rh vs. Pt for the Merensky Reef.

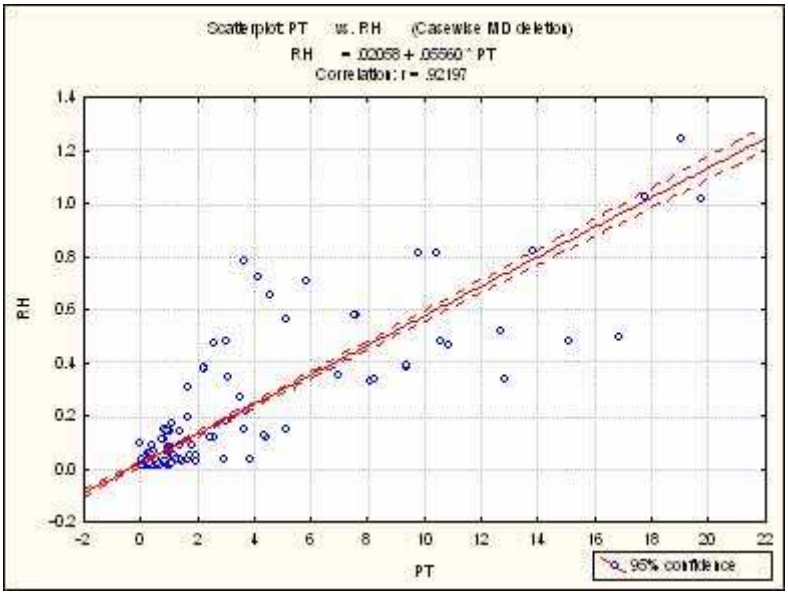
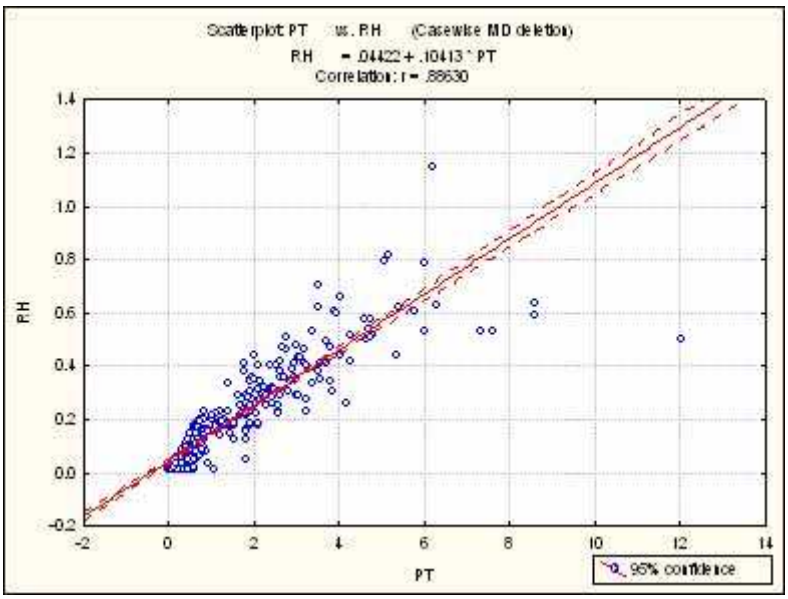


Diagram 14: Scatter plot of Rh vs. Pt for the UG2 Reef.



In the evaluation process the metal content (4E cmg/t) and reef width (cm) values are used. The reef width refers to the corrected reef width. The values have been interpolated into a 2D block model. The 4E grade (g/t) has been calculated from the interpolated content and reef width values. A 3D dip model was created from the 3D wireframes of the respective reefs. The dip values in the model were used for vertical thickness corrections in tonnage calculations.

For modelling purposes, the Merensky Reef was divided only into two facies types with respective geological domains (Diagram 15) whereas the UG2 consists of only one facies type with different geological domains (Diagram 16). Grade and reef width estimates were calculated within specific geological domains.

Statistical analysis

Descriptive statistics in the form of histograms (frequency distributions) and probability plots (to evaluate the normality of the distribution of a variable) were used to develop an understanding of the statistical relationships. Skewness is a measure of the deviation of the distribution from symmetry (0 = no skewness). Kurtosis measures the "peakedness" of a distribution (3 = normal distribution).

Descriptive statistics for the Merensky and the UG2 Reefs are summarised in Tables 3, 4 and 5.

Table 3: Descriptive statistics – mining width.

Merensky Reef (MR CR facies) – Domain 1.

Descriptive Statistics (Spreadsheet1)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_RC_MRMC_CW	55	106.3015	79.17573	185.8318	330.997	18.19331	2.482355	8.70669
DOM1_RC_MRMC_3PGE	55	0.5864	0.04000	1.9993	0.342	0.58491	0.999383	-0.29329
DOM1_RC_MRMC_CM3PG	55	64.3597	3.55713	280.5944	4728.438	68.76364	1.308469	0.97613
Ln_DOM1_RC_MRMC_CW	55	4.6543	4.37167	5.2248	0.022	0.14977	1.578111	4.81991
Ln_DOM1_RC_MRMC_3PGE	55	-1.1487	-3.21888	0.6928	1.476	1.21483	-0.101775	-1.31248
Ln_DOM1_RC_MRMC_CM3PG	55	3.5056	1.26895	5.6369	1.557	1.24772	-0.054840	-1.23291

Merensky Reef (MR FPP facies) - Domain 2.

Descriptive Statistics (Spreadsheet3)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2_RC_MRMC_CW	46	131.1978	95.05119	346.236	3491.3	59.0872	2.75078	7.173034
DOM2_RC_MRMC_3PGE	46	6.1961	0.05016	15.735	14.0	3.7452	0.59612	0.278907
DOM2_RC_MRMC_CM3PG	46	829.1300	6.22305	3483.975	432571.5	657.7017	1.85127	5.103641
Ln_DOM2_RC_MRMC_CW	46	4.8138	4.55442	5.847	0.1	0.3216	2.11042	4.095138
Ln_DOM2_RC_MRMC_3PGE	46	1.4762	-2.99254	2.756	1.4	1.1729	-2.48351	6.960357
Ln_DOM2_RC_MRMC_CM3PG	46	6.2900	1.82826	8.156	1.6	1.2544	-2.14928	5.658797

Merensky Reef (MR FPP facies) - Domain 3.

Descriptive Statistics (Spreadsheet5)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM3_RC_MRMC_CW	28	126.7440	96.8620	240.178	1509	38.847	1.910947	2.79393
DOM3_RC_MRMC_3PGE	28	6.5479	1.3729	24.667	25	4.963	2.430853	6.55365
DOM3_RC_MRMC_CM3PG	28	927.1442	162.1625	5924.381	1304805	1142.281	3.627222	14.41770
Ln_DOM3_RC_MRMC_CW	28	4.8067	4.5733	5.481	0	0.255	1.561294	1.57227
Ln_DOM3_RC_MRMC_3PGE	28	1.6844	0.3169	3.205	0	0.611	0.354583	1.17279
Ln_DOM3_RC_MRMC_CM3PG	28	6.4911	5.0886	8.687	1	0.739	1.027964	2.39723

UG2 – Domain 1.

Descriptive Statistics (Spreadsheet1)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_RC_UG2MC_CW	13	106.6515	92.45316	174.5989	477.4899	21.85154	2.862146	9.056873
DOM1_RC_UG2MC_3PGE	13	0.4876	0.19576	0.8565	0.0416	0.20402	0.437965	-0.544990
DOM1_RC_UG2MC_CM3PG	13	51.0110	21.35757	90.1405	464.6709	21.55623	0.749735	-0.071019
Ln_DOM1_RC_UG2MC_CW	13	4.6545	4.52670	5.1625	0.0290	0.17022	2.493327	7.174906
Ln_DOM1_RC_UG2MC_3PGE	13	-0.8053	-1.63089	-0.1549	0.2001	0.44730	-0.374405	-0.486741
Ln_DOM1_RC_UG2MC_CM3PG	13	3.8491	3.06141	4.5014	0.1847	0.42974	-0.167719	-0.123987

UG2 – Domain 2.

Descriptive Statistics (Spreadsheet3)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2_RC_UG2MC_CW	26	124.4473	92.67276	263.1005	1663.97	40.7918	2.35607	5.530925
DOM2_RC_UG2MC_3PGE	26	3.3085	0.68567	6.2535	2.23	1.4918	0.22317	-0.351385
DOM2_RC_UG2MC_CM3PG	26	422.2858	66.60158	901.0032	56623.41	237.9567	0.50435	-0.619991
Ln_DOM2_RC_UG2MC_CW	26	4.7858	4.52907	5.5725	0.07	0.2616	1.80756	3.002012
Ln_DOM2_RC_UG2MC_3PGE	26	1.0727	-0.37736	1.8331	0.30	0.5520	-1.00935	0.823114
Ln_DOM2_RC_UG2MC_CM3PG	26	5.8586	4.19873	6.8035	0.46	0.6761	-0.73558	0.117352

UG2 – Domain 3.

Descriptive Statistics (Spreadsheet5)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM3_RC_UG2MC_CW	30	135.2410	84.42822	302.2288	4321.21	65.7359	1.896148	2.266220
DOM3_RC_UG2MC_3PGE	30	0.9335	0.06113	4.4183	1.04	1.0209	2.209050	4.820856
DOM3_RC_UG2MC_CM3PG	30	118.3646	7.43372	425.7662	12220.89	110.5482	1.233347	0.883103
Ln_DOM3_RC_UG2MC_CW	30	4.8256	4.43590	5.7112	0.14	0.3767	1.575526	1.259343
Ln_DOM3_RC_UG2MC_3PGE	30	-0.5486	-2.79469	1.4857	1.08	1.0406	-0.268132	0.191025
Ln_DOM3_RC_UG2MC_CM3PG	30	4.2770	2.00603	6.0539	1.23	1.1112	-0.370891	-0.644517

UG2 – Domain 4.

Descriptive Statistics (Spreadsheet7)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM4_RC_UG2MC_CW	38	153.3827	95.7463	355.908	2616.40	51.1508	1.879903	5.511993
DOM4_RC_UG2MC_3PGE	38	4.1705	1.8166	6.949	1.50	1.2238	-0.062359	-0.630283
DOM4_RC_UG2MC_CM3PG	38	613.3670	235.1866	1503.531	44748.27	211.5379	2.012277	7.565625
Ln_DOM4_RC_UG2MC_CW	38	4.9882	4.5617	5.875	0.09	0.2925	0.762510	0.763876
Ln_DOM4_RC_UG2MC_3PGE	38	1.3810	0.5970	1.939	0.10	0.3217	-0.625484	-0.417663
Ln_DOM4_RC_UG2MC_CM3PG	38	6.3692	5.4604	7.316	0.10	0.3162	0.070851	2.339774

UG2 – Domain 5.

Descriptive Statistics (Spreadsheet9)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM5_RC_UG2MC_CW	12	190.8358	102.5171	419.5260	7527.70	86.7623	1.720356	4.04344
DOM5_RC_UG2MC_3PGE	12	1.1966	0.3701	3.7993	1.37	1.1720	1.861485	2.16955
DOM5_RC_UG2MC_CM3PG	12	203.2765	43.5891	506.0234	23514.02	153.3428	0.936770	-0.45108
Ln_DOM5_RC_UG2MC_CW	12	5.1717	4.6300	6.0391	0.16	0.4052	0.553121	0.51024
Ln_DOM5_RC_UG2MC_3PGE	12	-0.1270	-0.9939	1.3348	0.55	0.7433	1.225685	0.61455
Ln_DOM5_RC_UG2MC_CM3PG	12	5.0447	3.7748	6.2266	0.61	0.7825	0.058703	-1.02986

UG2 – Domain 6.

Descriptive Statistics (Spreadsheet11)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM6_RC_UG2MC_CW	25	151.2144	98.2704	295.869	2465.3	49.6521	1.814649	3.017533
DOM6_RC_UG2MC_3PGE	25	3.8868	1.7062	7.281	2.0	1.4299	0.460496	-0.324525
DOM6_RC_UG2MC_CM3PG	25	619.7642	190.6954	1814.423	166855.4	408.4793	1.815887	2.951137
Ln_DOM6_RC_UG2MC_CW	25	4.9772	4.5877	5.690	0.1	0.2799	1.172167	1.261143
Ln_DOM6_RC_UG2MC_3PGE	25	1.2899	0.5343	1.985	0.1	0.3821	-0.218694	-0.766289
Ln_DOM6_RC_UG2MC_CM3PG	25	6.2671	5.2507	7.504	0.3	0.5586	0.528788	0.243394

UG2 – Domain 7.

Descriptive Statistics (Spreadsheet13)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM7_RC_UG2MC_CW	37	149.1554	95.05119	403.4233	5314.24	72.8989	2.219297	4.94959
DOM7_RC_UG2MC_3PGE	37	1.0312	0.07432	4.8621	0.69	0.8308	2.936227	12.05824
DOM7_RC_UG2MC_CM3PG	37	145.1046	9.05023	544.9783	11894.09	109.0600	1.950886	4.99979
Ln_DOM7_RC_UG2MC_CW	37	4.9237	4.55442	6.0000	0.14	0.3765	1.437758	1.45158
Ln_DOM7_RC_UG2MC_3PGE	37	-0.2143	-2.59938	1.5815	0.55	0.7408	-0.667324	2.47203
Ln_DOM7_RC_UG2MC_CM3PG	37	4.7094	2.20279	6.3007	0.65	0.8087	-0.876051	1.80708

UG2 – Domain 8.

Descriptive Statistics (Spreadsheet15)

Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM8_RC_UG2MC_CW	5	172.7258	107.1500	241.701	3501.7	59.1754	0.167200	-2.58204
DOM8_RC_UG2MC_3PGE	5	4.5977	3.4887	6.472	1.3	1.1461	1.368143	2.16313
DOM8_RC_UG2MC_CM3PG	5	836.7735	439.5542	1564.303	228855.3	478.3883	1.049236	-0.15351
Ln_DOM8_RC_UG2MC_CW	5	5.1027	4.6742	5.488	0.1	0.3535	-0.096448	-2.38695
Ln_DOM8_RC_UG2MC_3PGE	5	1.5028	1.2495	1.867	0.1	0.2334	0.993584	1.29145
Ln_DOM8_RC_UG2MC_CM3PG	5	6.6055	6.0858	7.355	0.3	0.5486	0.568252	-1.71036

Table 4: Descriptive statistics – copper and nickel.

	Descriptive Statistics (Spreadsheet1)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM1_RC_MRMC_CU%	36	0.02324	0.00100	0.09934	0.000772	0.027792	1.260367	0.47790
DOM1_RC_MRMC_NI%	36	0.06620	0.01455	0.16948	0.002148	0.046343	0.855250	-0.48806
Ln_DOM1_RC_MRMC_CU%	36	-4.65075	-6.90776	-2.30925	2.169229	1.472830	0.045052	-1.37177
Ln_DOM1_RC_MRMC_NI%	36	-2.95974	-4.22989	-1.77500	0.518711	0.720216	0.094746	-1.25753

	Descriptive Statistics (Spreadsheet3)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM2_RC_MRMC_CU%	25	0.09008	0.00348	0.19227	0.002937	0.054198	0.15184	-0.366559
DOM2_RC_MRMC_NI%	25	0.20180	0.05216	0.39186	0.006599	0.081231	0.45940	0.162407
Ln_DOM2_RC_MRMC_CU%	25	-2.76465	-5.66159	-1.64885	1.234846	1.111236	-1.62147	1.832848
Ln_DOM2_RC_MRMC_NI%	25	-1.68862	-2.95354	-0.93686	0.204903	0.452663	-0.84345	1.258622

	Descriptive Statistics (Spreadsheet5)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM3_RC_MRMC_CU%	19	0.09506	0.06006	0.15882	0.000585	0.024192	0.738106	1.387010
DOM3_RC_MRMC_NI%	19	0.20442	0.11614	0.34094	0.003035	0.055095	0.653748	0.812018
Ln_DOM3_RC_MRMC_CU%	19	-2.38337	-2.81246	-1.83998	0.063715	0.252419	-0.029867	-0.013251
Ln_DOM3_RC_MRMC_NI%	19	-1.62166	-2.15300	-1.07606	0.072622	0.269485	-0.089175	-0.092617

	Descriptive Statistics (Spreadsheet7)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM1_RC_UG2MC_CU%	13	0.00241	0.00102	0.00490	0.000002	0.001273	0.781214	-0.64168
DOM1_RC_UG2MC_NI%	13	0.04955	0.03323	0.06913	0.000140	0.011813	0.649933	-1.04521
Ln_DOM1_RC_UG2MC_CU%	13	-6.15322	-6.88599	-5.31852	0.271272	0.520837	0.219171	-1.28364
Ln_DOM1_RC_UG2MC_NI%	13	-3.02996	-3.40439	-2.67182	0.053338	0.230951	0.384834	-0.95820

	Descriptive Statistics (Spreadsheet9)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM2_RC_UG2MC_CU%	18	0.00574	0.00119	0.02608	0.000034	0.005819	2.733253	9.08981
DOM2_RC_UG2MC_NI%	18	0.08353	0.04698	0.11187	0.000451	0.021243	-0.109305	-1.40199
Ln_DOM2_RC_UG2MC_CU%	18	-5.51095	-6.73128	-3.64674	0.718275	0.847511	0.214476	-0.13045
Ln_DOM2_RC_UG2MC_NI%	18	-2.51543	-3.05812	-2.19044	0.072331	0.268944	-0.415665	-0.99182

	Descriptive Statistics (Spreadsheet11)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM3_RC_UG2MC_CU%	21	0.00413	0.00100	0.01216	0.000009	0.003008	1.510110	2.122018
DOM3_RC_UG2MC_NI%	21	0.05765	0.02839	0.08855	0.000305	0.017455	-0.078548	-0.940517

Ln_DOM3_RC_UG2MC_CU%	21	-5.72042	-6.90776	-4.40960	0.494760	0.703392	0.018513	-0.461362
Ln_DOM3_RC_UG2MC_NI%	21	-2.90210	-3.56158	-2.42419	0.109017	0.330177	-0.559407	-0.690535

	Descriptive Statistics (Spreadsheet13)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM4_RC_UG2MC_CU%	30	0.01281	0.00374	0.06117	0.000176	0.013249	3.103223	9.520514
DOM4_RC_UG2MC_NI%	30	0.09487	0.06425	0.17956	0.000581	0.024099	1.780897	4.266636
Ln_DOM4_RC_UG2MC_CU%	30	-4.62356	-5.58947	-2.79418	0.426948	0.653412	1.234754	2.168024
Ln_DOM4_RC_UG2MC_NI%	30	-2.38157	-2.74494	-1.71722	0.050717	0.225204	1.009899	1.360407

	Descriptive Statistics (Spreadsheet15)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM5_RC_UG2MC_CU%	12	0.00484	0.00222	0.00887	0.000005	0.002170	0.638115	-0.73535
DOM5_RC_UG2MC_NI%	12	0.06208	0.02122	0.11654	0.000725	0.026925	0.544409	0.02517
Ln_DOM5_RC_UG2MC_CU%	12	-5.42377	-6.10845	-4.72508	0.205141	0.452925	0.030977	-1.04371
Ln_DOM5_RC_UG2MC_NI%	12	-2.87301	-3.85276	-2.14956	0.220025	0.469068	-0.527205	0.34523

	Descriptive Statistics (Spreadsheet17)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM6_RC_UG2MC_CU%	21	0.00838	0.00242	0.01788	0.000015	0.003866	0.599460	0.208075
DOM6_RC_UG2MC_NI%	21	0.08882	0.05090	0.11746	0.000464	0.021530	-0.572502	-0.992574
Ln_DOM6_RC_UG2MC_CU%	21	-4.89288	-6.02316	-4.02407	0.252292	0.502287	-0.474302	-0.175009
Ln_DOM6_RC_UG2MC_NI%	21	-2.45344	-2.97787	-2.14164	0.073225	0.270602	-0.850545	-0.587297

	Descriptive Statistics (Spreadsheet19)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM7_RC_UG2MC_CU%	22	0.00389	0.00100	0.00939	0.000006	0.002431	0.933324	-0.140667
DOM7_RC_UG2MC_NI%	22	0.05474	0.02723	0.08971	0.000183	0.013525	0.571243	1.155204
Ln_DOM7_RC_UG2MC_CU%	22	-5.73547	-6.90776	-4.66854	0.396788	0.629911	0.041629	-0.879294
Ln_DOM7_RC_UG2MC_NI%	22	-2.93480	-3.60358	-2.41123	0.063811	0.252609	-0.423641	1.442905

	Descriptive Statistics (Spreadsheet21)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM8_RC_UG2MC_CU%	3	0.01378	0.00723	0.01864	0.000035	0.005889		-1.18615
DOM8_RC_UG2MC_NI%	3	0.10222	0.08008	0.12226	0.000448	0.021171		-0.44431
Ln_DOM8_RC_UG2MC_CU%	3	-4.36018	-4.92938	-3.98250	0.251655	0.501652		-1.46760
Ln_DOM8_RC_UG2MC_NI%	3	-2.29550	-2.52477	-2.10160	0.045706	0.213789		-0.72396

Table 5: Descriptive statistics – channel width.

	Descriptive Statistics (Spreadsheet1)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM1_DD_MRCW_PT	46	1.05279	0.01000	7.91800	2.3281	1.52581	2.583347	8.57862
DOM1_DD_MRCW_PD	46	0.50017	0.01000	3.68400	0.4734	0.68802	2.562089	9.26987
DOM1_DD_MRCW_RH	46	0.09961	0.01000	0.67000	0.0235	0.15346	2.165817	4.30291
DOM1_DD_MRCW_AU	46	0.06822	0.01000	0.54400	0.0103	0.10154	2.803323	10.04271
DOM1_DD_MRCW_3PGE	46	1.72080	0.04000	12.58774	5.7518	2.39830	2.555523	8.70968
DOM1_DD_MRCW_OS	46	0.03715	0.01776	0.14508	0.0009	0.03014	2.281233	4.93290
DOM1_DD_MRCW_IR	46	0.05349	0.01641	0.26473	0.0032	0.05694	2.297986	5.26839
DOM1_DD_MRCW_RU	46	0.22452	0.00300	1.02766	0.0554	0.23536	2.321971	5.14380
DOM1_DD_MRCW_CU%	31	0.03979	0.00100	0.31560	0.0048	0.06898	2.837155	8.66554
DOM1_DD_MRCW_NI%	31	0.09847	0.02250	0.42020	0.0094	0.09713	2.199626	4.93675
DOM1_DD_MRCW_CW	46	33.26440	10.35219	96.58302	511.0751	22.60697	1.576209	1.37923
Ln_DOM1_DD_MRCW_PT	46	-1.21220	-4.60517	2.06914	3.6121	1.90054	-0.225979	-1.21595

	Descriptive Statistics (Spreadsheet3)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM2_DD_MRCW_PT	39	7.69709	0.02000	21.3874	28.522	5.34065	0.66789	-0.10211
DOM2_DD_MRCW_PD	39	2.89726	0.01000	7.7565	3.236	1.79901	0.42147	0.06011
DOM2_DD_MRCW_RH	39	0.44031	0.01000	1.0300	0.071	0.26601	0.31744	-0.61000
DOM2_DD_MRCW_AU	39	0.47806	0.01000	1.3420	0.110	0.33102	0.73131	0.36761
DOM2_DD_MRCW_3PGE	39	11.51272	0.05000	27.2168	56.251	7.50006	0.42535	-0.63064
DOM2_DD_MRCW_OS	39	0.14061	0.01792	0.3037	0.007	0.08185	0.40839	-0.77210
DOM2_DD_MRCW_IR	39	0.24582	0.01673	0.5741	0.023	0.15259	0.51831	-0.45637
DOM2_DD_MRCW_RU	39	0.99133	0.00800	2.2099	0.373	0.61092	0.38085	-0.76274
DOM2_DD_MRCW_CU%	26	0.13169	0.00112	0.3739	0.008	0.09185	0.57224	0.49078
DOM2_DD_MRCW_NI%	26	0.27280	0.09897	0.5644	0.020	0.14198	0.36882	-1.00512
DOM2_DD_MRCW_CW	39	75.63406	18.37574	443.1454	9094.338	95.36424	2.58161	6.45744

	Descriptive Statistics (Spreadsheet5)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM3_DD_MRCW_PT	27	8.45649	2.01000	23.1704	30.2849	5.50318	1.268653	1.409072
DOM3_DD_MRCW_PD	27	3.00466	0.75239	8.9761	3.3303	1.82490	1.683851	3.400032
DOM3_DD_MRCW_RH	27	0.46233	0.04000	1.5709	0.1037	0.32207	1.657046	4.200551
DOM3_DD_MRCW_AU	27	0.53928	0.05985	1.4268	0.1050	0.32399	1.086158	0.863606
DOM3_DD_MRCW_3PGE	27	12.46276	3.09745	34.6532	60.0487	7.74911	1.361689	1.804549
DOM3_DD_MRCW_OS	27	0.15412	0.02700	0.3906	0.0086	0.09252	0.954148	0.903493
DOM3_DD_MRCW_IR	27	0.26991	0.02800	0.7437	0.0337	0.18346	1.075979	1.003464
DOM3_DD_MRCW_RU	27	1.09717	0.16000	2.8580	0.4681	0.68421	0.994919	0.967748
DOM3_DD_MRCW_CU%	20	0.15090	0.06610	0.3144	0.0035	0.05910	0.902719	1.735954

DOM3_DD_MRCW_NI%	20	0.28850	0.16550	0.4764	0.0059	0.07694	0.607302	0.460219
DOM3_DD_MRCW_CW	27	43.86811	17.51993	116.2606	661.2914	25.71559	1.342704	1.476597

	Descriptive Statistics (Spreadsheet7)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM1_DD_UG2CW_PT	13	0.52521	0.09354	1.1300	0.096	0.31052	0.84697	0.27240
DOM1_DD_UG2CW_PD	13	0.21610	0.02000	0.7700	0.045	0.21118	1.74972	3.13855
DOM1_DD_UG2CW_RH	13	0.09348	0.02342	0.1600	0.002	0.04036	0.11535	-0.11300
DOM1_DD_UG2CW_AU	13	0.01282	0.01000	0.0200	0.000	0.00448	1.05320	-0.94309
DOM1_DD_UG2CW_3PGE	13	0.84762	0.16437	2.0800	0.249	0.49929	1.11112	2.09119
DOM1_DD_UG2CW_OS	13	0.05619	0.04014	0.0841	0.000	0.01415	0.93232	-0.26927
DOM1_DD_UG2CW_IR	13	0.06112	0.03286	0.1033	0.000	0.01866	0.69264	0.93895
DOM1_DD_UG2CW_RU	13	0.35025	0.21328	0.5252	0.011	0.10547	0.69003	-0.74319
DOM1_DD_UG2CW_CU%	13	0.00252	0.00100	0.0067	0.000	0.00203	1.17120	-0.18161
DOM1_DD_UG2CW_NI%	13	0.06272	0.03900	0.0912	0.000	0.01569	0.06886	-0.85163
DOM1_DD_UG2CW_CW	13	41.80465	3.69813	151.5800	1643.146	40.53574	1.83095	3.85105
Ln_DOM1_DD_UG2CW_PT	13	-0.82894	-2.36933	0.1222	0.464	0.68096	-0.77548	0.98242

	Descriptive Statistics (Spreadsheet9)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM2_DD_UG2CW_PT	22	2.50668	0.72571	4.2175	0.662	0.81355	-0.26251	0.44706
DOM2_DD_UG2CW_PD	22	1.02356	0.26616	2.4107	0.244	0.49413	0.99971	1.76992
DOM2_DD_UG2CW_RH	22	0.34483	0.11100	0.4630	0.011	0.10250	-0.87919	0.03230
DOM2_DD_UG2CW_AU	22	0.02678	0.00778	0.0549	0.000	0.01496	0.45342	-1.16064
DOM2_DD_UG2CW_3PGE	22	3.90185	1.21626	6.5705	1.722	1.31236	-0.09331	0.42752
DOM2_DD_UG2CW_OS	22	0.11965	0.06209	0.1756	0.001	0.02799	-0.09552	-0.04444
DOM2_DD_UG2CW_IR	22	0.19965	0.07585	0.3133	0.003	0.05812	-0.20366	0.15918
DOM2_DD_UG2CW_RU	22	0.88414	0.40514	1.3263	0.050	0.22352	-0.24713	0.15192
DOM2_DD_UG2CW_CU%	15	0.00698	0.00100	0.0275	0.000	0.00722	1.85029	3.91426
DOM2_DD_UG2CW_NI%	15	0.08449	0.06184	0.1149	0.000	0.01742	0.36163	-1.16964
DOM2_DD_UG2CW_CW	22	77.62291	18.95448	237.2637	2745.977	52.40207	1.38901	2.74102
Ln_DOM2_DD_UG2CW_PT	22	0.85328	-0.32060	1.4392	0.164	0.40501	-1.53261	2.94808

	Descriptive Statistics (Spreadsheet11)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM3_DD_UG2CW_PT	25	0.73136	0.11000	3.0064	0.598	0.77323	1.995148	3.13746
DOM3_DD_UG2CW_PD	25	0.39334	0.03000	2.2181	0.307	0.55409	2.467262	5.70025
DOM3_DD_UG2CW_RH	25	0.13324	0.02000	0.4411	0.010	0.10134	1.840755	3.26505
DOM3_DD_UG2CW_AU	25	0.01642	0.00100	0.1101	0.000	0.02218	3.762184	14.58960
DOM3_DD_UG2CW_3PGE	25	1.27435	0.20000	5.3346	1.999	1.41377	2.079925	3.48734
DOM3_DD_UG2CW_OS	25	0.06427	0.02300	0.1362	0.001	0.02621	1.407903	1.84290
DOM3_DD_UG2CW_IR	25	0.07666	0.02300	0.2309	0.003	0.05286	1.910566	2.96411
DOM3_DD_UG2CW_RU	25	0.42212	0.14300	1.0068	0.043	0.20803	1.582170	2.16820
DOM3_DD_UG2CW_CU%	18	0.00433	0.00100	0.0130	0.000	0.00335	1.039705	0.95499
DOM3_DD_UG2CW_NI%	18	0.06727	0.04500	0.0969	0.000	0.01394	0.244778	-0.24874
DOM3_DD_UG2CW_CW	25	70.68277	3.80205	299.3496	6148.926	78.41509	2.155782	4.25057
Ln_DOM3_DD_UG2CW_PT	25	-0.70565	-2.20727	1.1007	0.751	0.86634	0.471712	0.07094

	Descriptive Statistics (Spreadsheet13)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM4_DD_UG2CW_PT	18	2.87378	0.02000	3.8882	0.6891	0.83012	-2.50345	8.45865
DOM4_DD_UG2CW_PD	18	1.22632	0.01000	1.8692	0.1672	0.40885	-1.38506	4.03499
DOM4_DD_UG2CW_RH	18	0.42038	0.01000	0.5673	0.0148	0.12161	-2.28721	7.65217
DOM4_DD_UG2CW_AU	18	0.02444	0.01000	0.0779	0.0003	0.01664	2.09077	5.67956
DOM4_DD_UG2CW_3PGE	18	4.54492	0.05000	6.0888	1.5713	1.25351	-2.82759	10.55008
DOM4_DD_UG2CW_OS	18	0.13253	0.03915	0.1649	0.0007	0.02739	-2.43257	8.09871
DOM4_DD_UG2CW_IR	18	0.22282	0.02786	0.2909	0.0032	0.05678	-2.50927	8.38047
DOM4_DD_UG2CW_RU	18	0.97547	0.21898	1.2394	0.0488	0.22098	-2.47161	8.26191
DOM4_DD_UG2CW_CU%	15	0.00992	0.00383	0.0249	0.0000	0.00592	1.48990	1.87762
DOM4_DD_UG2CW_NI%	15	0.07734	0.06092	0.0888	0.0001	0.01003	-0.41007	-1.39320
DOM4_DD_UG2CW_CW	18	87.44601	18.37574	153.7759	983.5020	31.36084	0.17870	1.18817
Ln_DOM4_DD_UG2CW_PT	18	0.82369	-3.91202	1.3579	1.4175	1.19060	-4.13944	17.38081

	Descriptive Statistics (Spreadsheet15)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM5_DD_UG2CW_PT	9	1.1196	0.42580	3.1668	0.883	0.93947	1.585656	1.91514
DOM5_DD_UG2CW_PD	9	0.3962	0.11000	1.1728	0.171	0.41299	1.597463	0.78882
DOM5_DD_UG2CW_RH	9	0.1801	0.09689	0.4206	0.011	0.10671	1.770480	2.79022
DOM5_DD_UG2CW_AU	9	0.0112	0.00110	0.0304	0.000	0.01091	0.917680	-0.45426
DOM5_DD_UG2CW_3PGE	9	1.7070	0.74207	4.6804	2.047	1.43084	1.541253	1.26217
DOM5_DD_UG2CW_OS	9	0.0761	0.05234	0.1414	0.001	0.02983	1.584264	2.03202
DOM5_DD_UG2CW_IR	9	0.1069	0.05545	0.2418	0.004	0.06197	1.520859	1.93573
DOM5_DD_UG2CW_RU	9	0.5149	0.32603	1.0491	0.060	0.24410	1.600604	2.01684
DOM5_DD_UG2CW_CU%	9	0.0055	0.00179	0.0161	0.000	0.00456	1.863939	3.71242
DOM5_DD_UG2CW_NI%	9	0.0660	0.02960	0.1165	0.001	0.03242	0.733911	-1.19551
DOM5_DD_UG2CW_CW	9	101.5421	29.22506	209.8703	3219.048	56.73666	0.662532	0.21812

	Descriptive Statistics (Spreadsheet17)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM6_DD_UG2CW_PT	17	2.9572	1.36889	4.9581	0.975	0.98755	0.143523	-0.57576
DOM6_DD_UG2CW_PD	17	1.1494	0.23000	2.8848	0.459	0.67746	0.925341	1.06148
DOM6_DD_UG2CW_RH	17	0.4125	0.28571	0.6627	0.012	0.11055	0.633991	-0.25079
DOM6_DD_UG2CW_AU	17	0.0262	0.00524	0.0903	0.001	0.02407	1.430754	1.57846
DOM6_DD_UG2CW_3PGE	17	4.5454	2.28159	8.3628	2.966	1.72225	0.494544	-0.40798
DOM6_DD_UG2CW_OS	17	0.1348	0.08299	0.1996	0.001	0.03211	0.101357	-0.64735
DOM6_DD_UG2CW_IR	17	0.2272	0.11958	0.3637	0.004	0.06625	0.097833	-0.55935
DOM6_DD_UG2CW_RU	17	0.9992	0.57481	1.5216	0.070	0.26452	0.106725	-0.71903
DOM6_DD_UG2CW_CU%	13	0.0075	0.00279	0.0159	0.000	0.00389	0.814219	0.02691
DOM6_DD_UG2CW_NI%	13	0.0792	0.04751	0.1175	0.001	0.02297	0.182588	-1.13744
DOM6_DD_UG2CW_CW	17	108.1626	19.41659	233.1791	3602.102	60.01751	0.811920	-0.03757

	Descriptive Statistics (Spreadsheet19)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM7_DD_UG2CW_PT	35	1.07760	0.03000	3.3511	0.637	0.79836	1.13379	0.63866
DOM7_DD_UG2CW_PD	35	0.38266	0.02000	1.5900	0.179	0.42262	1.71429	2.19077
DOM7_DD_UG2CW_RH	35	0.20954	0.01000	0.5931	0.016	0.12603	1.31546	1.47786
DOM7_DD_UG2CW_AU	35	0.01332	0.01000	0.0400	0.000	0.00755	2.54735	5.95635
DOM7_DD_UG2CW_3PGE	35	1.68313	0.07000	5.4729	1.614	1.27027	1.22114	0.86383
DOM7_DD_UG2CW_OS	35	0.08110	0.00500	0.1391	0.001	0.02785	-0.02226	0.57187
DOM7_DD_UG2CW_IR	35	0.10461	0.00500	0.2506	0.003	0.05314	0.93869	0.67885
DOM7_DD_UG2CW_RU	35	0.54617	0.02600	1.1037	0.048	0.21800	0.42974	0.66238
DOM7_DD_UG2CW_CU%	20	0.00447	0.00100	0.0142	0.000	0.00330	1.51612	2.74010
DOM7_DD_UG2CW_NI%	20	0.05929	0.03474	0.1167	0.000	0.01694	2.09223	6.56668
DOM7_DD_UG2CW_CW	35	66.70270	6.32052	201.9808	2111.990	45.95639	0.92131	0.68059
Ln_DOM7_DD_UG2CW_PT	35	-0.21944	-3.50656	1.2093	0.764	0.87431	-1.28209	4.55947

	Descriptive Statistics (Spreadsheet21)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
Variable								
DOM8_DD_UG2CW_PT	5	3.2846	2.80726	3.8600	0.171	0.41362	0.35852	-0.70166
DOM8_DD_UG2CW_PD	5	1.2862	0.87040	2.7206	0.646	0.80390	2.20823	4.89884
DOM8_DD_UG2CW_RH	5	0.4186	0.37953	0.4793	0.002	0.04118	0.88972	-0.64461
DOM8_DD_UG2CW_AU	5	0.0389	0.02000	0.0910	0.001	0.02937	2.14827	4.70957
DOM8_DD_UG2CW_3PGE	5	5.0283	4.14627	6.6619	1.007	1.00358	1.38135	1.78840
DOM8_DD_UG2CW_OS	5	0.1444	0.12909	0.1574	0.000	0.01173	-0.43151	-1.86764
DOM8_DD_UG2CW_IR	5	0.2465	0.21382	0.2664	0.001	0.02346	-0.81216	-1.88738
DOM8_DD_UG2CW_RU	5	1.1029	0.99483	1.2426	0.011	0.10522	0.17939	-1.56937
DOM8_DD_UG2CW_CU%	3	0.0151	0.00741	0.0196	0.000	0.00669	-1.64955	
DOM8_DD_UG2CW_NI%	3	0.1033	0.07549	0.1262	0.001	0.02569	-0.82248	
DOM8_DD_UG2CW_CW	5	118.0094	38.17538	206.6192	4177.214	64.63137	0.19418	-0.41515

No corrections were made to the data and the statistical analyses show the expected relationships for these types of reef.

Variography

Variograms are a useful tool for investigating the spatial relationships of samples. Variograms for metal content (4E cmg/t) and reef width (cm) were modelled. The log variogram is used to assist in establishing the expected structures, ranges and nugget effect for the untransformed 4E cmg/t values in specific domains. Note: the untransformed variograms and not the log-variograms are used for the kriging.

No anisotrophy was found and therefore all variograms were modelled as omnidirectional. All variograms were modelled as two structure variograms. Table 6 summarises the variogram model parameters for the different reefs and domains.

Table 6: Variogram parameters.

Reef	Parameter	Domain	Nugget %	Sill 1 %	R1 (m)	R2 (m)	R3 (m)	Sill 2 %	R1 (m)	R2 (m)	R3 (m)
UG2MC	cw	1	40	78	116	116	1	100	270	270	1
UG2MC	cw	2	39	100	354	354	1	100	-	-	-
UG2MC	cw	3	37	87	302	302	1	100	546	546	1
UG2MC	cw	4	42	100	305	305	1	100	-	-	-
UG2MC	cw	5	36	100	251	251	1	100	-	-	-
UG2MC	cw	6	21	100	342	342	1	100	-	-	-
UG2MC	cw	7	39	100	258	258	1	100	-	-	-
UG2MC	cw	8	41	100	257	257	1	100	-	-	-
CRMC	cw	1	44	80	109	109	1	100	258	258	1
FPPMC	cw	1	40	74	203	203	1	100	407	407	1
FPPMC	cw	2	40	75	217	217	1	100	546	546	1
UG2MC	cmgt	1	34	74	107	107	1	100	262	262	1
UG2MC	cmgt	2	40	100	254	254	1	100	-	-	-
UG2MC	cmgt	3	38	100	254	254	1	100	-	-	-
UG2MC	cmgt	4	17	100	307	307	1	100	-	-	-
UG2MC	cmgt	5	40	100	252	252	1	100	-	-	-
UG2MC	cmgt	6	41	100	410	410	1	100	-	-	-
UG2MC	cmgt	7	44	100	254	254	1	100	-	-	-
UG2MC	cmgt	8	42	100	254	254	1	100	-	-	-
CRMC	cmgt	1	40	74	99	99	1	100	254	254	1
FPPMC	cmgt	1	33	75	204	204	1	100	550	550	1
FPPMC	cmgt	2	39	74	201	201	1	100	396	396	1

R = range; cw = mining cut width; cmgt = 4E content

Grade estimation

The full reef composite values (4E content – cmg/t) and reef width (cm) have been interpolated into a 2D block model. Both simple kriging (SK) and ordinary kriging (OK) techniques have been used. It has been shown that the SK technique is more efficient when limited data are available for the estimation process.

The 4E grade concentration (g/t) was calculated from the interpolated kriged 4E content (cmg/t) and reef width (cm) values. Detailed checks were done to validate kriging outputs, including input data, kriged estimates and kriging efficiency checks.

The simple kriging process uses a local or global mean as a weighting factor. For this exercise 800m x 800m blocks have been selected to calculate the local mean value for each block in respective domains. A minimum of 16 samples were required for a 800m x 800m block to be assigned a local mean value; otherwise a domain

global mean was assigned. Most of the blocks were assigned a global domain mean in the SK process with only a few blocks that used a local mean where there was enough data support.

The following parameters were used in the kriging process:

1. point data – metal content (4E cmg/t) and reef width (cm)
2. 200m x 200m x 1m block size
3. discretisation 40 x 40 x 1 for each 200m x 200m x 1m block
4. first search volume – 500m
    a. minimum number of samples 4
    b. maximum number of samples 40
5. second search volume
    a. 1.5 x first search volume
    b. minimum number of samples 2
    c. maximum number of samples 40
6. third search volume
    a. 3 x first search volume
    b. minimum number of samples 1
    c. maximum number of samples 20
7. interpolation methods – simple kriging and ordinary kriging
8. local and domain global mean values used in the simple kriging process.

Diagrams 17 to 22 show the reef width, 4E grade (g/t) and 4E content (cmg/t) plots for the Merensky and UG2 Reefs.

Post-processing

During early stages of projects the data is invariably on a relatively large grid. This grid is much larger than the block size of a selective mining interest, i.e. selective mining units (SMU). Efficient kriging estimates for SMUs or of much larger blocks units will then be smoothed due to information effect or size of blocks. Any mine plan or cash flow calculations made on the basis of the smoothed kriged estimates will misrepresent the economic value of the project, i.e., the average grade above cut-off will be underestimated and the tonnage overestimated. Some form of post-processing is required to reflect the realistic tonnage grade estimates for respective cut-offs. Using the limited data available preliminary post-processed analysis has been done.

An SMU of 20m x 30m was selected with an expected future underground sampling configuration on a 20m x 20m grid. Information effects were calculated based on the SMU and the expected future production underground sampling configuration.

Within the parent blocks of 200m x 200m x 1m, the distribution of selective mining units has been estimated for various cut-offs. The latter has been estimated using lognormal distribution of SMUs within the large parent blocks – 200m x 200m x 1m (see Assibey-Bonsu and Krige, 1999). This technique for post-processing has been used based on the observed lognormal distribution of the underlying 4E values in the project area (i.e. the indirect lognormal post-processing technique has been used for the change of support analysis).

For each parent block the grade, tonnage and metal content above respective cut-offs (based on the SMUs) were translated into parcels to be used for mine planning. Grade tonnage curves were therefore calculated for each parent block. The following cut-offs were considered 100, 200, 300, 400, 500 and 600 cmg/t.

A specific gravity (SG) of 3.13 was used for the Merensky Reef and 3.60 for the UG2 Reef tonnage calculations. SG values are average values based on measured values for specific reef intersections.

Resource classification

The mineral resource classification is a function of the confidence of the whole process from drilling, sampling, geological understanding and geostatistical relationships. The following aspects or parameters were considered for resource classification:

1. Sampling – quality assurance & quality control
    a. Measured: high confidence, no problem areas
    b. Indicated: high confidence, some problem areas with low risk
    c. Inferred: some aspects might be of medium to high risk
2. Geological confidence
    a. Measured: high confidence in the understanding of geological relationships, continuity of geological trends and sufficient data
    b. Indicated: Good understanding of geological relationships
    c. Inferred: geological continuity not established
3. Number of samples used to estimate a specific block
    a. Measured: at least 4 boreholes within variogram range and minimum of twenty one-metre composited samples
    b. Indicated: at least 3 boreholes within variogram range and a minimum of twelve one-metre composite samples
    c. Inferred: less than 3 borehole within the variogram range
4. Kriged variance
    a. This is a relative parameter and is only an indication and used in conjunction with the other parameters

5. Distance to sample (variogram range)
    a. Measured: at least within 60% of variogram range
    b. Indicated: within variogram range
    c. Inferred: further than variogram range
6. Lower confidence limit (blocks)
    a. Measured: less than 20% from mean (80% confidence)
    b. Indicated: 20%–40% from mean (80%–60% confidence)
    c. Inferred: more than 40% (less than 60% confidence)
7. Kriging efficiency
    a. Measured: more than 40%
    b. Indicated: 20–40%
    c. Inferred: less than 20%
8. Deviation from lower 90% confidence limit (data distribution within resource area considered for classification)
    a. Measured: less than 10% deviation from mean
    b. Indicated: 10–20%
    c. Inferred: more than 20%

Using the above criteria the current Merensky Reef and UG2 reefs in the delineated project area were classified as Measured, Indicated and Inferred Mineral Resources (Diagrams 23 and 24).

Item 19(g): Potential impact of reserve and resource declaration

The intention of this report is to produce a resource update base on the Inferred, Indicated and Measured Resources. However it is assumed in this report that the environmental conditions, permitting, legal and political issues are favourable. Taxation and marketing issues will be applied in real and un-escalated terms. At this time, the project does not have sufficient confidence levels in the legal, permitting, social and engineering aspects for the resources to be converted to reserves.

Item 19(h): Technical parameters affecting the reserve and resource declaration

Technical parameters specific to a planar and tabular precious metal deposit are well understood and are referred to as the flow-of-ore parameters. The methodology takes into account the intentional and unintentional increase in tonnage due to mining. It also takes into account the unintentional and unaccounted loss of metal or metal not reaching the plant or recovered by the plant.

A cut-off grade (4E) of 100cmg/t (MR FPP and UG2) and 300cmg/t (MR CR facies) was applied to the grade tonnage tabulations for both the Merensky and the UG2 Reef in anticipation of tonnages falling below the cut-

off that would not be economically viable. It is clear that detailed optimisation studies need to be done in order to declare specific cut-offs based on e.g. the working costs, metallurgical recoveries, metal prices and previous work done in the Preliminary Assessment Report (SEDAR-filed 12 December 2005). It is however the opinion of the expert estimating the resources that a provisional 100cmg/t and 300cmg/t cut-off for the MR and UG2 respectively, would be fair and reasonable for the declaration of the resources in this report.

Item 19(i): 43-101 rules applicable to the reserve and resource declaration

Regarding the terms on which this report is issued, the estimates of Inferred, Indicated and Measured Resources are permissible. The 43-101 regulations pertaining to this declaration are set out in the Canadian National Instrument 43-101 *Technical Report* (Form F1) and the National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (Companion Policy).

Item 19(j): Disclosure of Inferred Resource

Inferred Resources are NOT included in the economical analyses.

Item 19(k): Demonstrated viability

Mineral resources are not reserves and do not have demonstrated viability. The project currently does not have sufficient confidence levels regarding legal, permitting, social and engineering aspects to convert the resources to reserves.

Item 19(l): Quality, quantity and grade of declared resource

See Table 1(a) and Table 1(b) – Appendix A.

Item 19(m): Metal splits for declared resource

See Table 1(a) and Table 1(b) – Appendix A.

ITEM 20: OTHER RELEVANT DATA AND INFORMATION

The economic viability of mineral resources declared in this report has not been demonstrated. Such deductions can only be made once, among other things, financial and working cost estimates are applied to the resource. See Item 19(h).

RSA reserve and resource declaration rules

The South African Code for Reporting of Mineral Resources and Mineral Reserves (SAMREC Code) sets out minimum standards, recommendations and guidelines for public reporting of exploration results, mineral resources and mineral reserves in South Africa.

Documentation prepared for public release must be done by or under the direction of, and signed by, a Qualified Person. A Qualified Person (QP) is a person who is a member of the South African Council for Natural Scientific Professions (SACNASP) or the Engineering Council of South Africa (ECSA) or any other statutory South African or international body that is recognised by SAMREC. A QP should have a minimum of five years experience relevant to the style of mineralisation and type of deposit under consideration.

A mineral resource is a concentration (or occurrence) of material of economic interest in or on the earth's crust in such form, quality and quantity that there are, in the opinion of the QP, reasonable and realistic prospects for eventual economic extraction.

The definitions of each of the reserves and resource categories can be found under Item 19(f).

Resource block estimation

To further clarify the distribution of the resources declared under Item 19, it is useful to geographically apply the resource results to the geometry of the deposit.

In this regard the structural model for the project area is shown in Diagrams 9(a) and 9(b). The structure then allows for specific structurally related blocks – see Diagrams 10(a) and 10(b) – to be allocated a resource estimate.

In delineating the structural blocks used for the resource evaluation, only major structure was considered.

ITEM 21: INTERPRETATION AND CONCLUSIONS

Results

A mineral resource estimate has been calculated for the Merensky Reef and UG2 Reef from available borehole information and in both instances is classified as Inferred and/or Indicated or Measured Mineral Resources. The Merensky Reef was divided into two distinct domains based on facies with specific lithological and mineralised characteristics.

Interpretation of the geological model

The stratigraphy of the project area is well understood and specific stratigraphic units could be identified in the borehole core. The Merensky Reef and UG2 Reef units could be recognised in the core and are correlatable across the project area. It was possible to interpret major structural features from the borehole intersections as well as from geophysical information.

Evaluation technique

The evaluation of the project was done using best practices. Simple kriging was selected as the best estimate for the specific borehole distribution. Change of support (SMU blocks) was considered for the initial large estimated parent blocks with specific cut-off grades. The resource is classified as an Inferred, Indicated and Measured Mineral Resource and with additional data could result in grade and variance relationship changes and improvements.

Reliability of the data

The data was specifically inspected by the relevant qualified persons and found to be reliable and consistent.

Strengths and weaknesses with respect to the data

The QA&QC process is of a high standard and applies to the full audit trail from field data to resource modelling. The data have been found to be accurate, consistent and well structured. The system of support for the digital data by paper originals and chain-of-custody and drilling records is well developed. Additional geotechnical work will, however, be required to assess mineability.

Objectives of adherence to the scope of study

The intention of this phase of the work programme was to collect sufficient data to be able to confidently raise the estimate of resource. This has been achieved and thus the objectives of the programme have been met.

ITEM 22: RECOMMENDATIONS

Further work required

The current mineral resource is classified partly as Indicated and partly as Measured, with additional resources classified as Inferred.

For the resource categories (Inferred and Indicated) to be potentially upgradeable, infill drilling needs to be done. After completion of the drilling and the subsequent QA&QC process, the additional data will be incorporated into the current model as presented in this document.

Objectives to be achieved in future work programmes

The objectives in the immediate future will be to confirm the potential for upgrading of the mineral resource and to provide a basis for increased confidence, as well as increasing the size of resources in the Indicated and Measured categories. The Pre-feasibility study allows for the engineering and economic evaluation whilst drilling continues as per the recommendation.

The infill-drilling phase will include at least 30 additional boreholes. Eight of these will be drilled with a specific view to upgrading current Inferred to Indicated Resources where potential mining is expected; and at least 22 boreholes will be aimed at reclassifying the current Indicated to Measured Resources, especially in the areas deemed to be in start-up sections of the potential mine.

Detailed future work programmes

To achieve the above-named objectives, the additional drilling will need to be done on a 250m x 250m grid and in some instances on a 125m x 125m grid. Geostatistical parameters based on the modelled variograms indicate that a range of 200m suffices for purposes of upgrading the resource classification. The following table summarises the proposed drilling programme for Project 1.

No. of boreholes	Average Depth	Total inclusive cost/metre	Total metres (plus deflection drilling)	Rate of drilling	Total cost
30	500m	R550	19,500	30 days	R10.73 million

Declaration by QP with respect to the project's warranting further work

It was recommended that additional infill drilling be done for both the Merensky Reef and UG2 Reef. It is further recommended that the Pre-feasibility work be continued while the drilling programme advances.

At the time of the compilation of this report, the above-mentioned drilling programme had been completed up to WBJV156 with 81,883 the total metres drilled and 20,736 the total number of samples. After passing the rigorous QA&QC process, these boreholes will be added to the database.

ITEM 23: REFERENCES

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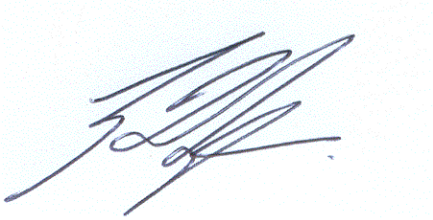
ITEM 24: DATE

The date of this report is 15 January 2007.



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GI Cunningham

B E (Chemical) . MSAIMM, Pr Eng

ITEM 25: ADDITIONAL REQUIREMENTS ON DEVELOPMENT AND PRODUCTION

During this Pre-Feasibility Study phase of the WBJV Project 1 evaluation, a number of access options have been considered, as well a number of tonnage profiles. These scenarios are:

- Processing 140,000 tons per month - vertical shaft system with a decline
- Processing 140,000 tons per month - vertical shaft system
- Processing 120,000 tons per month - vertical shaft system with a decline
- Processing 120,000 tons per month - vertical shaft system
- Processing 160,000 tons per month - vertical shaft system with a decline
- Processing 160,000 tons per month - vertical shaft system
- Processing 200,000 tons per month - vertical shaft system with a decline
- Processing 200,000 tons per month - vertical shaft system

Initially the Merensky Reef alone was to be recovered, but as the UG2 Reef moved into the indicated category, it was apparent that the mine will firstly recover the Merensky and utilizing this infrastructure, develop the lower-grade UG2 resource.

Each of these options was reviewed firstly according to the ability of the deposit to supply sufficient quantity of reef, secondly according to the economic returns applicable to the resource.

As the geological model became more defined and the underlying structural model was developed, it was apparent that tonnages in excess of 160,000 tons per month could not be achieved. The near-surface mining through the declines did not add any significant tonnage to the project due to the limited accessible resource.

The options which gave the best economic returns were those associated with the 140,000 tons per month case.

The project team is of the opinion that the best option for consideration of the WBJV Project 1 will be a vertical shaft from surface to 712m below surface with 7 working levels spaced at 60m intervals. Whilst the decline system may detract from the project value, it will be further investigated in the Feasibility Study, subject to available processing capacity within the area.

The project team has spent considerably more time on the mine planning associated with the geological model than the associated engineering of the mine. This has resulted in a project with very good understanding of the mining, the mining layout and the most appropriate access option. The engineering of the design will be further developed during the Feasibility study.

To date, the project team has not considered the added value of products other than the normal platinum-group metals in the reef. The possibility of chromite recovery has been rejected as the first reef will be Merensky and it is not expected that the required grade of chromite will be achieved from this source. There is a possibility that when UG2 is being processed, the chromite recovery option may be viable, but this option is excluded from the project model at this time.

The potential upgrading of the reef by the use of a dense medium cyclone or similar device through the process of dense medium separation (DMS) has not been considered for the WBJV Project 1, as there are little to no bands of internal waste. During production, this could be reviewed and there may be a future possibility for DMS technology to be employed. This has not been considered for the WBJV Project 1 Pre-feasibility study.

The project is based on well-proven and reliable methodologies and equipment and there will be no new technology applied. There will be no second hand plant installed and only new equipment has been costed for this project.

The decline option considers toll milling for initial production with moderate tonnages. These amounts are not in excess of the amounts that might be of interest to an existing mill and this option is considered in the WBJV pro-forma agreements.

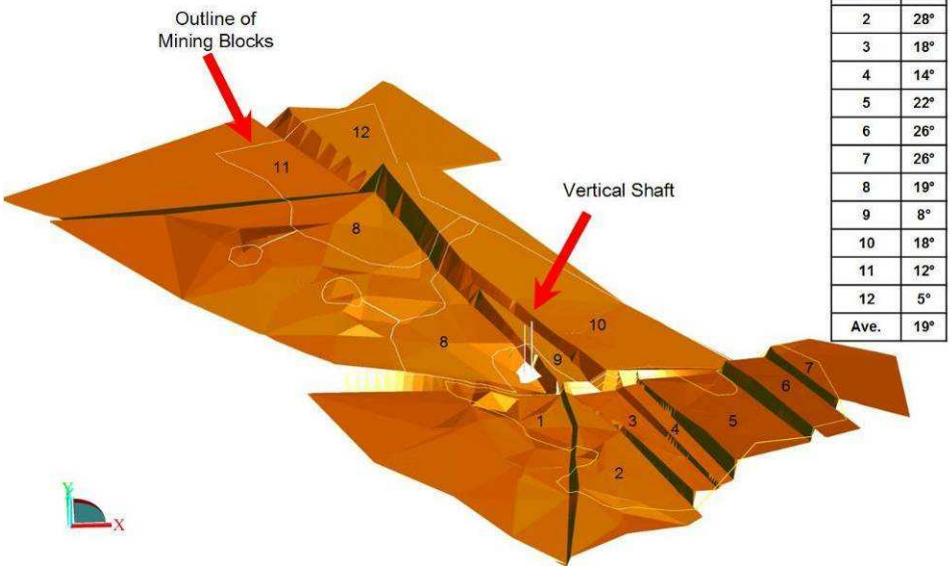
Item 25 (a): Mining Operations

Mine Design Criteria

The deposit being considered for mining consists of typical BIC tabular and planar Merensky and UG2 Reefs which are separated by a stratigraphic distance varying from about 10–70m. Given this distance between the two reefs, the proposed design for the mine is aimed initially at extracting the higher-grade Merensky Reef, but the design also makes provision for the eventual extraction of profitable UG2 Reef. The design has taken into consideration certain rock mechanics requirements necessary to prevent the profitable UG2 Reef from being sterilised. It is estimated that over the life of the mine the Merensky Reef will yield three times as much PGM as the UG2 Reef, and the Merensky Reef thus remains the primary focus of the mine design.

The Merensky Reef has the following characteristics which influence mine design:

- variable dips over the property which range from 5 to 28 degrees (average 19 degrees);
- mining width of 126cm;
- significant faults and dykes which intersect the Merensky Reef and subdivide the deposit into a number of discrete mining blocks, each of which requires access development at and on different mining levels (see Diagram 25).



Block No	Dip
1	19°
2	28°
3	18°
4	14°
5	22°
6	26°
7	26°
8	19°
9	8°
10	18°
11	12°
12	5°
Ave.	19°

Diagram 25: Merensky Reef structural blocks (isometric view looking north).

Other significant geological considerations requiring specific and appropriate mine design include the following:

- The initial occurrence of payable reef is at 130m below surface. The payable reef then extends to a final depth of 630m below surface.
- There are no near-surface or outcropping reefs of sufficient width and geometry to provide an opportunity for opencast mining.
- The fact that the reefs become steeper near to surface is not conducive to mechanised mining.

Design description

Overall design

The need for overall access to both the Merensky Reef – Diagram 26 Appendix A– and the UG2 Reef – Diagram 27 Appendix A – was taken into consideration when siting the main vertical shaft. In Diagram 26 and 27 the main shaft is situated at the centre of gravity of the deposit with respect to both the eastern and western limits of the mineable ground, and midway between the economic upper areas and the lower property boundary. The main shaft was positioned in an unpay block of ground so as to minimise the loss of payable ground. The ground being considered for mining has to deliver a cut-grade of greater than 2g/t. This ground falls into the resource categories of Measured or Indicated.

As regards the UG2 Reef, the layout design has been aimed at providing efficient and effective access to blocks of ground with the potential to yield 3.5g/t (blocks falling into the Measured or Indicated Resource

categories). Isolated blocks need to be considered in detail and in relation to the viability and timing of the Merensky Reef mining sequence.

Stope design

Given the geological characteristics of the deposit under consideration (described in Items 9–11), as well as the experience of mines adjacent to this property, it has been decided that a conventional breast-panel scattered-stoping layout would be the most suitable mining method as illustrated below.

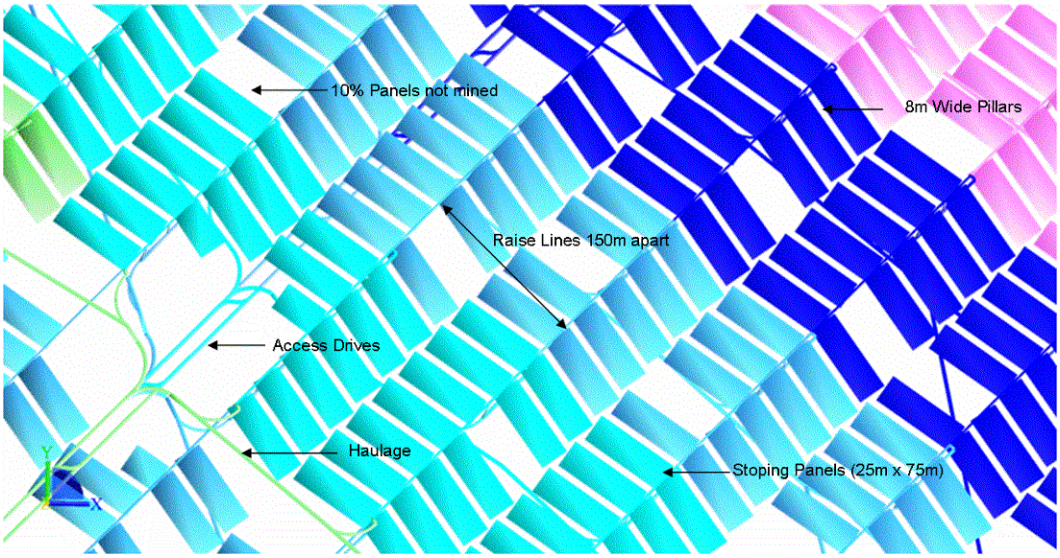


Diagram 28: General breast-mining stope layout.

The methodology is as follows:

- Stopes are established every 150m along strike with an inter-level vertical distance of 60m resulting in an average back length of 184m.
- The breast panel face length is 25m; supported by 8m dip pillars (see also the design as described under *Rock-engineering considerations*).
- The broken rock is transported to the strike gully by scraper winch and in a similar manner along strike to the centre gully.
- Within the centre gully, the rock is either tipped directly into stope orepasses or scraped down-dip to the next orepass. These orepasses then deliver broken rock onto the strike haulages, which are placed 30m below the reef horizon.
- Within the stopes, and in between the 8m supporting pillars, additional 200mm sticks each supporting 3m ^{2} are installed as a final support mechanism.

Development design

The infrastructure (see Diagram 29) appropriate for servicing of stoping faces and the removal of broken rock is as follows:

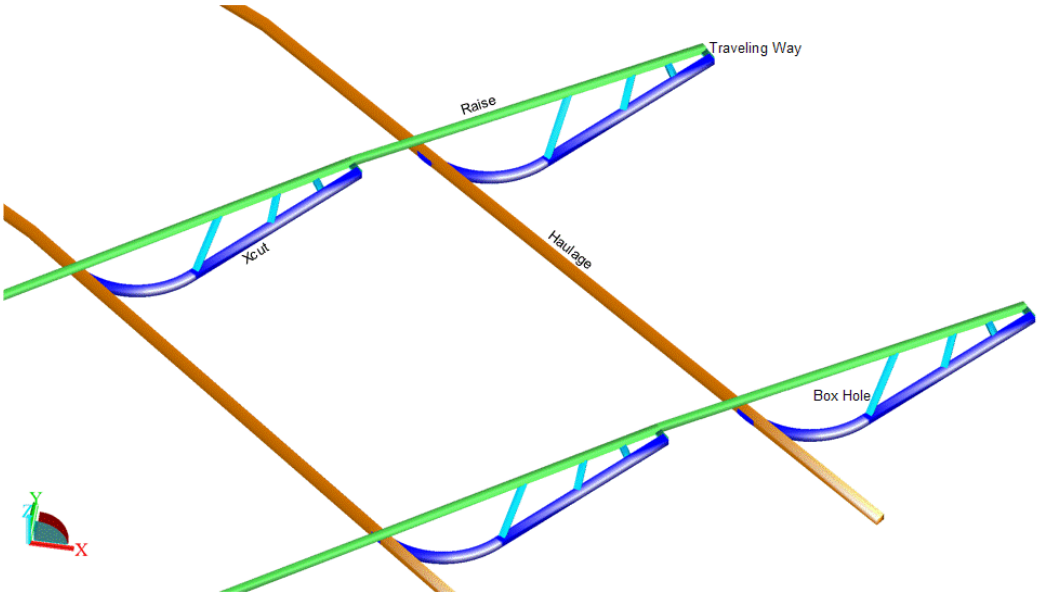


Diagram 29: Block development layout.

- To access the deposit, access drives developed perpendicular to strike, and supported by return airways, have been designed for each level. It is possible to develop these drives at a rate of 80m per month. Simultaneous development of the six main levels will facilitate the planned production build-up.
- Strike haulages are carried 30m into the footwall of the reef in accordance with rock mechanics specifications.
- The level spacing chosen for the vertical shaft options is 60m for all levels so as to optimise back length for reef dips averaging 19 degrees. The result is an average back length of 184m per raise line. An overall panel length of 33m (face plus pillar) means that there will be 11 panels in one average stope. Of the 11 panels, one is permanently excluded from the planning process to compensate for unaccounted-for geological losses. Of the remaining ten panels, five would be considered available at any given time.
- If each working panel advances at a rate of 11 metres per month, the raise line will produce 1,375m ^{2} (5m x 25m x 11m). This equates to approximately 5,250 stope tons per producing raise line.
- The delivery of rock from any half-level (single mining entity or production unit) infrastructure has been designed to cater for two producing raise lines, one vamping line and one stope-preparation line as well as the necessary development in the area.

- Ventilation requirements for each production unit, as well as the mine as a whole, are based on the above parameters. These ventilation requirements play a significant part in determining downcast and upcast shaft dimensions.
- Each production unit is served by a single strike haulage which serves as the intake airway and service way. This haulage is situated 30m below the reef elevation.
- Over the life of mine the actual ratio of reef tons to waste tons is 4.8:1.
- All stope block development is restricted to an initial rate of 30m per month per development end, and at steady state the rate of development is reduced to 22m per month (a mining month is 25 working days). This creates a balance between generation and depletion of ore reserves, using acceptable production parameters.
- The development drilling is done by hand-held rockdrill and air-leg drilling equipment.
- The cleaning of flat development-end broken waste is done by trackbound rocker shovels.
- Consideration has been given to mechanising the development operations. The main reason for not pursuing mechanised development mining methods is the negative effect this would have on the overall mine ventilation design.

The above production parameters form the basis of the design and evaluation of a vertical shaft system for access to the reef and servicing of the mining operation. The system is depicted in Diagram 30.

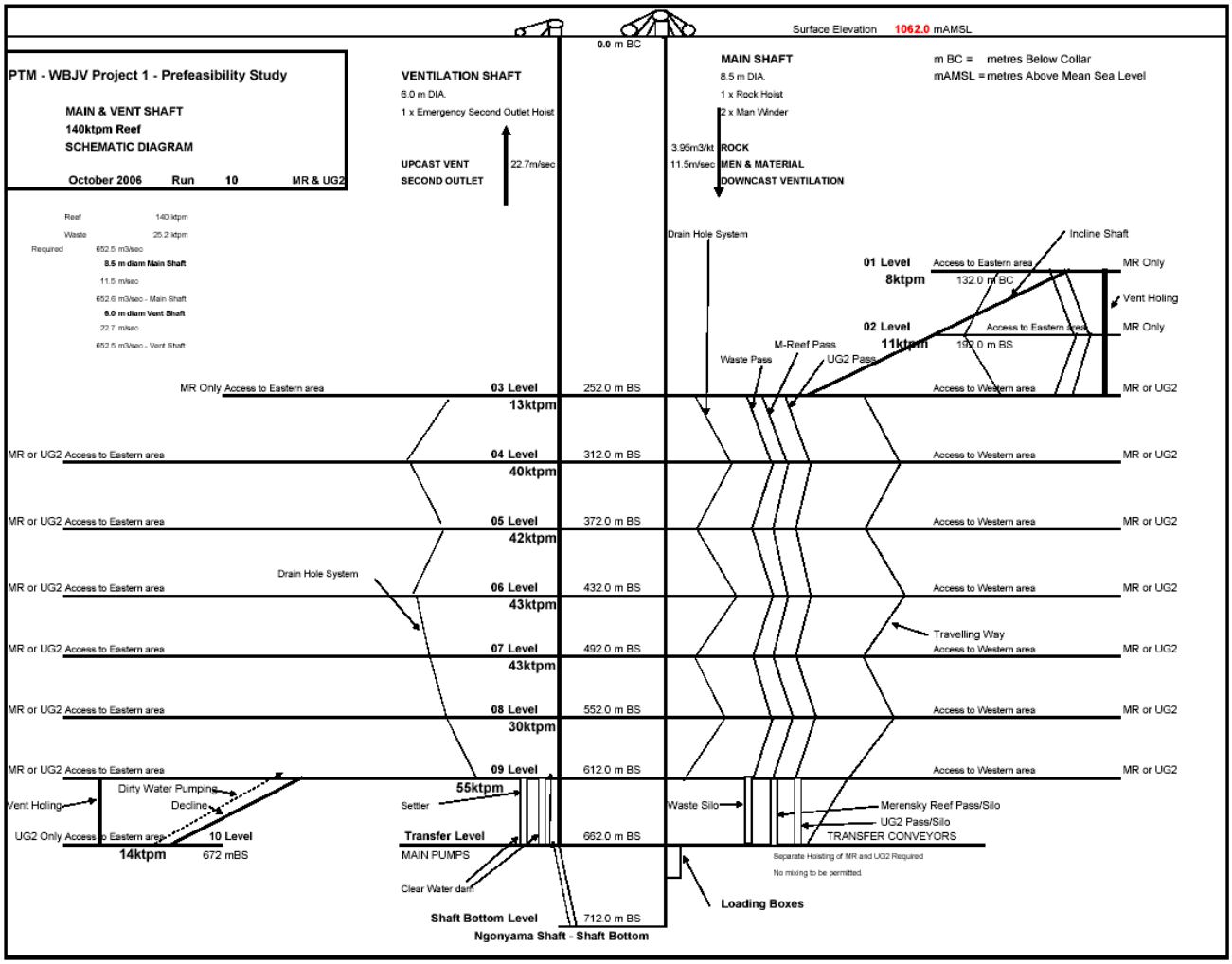


Diagram 30: Vertical shaft arrangement.

The construction of this system forms a crucial part of the mine's build-up to full and steady-state production. It is estimated that the shaft system will take four years to complete.

Production schedule

Design methodology

This mine design model made use of commercially available software traded as Surpac Vision. A scaleable system with numerous design options has a 3D immersion environment taking advantage of the real-world perspective from all angles. The visual design of all underground features has been made possible by mining-specific CAD tools in a 3D environment. From a design centreline, a 3D model of the underground excavations was created, using the solids modelling tool. A powerful scheduler known as Mineshed was then used to schedule the mining by applying practical mining constraints, real-time calendar, benchmarked rates and resources models. The Surpac Vision block evaluation reporter was then used to read the Datamine block model information into the production schedule. A database with the scheduled production periods and the

associated volumes, values and mining widths was generated, allowing for a production profile output and analysis.

Flow-of-ore

Based on the best cut prediction, the most likely stoping width is adopted for modelling purposes. At this stoping width, the deposit has an in situ grade of 5.68g/t. This grade is diminished (diluted) by measurable factors such as gully dilution and reef development.

To model the expected unaccounted for dilution, such as falls of ground, incorrect tramming and waste rock excavations that cannot be separated from the ore, a dilution factor of 5%, based on similar ore bodies and mine systems is used.

In addition to the dilution, some of the metal measured in situ is never recovered. This under recovery is basically split into losses accounted for (metallurgical recoveries) and unaccounted losses usually in the mining operation due to imperfect mining practice. This unaccounted for loss is represented by the mine call factor of 97.50%.

For the WBJV Project 1, applying the known dilution factors and the Mine Call Factor, results in a plant feed of 4.86 g/t from a stope grade of 5.68g/t.

Applying the metallurgical recovery factor of 87.5% concentrator for Merensky Reef and 82.5% for UG2 Reef, the resultant net estimated recovered grade is 4.19 g/t on both reefs combined.

WBJV - PROJECT 1

STOPING		
Data	Units	Grand Total
Face Advance	m	5.1
Square Metres	m²	4,090,943
Stope Width	cm	134
Tons Broken	t	17,998,048
Content	g	102,299,119
Face Value	g/t	5.68
Gully Sundries	t	788,787
Tons To Mill	t	18,786,835

REEF DEVELOPMENT		
Data	Units	Grand Total
Reef Metres	m	54,666
Square Metres	m²	98,398
Tons Broken	t	804,624
Content	g	2,433,155
Face Value	g/t	3.02
Tons To Mill	t	804,624

PLANT			
Data	%	Units	Grand Total
Tons To Mill		t	19,591,460
Dilution Tons	5%	t	1,031,129
Tons Milled		t	20,622,589
Total Received		kg	104,732
MCF	97.50%	kg	102,114
MCF		g/t	4.95
Concentrator	MR 87.5% & UG2 82.5%	kg	88,004
Concentrator		g/t	4.27
Total Oz's		oz	2,829,383

Table 7: Methodology for flow-of-ore.

See also Table 10(a), (b) and (c) – Appendix A – for additional detail on the flow-of-ore parameters.

Design results

The combination of stope, block and access development plans has resulted in the following Merensky Reef extraction plan (Diagram 31), which starts in Year Five of the project.

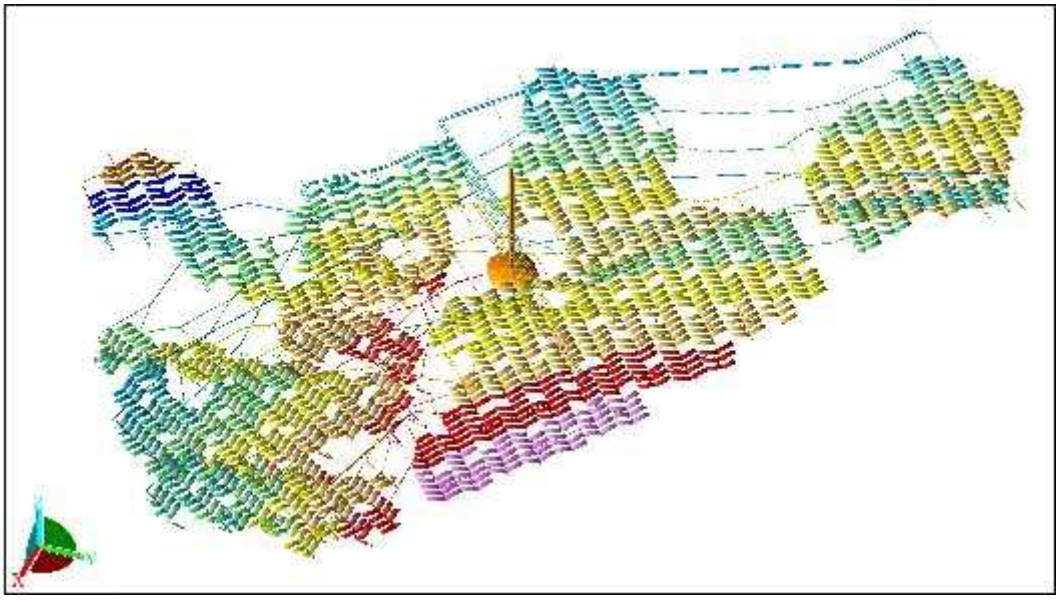


Diagram 31: Merensky Reef mining layout.

With limited overlapping on individual levels, the mining of the UG2 Reef essentially follows the Merensky Reef extraction, utilising the Merensky Reef footwall infrastructure wherever possible (see Diagram 32).

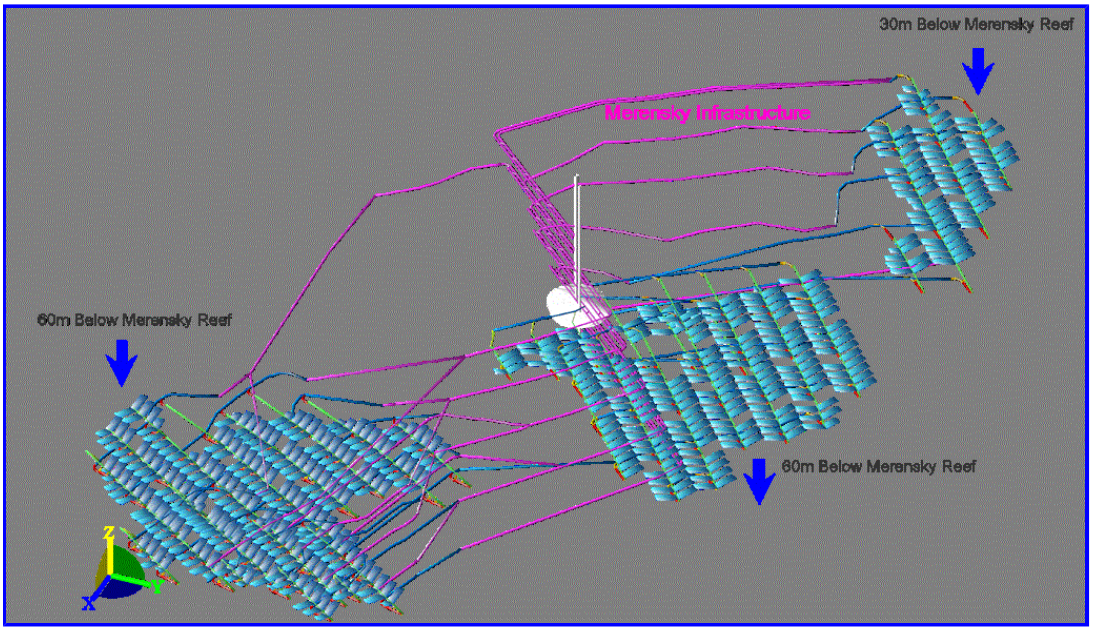
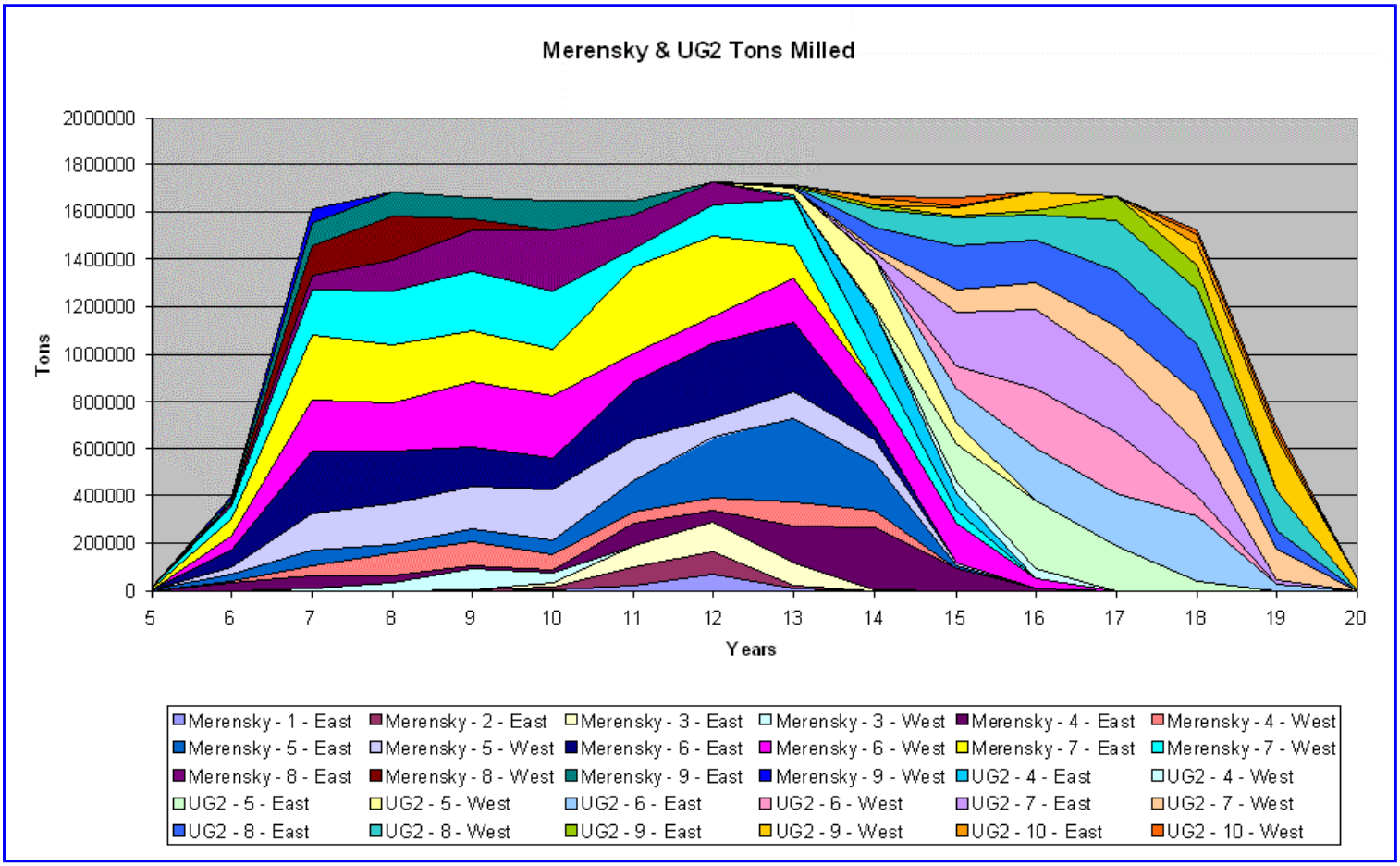
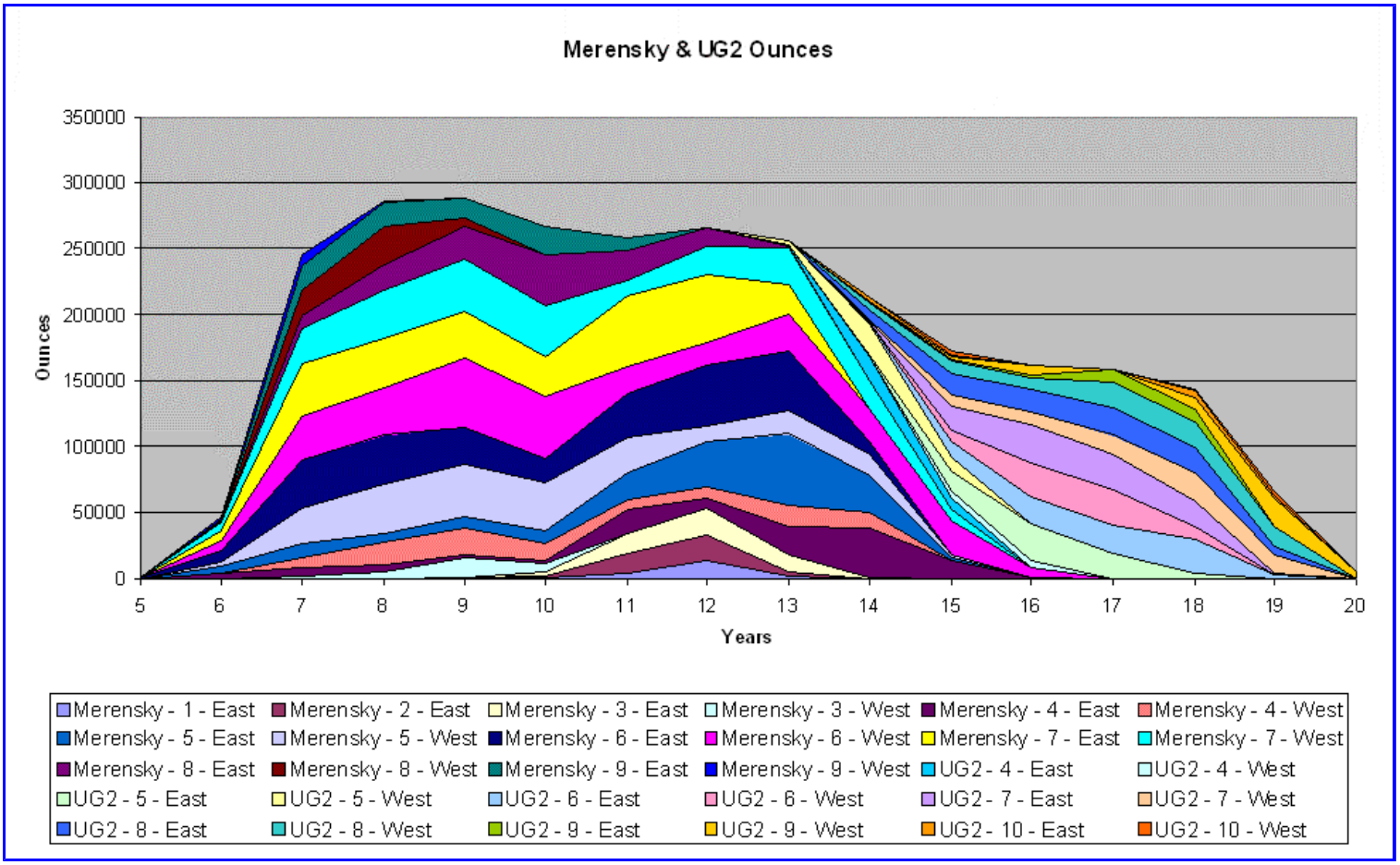


Diagram 32: UG2 Reef mining layout.

Graph 16 shows the yearly production at 140,000 tons per month milled per level for the Merensky and UG2 Reefs and Graph 17 the ounces milled per level per year.

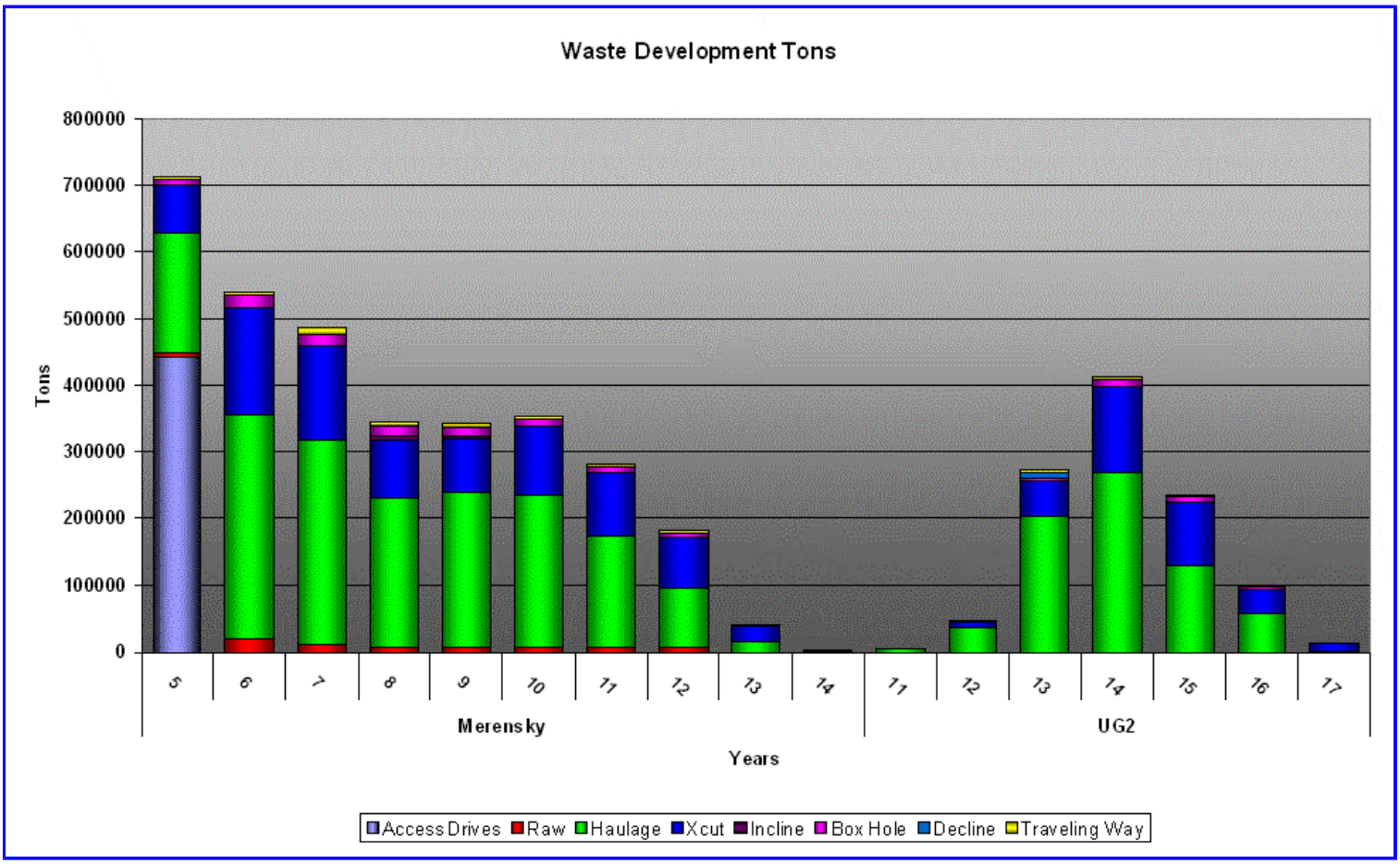


Graph 16: Combined Reef production profile in tons.



Graph 17: Merensky Reef and UG2 Reef ounce profile.

See Graph 18 below (and Table 11 – Appendix A) for the waste tonnage build-up in order to access the Merensky and UG2 Reefs.



Graph 18: Waste tonnage profile (see also Table 11 – Appendix A).

Reconciliation of ore extraction

Table 12 (Appendix A) shows the reconciliation of square metres as shown in the resource statement against the square metres that are eventually mined and report to the mill.

Merensky Reef

In the case of the Merensky Reef, 10,762,396m ^{2} have been defined as the potential mining area. The Merensky Reef resource is made up of contact reef and pegmatoidal feldspathic pyroxenite and is spread over the resource categories of Measured, Indicated and Inferred. Of the total area, 6.8% is taken up by iron-replacement, 7.0% remains in the Inferred category (the level of confidence in this category does not allow for it to be scheduled), 37.3% falls below a 2g/t cut-off and are currently not considered to be mined economically, 1% of the blocks of ground are isolated and uneconomic to develop, 7.9% of the ground is taken up by fault losses, 4% of the ground is intruded by dykes and other intrusives, 0.7% for raises, 8.4% for pillars and 0.3% of the panels have been removed from the plans to accommodate for potential and unforeseen minor in-stope faulting. Thus 26.6% of the square metres on the Merensky Reef is mined and taken to the mill.

UG2 Reef

In the case of the UG2, 9,004,219m ^{2} have been defined as the potential mining area. Like the Merensky Reef, the UG2 has been declared in Measured, Indicated and Inferred Resource categories. Some 28.5% of the UG2

remains in the Inferred category and owing to the level of confidence in this category does not allow for it to be scheduled. The iron-replacement area amounts to 8.5% and was taken out of the mine plan; and 16.6% of the deposit falls below 3.5g/t which is currently not considered as economical to mine. Isolated blocks which would necessitate excessive mining infrastructure amount to 15.3% of the deposit and they were also left out of the mining plan. Faults and dykes that have been identified amount to 8.3% and 4.1% of the area respectively and are excluded from the mining plan. A further 0.2% of the area was excluded to account for potential unforeseen faulting and 4.4% of the area will remain for stability pillars with 0.4% for raises. Thus the total amount of rock mined on the UG2 horizon and sent to the mill is in the order of 13.8% of the total available.

Alternative deposit access – decline option

Ways of gaining quicker access to the shallower parts of the mine have been investigated (see Diagram 33 and Diagram 34). Apart from the obvious advantage of reaching the resource sooner, its early exposure would give a better indication of the complexity of the geology and modifications could be made to the overall mine design.

The findings were inter alia as follows:

- Access to the ore body and all operations in the footwall of the reef would be done using rubber tyred diesel powered equipment.
- Due to the Merensky Reef dimensions, on reef mining would still be by conventional, labour intensive means.
- The introduction of diesel powered rubber tyred machinery into the workplace would significantly change the ventilation requirements – and in this case would require additional trackless development.
- The dip of the reef in the shallow area is steep and additional development would be needed for the ideal machine slope of 9 degrees to be maintained.
- It would be possible to produce ore within 18 months but the tonnage would be limited.

Additional work in this regard should be considered. In the present circumstances, however, the advantage to be gained from the surface decline appears limited and the method needs to be seen as a supplement to the vertical shaft system rather than as an alternative to it.

The ultimate choice of design is dependent on financial and risk factors. The financial viability of both options is discussed under Item 25(h).

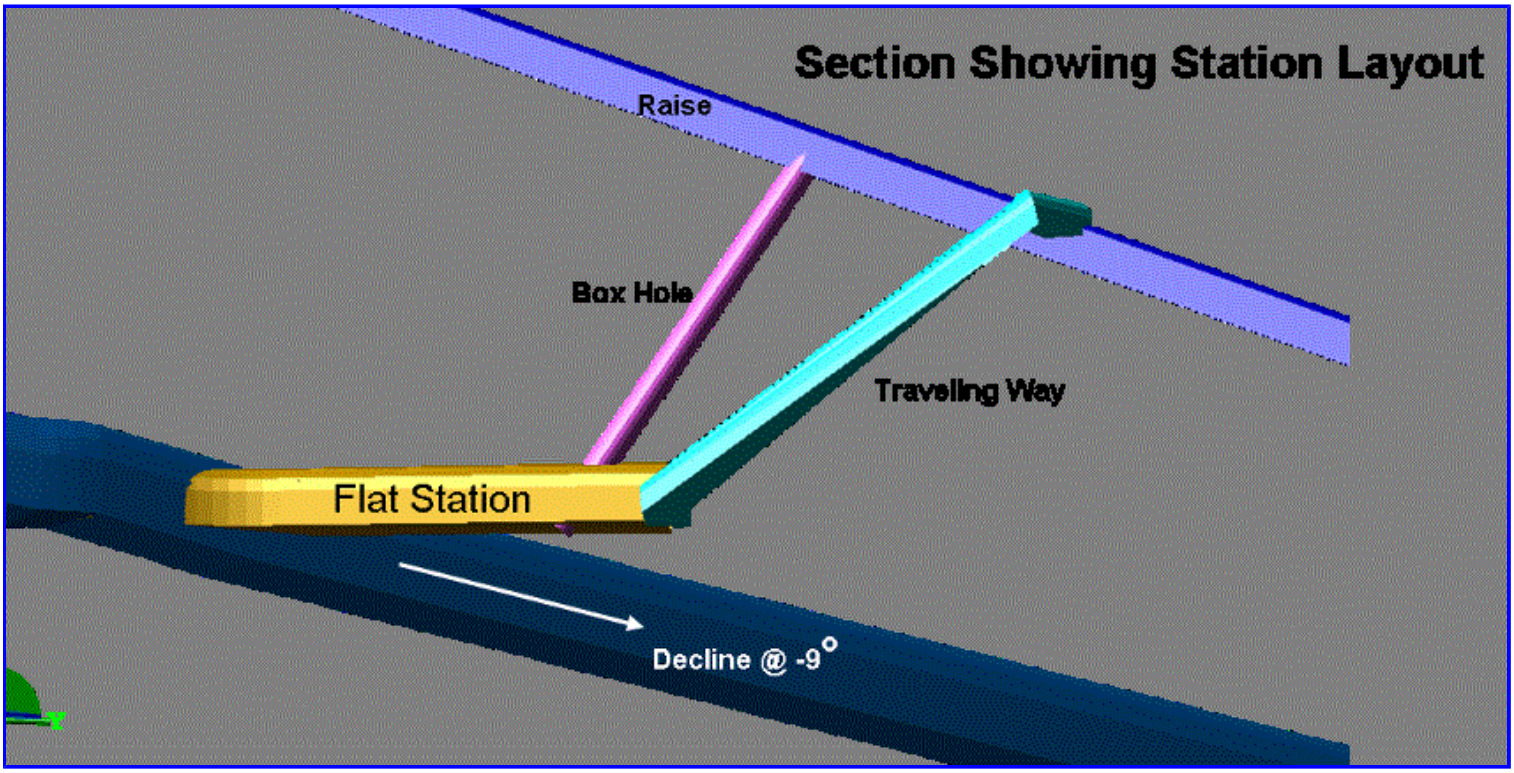


Diagram 33: Arrangement for the decline access.

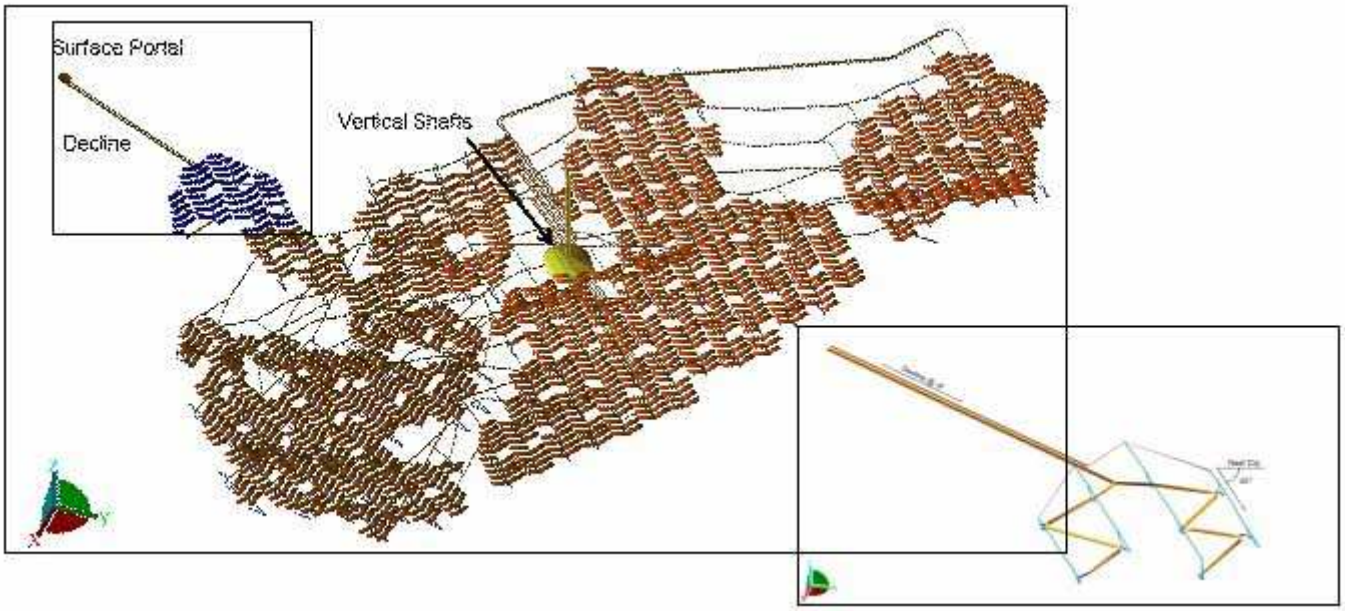


Diagram 34: Decline access to shallower resources.

Manpower

The mining methodology envisaged for this type of deposit is common throughout the South African platinum industry. Similar systems of staffing and skills selection are also used in the country's gold mines. The reason for this is the relatively thin and tabular nature of the deposit.

Manpower categories would typically include the following:

- **Developers** particularly skilled in flat-end development (3m x 3-4m) tunnelling, raise development (3m x 2m) and box-holing (2m x 2m) development.
- **Stopers** skilled in narrow reef mining methods
- **Horizontal logistics** personnel trained and skilled in the logistic of removing broken ore and the transportation of men and materials to the working areas.

- **Shaft logistics** personnel trained and skilled in the running of vertical and incline shaft systems. Necessary skills would include operational (hoist drivers and operators), and engineering (electrical and mechanical).
- **Services** to the mining and logistics would include geological, survey, evaluation, ventilation, maintenance (electrical and mechanical), human resources, training, security, financial, stores and salvage personnel.
- **Management** personnel would be employed to cover the major disciplines of mining, engineering, financial, geological, survey, ventilation, rock engineering, health and safety, human resources and industrial relations.

Staffing requirements are based on the maximum production achieved for stoping, which occurs in Year Five after shaft commissioning and development (development is in Year Three after shaft commissioning). Using current productivity parameters for each activity on similar operations one arrives at the grand total of 3,397 on-mine employees. At a milling rate of 140,000 tons per month, the overall ratio of tons milled per total number of employees is 41.

Rock engineering considerations

Four primary geotechnical aspects are dealt with in this section of the report:

- the regional stability of the proposed mine;
- the local stability of stoping excavations;
- the stability of access tunnels;
- issues associated with multi-reef mining.

1. Regional stability

Bracket pillars

Stoping will take place through minor faults and right up to major faults, without leaving bracket pillars, provided that ground adjacent to such features is not friable and allows mining to take place safely. Actual mining experience on the WBJV Project 1 property will be required to test this assumption.

Shaft pillar design

Stoping will not take place less than 100m from the vertical shafts. These pillars cater for both a hoisting shaft and a ventilation shaft, and as a result are semi-elliptical, measuring some 200m x 260m. The shafts are situated within an even larger area of unpay ground, so the specification of a shaft pillar size is somewhat academic. Of more immediate concern is the fact that the shaft pillars are bounded by a number of major faults. The nature of these faults and their potential impact on the shafts need to be evaluated.

Panel pillar design

Using standard industry-accepted formulae and adopting rockmass strength criteria appropriate to the project, optimal panel pillar dimensions and stope extraction ratios have been calculated at various depth ranges to ensure geotechnical stability down to the maximum depth of mining operations envisaged in this study. A summary of the results of these calculations is given in Table 8.

Extraction ratios suggested by these calculations range from about 93% close to surface to about 66% at a depth of 600m below surface (see Table 8). Pillar widths can be maintained at 8m down to a depth of 500m, after which they increase progressively to 10m at 600m below surface. Similarly, panel spans are reduced to less than 25m for depths exceeding 350m below surface in order to maintain safety factors above the 1.5 criterion.

Table 8: Summary of theoretical panel pillar design criteria.

Depth (mbs)	**Pillar Height** (m)	**Holing Width** (m)	**Pillar Length** (m)	**Pillar Width** (m)	**Panel Span** (m)	**Panel Length** (m)	**Safety Factor**	**Percnt Extract** (%)
10	1.3	3	5	3	25	28	10.87	93.3
50	1.3	3	5	3	25	28	2.17	93.3
100	1.3	3	5	6	25	31	2.37	87.9
150	1.3	3	5	8	25	33	2.10	84.8
200	1.3	3	5	8	25	33	1.58	84.8
250	1.3	3	10	8	25	33	1.86	81.3
300	1.3	3	10	8	25	33	1.55	81.3
350	1.3	3	15	8	25	33	1.56	79.8
400	1.3	3	15	8	20	28	1.61	76.2
450	1.3	3	15	8	15	23	1.74	71.0
500	1.3	3	15	8	15	23	1.57	71.0
550	1.3	3	15	9	15	24	1.60	68.7
600	1.3	3	15	10	15	25	1.61	66.7

In order to simplify and streamline (see Table 9) the many production modelling exercises that were associated with this pre-feasibility study, an average support pillar extraction ratio of just over 75% was accepted for the entire deposit down to 600m below surface. Table 9 also shows how the factor of safety is modified by this rationalisation of the extraction ratio and simplification of the mining geometry.

Table 9: Summary of the rationalised panel pillar design.

Depth (mbs)	**Pillar Height** (m)	**Holing Width** (m)	**Pillar Length** (m)	**Pillar Width** (m)	**Panel Span** (m)	**Panel Length** (m)	**Safety Factor**	**Percnt Extract** (%)
10	1.3	0	75	8	25	33	77.26	75.7
50	1.3	0	75	8	25	33	15.45	75.7
100	1.3	0	75	8	25	33	7.73	75.7
150	1.3	0	75	8	25	33	5.15	75.7
200	1.3	0	75	8	25	33	3.86	75.7
250	1.3	0	75	8	25	33	3.09	75.7
300	1.3	0	75	8	25	33	2.58	75.7
350	1.3	0	75	8	25	33	2.21	75.7
400	1.3	0	75	8	25	33	1.93	75.7
450	1.3	0	75	8	25	33	1.72	75.7
500	1.3	0	75	8	25	33	1.55	75.7
550	1.3	0	75	8	25	33	1.40	75.7
600	1.3	0	75	8	25	33	1.29	75.7

At shallower depths, the simplification results in a conservative over-design of the panel pillars, but this can be offset to some extent against the optimistic design associated with the levels 500m below surface.

Surface effects

Allowance has been made in the panel pillar design calculations for the protection of the general surface topography and any general surface installations by applying an increased safety factor of 2.0 for mining depths down to 150m below surface. Actual mining is in any event currently planned to commence only some 130m below surface. The adequacy of these measures may, however, need to be tested in the event of mining below the public infrastructure such as provincial roads, power lines and reservoirs, and sensitive mine installations such as processing and tailings facilities.

2. Local stope stability

Inner-pillar spans

Inner-pillar stope spans have been limited to not more than 25m for both the Merensky and the UG2 Reef horizons. In terms of the current understanding of the deposit, this conservative criterion is expected to be easily met in practice, and it has been incorporated into the pillar design calculations outlined above.

Stope support

An estimate of the required support demand has been made on the basis of a preliminary visual inspection of borehole core and taking note of conditions in the neighbouring BRPM. It is estimated that internal stope

support will have to be such as to resist a dead weight equivalent to at least 1.5m height of hanging wall rock. This equates to a load-bearing capacity of approximately 50kN/m ^{2} .

Because this stope support will need to be stiff and preferably proactive, large-diameter pre-stressed elongate units will be used. On the basis a conservatively calculated support density of 3m ^{2} per elongate (2.0m dip spacing and 1.5m strike spacing), an internal stope support material cost of approximately R106/m ^{2} (R24/ton) is estimated.

Temporary support requirements at the stope face will be provided for by the setting of one or two rows of mechanical props, with or without the addition of load-spreaders.

The problem of localised areas needing additional support may be addressed by either increasing the elongate support density, installing rock bolts as an additional or secondary support measure or, where conditions so dictate, leaving small internal pillars. In situations of excessive ride in more steeply-dipping stoping environments, stiff grout-based or composite concrete packs may be needed to replace elongate units that are liable to buckle in such circumstances.

Compared with what is required on the Merensky horizon, operations on the UG2 horizon are likely to need a significantly higher level of stope support in order to effectively deal with the more laminated and less competent nature of the immediate UG2 hanging wall. In addition, geotechnical evaluation and classification of the available borehole cores would seem to indicate a variety of weathering effects, and a wide range of hanging wall conditions and competencies may be expected on both the Merensky and the UG2 horizons.

3. Stability of access tunnels

Middling distance

The mining layout contemplated in this study comprises horizontal access tunnels sited 30m into the footwall of the reef throughout. This is well within the limit stipulated by geotechnical modelling work, and the middling distance could thus be significantly reduced with little or no adverse effect, particularly in the shallower regions of the mine.

Tunnel support

Evaluation of the rockmass quality on the basis of borehole information suggests that not only will access tunnels be situated in a variety of rock types, but also the rock types themselves are likely to exhibit a wide range of quality and competence. Support demand will thus range across a broad spectrum from little or no support to intense support. This is likely to be exacerbated by high (horizontal) stress considerations at the deeper levels.

Evaluation of the rockmass quality on the basis of borehole information suggests that not only will access tunnels be situated in a variety of rock types, but also the rock types themselves are likely to exhibit a wide range of quality and competence. Support demand will thus range across a broad spectrum from little or no support to intense support. This is likely to be exacerbated by high (horizontal) stress considerations at the deeper levels.

To cater for this wide range of tunnel and development support requirements, a system of support categories has been devised which can be applied in estimates of support requirements and costs as more detailed information becomes available. Tunnel support material costs are currently estimated at approximately R2.50/m ^{2} (R0.60/t). Good conditions prevail in 75% of tunnels, moderate conditions in 20% of tunnels, and poor conditions in 5% of tunnels.

4. Multi-reef mining

Stope interaction

Where the separation between the reefs is small (less than 20m), stoping activities on one reef will affect conditions on the other. Matters such as the superimposition of support pillars, the integrity and stability of the hanging wall beam, and the sequence and timing of operations on each of the reefs, will need to be addressed. These inter-reef effects obviously decrease and become less problematic with increasing separation, and are negligible once the separation between the reef horizons exceeds 30m.

Options for the sequence of mining in such circumstances include simultaneous stoping of the two reefs, or primary extraction of either the uppermost horizon or lowermost horizon respectively. Each of these alternatives carries its own constraints, advantages, and disadvantages. These are not necessarily onerous, but must be strictly adhered to if optimal extraction is to be achieved and sterilisation of ore reserves is to be avoided.

Multiple access

It has been assumed for the purposes of this study that separate and independent access layouts will be provided for the Merensky and UG2 reefs respectively. There are two main reasons for this approach. Firstly, access (backward or forward) through the hanging wall of the lowermost stoping horizon is likely to be problematic. Secondly, a cumulatively greater amount of footwall development is required for single-access layouts than for duplicate-access layouts when the horizontal reef separation distance approaches or exceeds the crosscut spacing distance.

Future work

Rockmass conditions

Rockmass conditions that are significantly weaker than those which have been anticipated in the modelling could have a negative impact on numerous critical design features such as pillar sizes, extraction ratios, stope and tunnel support requirements and so on.

Footwall characteristics

The geotechnical characteristics of the footwall succession appear variable, and remain largely unknown because of the termination of most exploration boreholes immediately following the intersection of the reef zones of financial interest.

Potholes

The intensity and frequency of potholing is largely unknown at this stage. The elimination of one in every ten panels from the mining profile provides some allowance for the occurrence of potholes potentially disrupting the reef layering.

Ventilation considerations

The focus of this work is on designing the ventilation infrastructure to mine the Merensky deposit. It is assumed that the UG2 will be accessed where possible, using redundant Merensky infrastructure with minimal need for development.

There will be two vertical shafts, spaced approximately 60m apart, and seven working levels from 3 Level, 252m below surface to 9 Level, 612 m below surface. Although the dip varies considerably, the level spacing will be constant at 60m.

Production units typically consist of four ventilated stopes; two producing, one for stope preparation and one for vamping as well as footwall, crosscut and raise development. At any one time up to 22 units will be mined so as to produce 140,000t of reef monthly.

The mining method will be conventional hand-held drill-and-blast with scraper cleaning and battery operated track bound equipment – no diesel equipment will be applied underground in the vertical shaft option. The decline with vertical shaft takeover option would involve diesel equipment during startup operation and would subsequently require increased ventilation.

Design criteria

Stoping and development

Stope face velocity	0.4m/s
Stope ventilation control to face	10m
Stope air utilisation	80%
Stope width (Merensky)	1.26m
Double-sided breast mining	$30m^3/s$
Preparation	$15m^3/s$
Vamping	$10m^3/s$
Development headings	$12m^3/s$

Note : The preparation volume includes sufficient air to develop one production end (footwall, raise line and crosscut).

Shaft and airway velocities

Downcast shafts	10–12m/s
Upcast shafts	20–23m/s
Intake airways	7–8m/s
Return airways	10–12m/s

Leakage

Footwall drive to worked-out-areas	20%
Primary	10%

Surface ambient conditions

Summer design wet-bulb	22°C
Summer design dry-bulb	30°C

Underground design condition

Average stope reject temperature	27.5°C wet-bulb

Primary ventilation

The mine divides naturally into a number of ventilation zones which vary in complexity and span a number of levels. Diagram 35 below shows the typical ventilation layout for the mine at a specific critical snapshot in time (view manipulated to show the four ventilation zones).

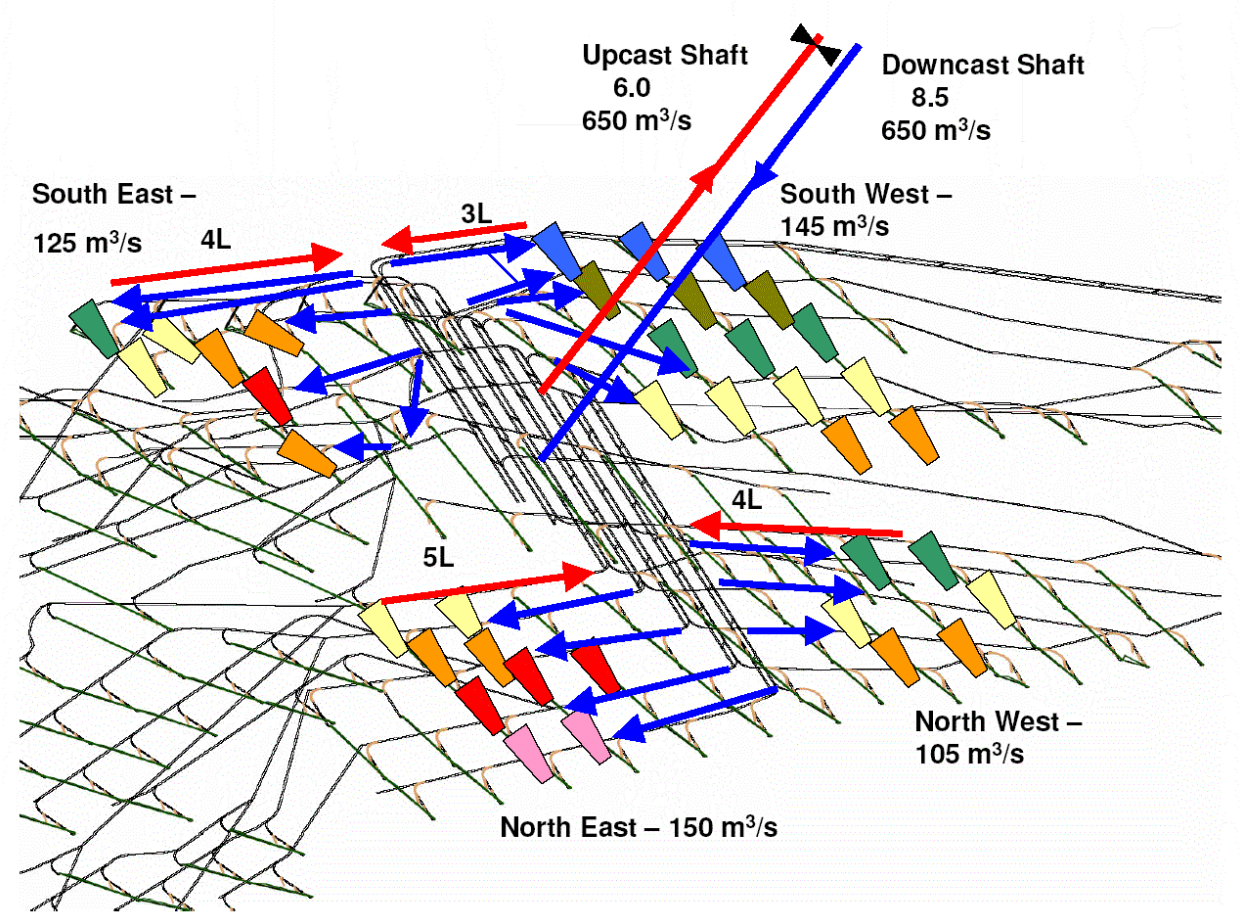


Diagram 35: Ventilation layout – production Year Five.

The proposed strategy uses air in a cascade system to ventilate levels in series. Although most air is reused, some returns through worked-out areas and is replaced with fresh air on the intermediate levels. Typically, after allowing for leakage the strategy will introduce sufficient fresh air on the bottom level to ventilate four raise lines. There is a limit to how often the reuse cycle can be repeated, but for conditions at WBJV Project 1 air can be discarded after ventilating 800m of back length which, depending on the dip, covers up to five levels.

Planning of the reuse strategy will depend on the exact position and sequence of mining and will be an ongoing function of the mine ventilation staff.

A “worst case snapshot” in time, with the maximum number of production units, was used to estimate the air quantities.

Stoping and Merensky development

Total (all ventilation zones)	525m ^{3} /s

Shaft bottom, pump stations, workshops and additional (UG2) development

Estimated	65m ^{3} /s

Primary leakage

Leakage in shaft stations	60m ^{3} /s
Total	**650m ^{3} /s**

Cooling requirements

Some neighbouring deep platinum mines use refrigeration to reduce the high temperatures. Heat is introduced in to the underground workings by surrounding rock, broken rock, autocompression of air, auxiliary fans and other equipment. With regards to Project 1, at the planned depth and virgin rock temperature, ventilation air will be sufficient to remove mine heat and no refrigeration will be required.

Note: the deepest mining level at Project 1 is 612m below surface and the corresponding virgin rock temperature is approximately 35.5 ^{o} C.

The maximum in-stope air reject temperatures will remain below 27.5 C wet-bulb at the stope face velocity of 0.4m/s. This is in accordance with general South African practice. At 27.5 C wet-bulb and above, a heat tolerance screening programme would be required.

The DME Guideline for the Compilation of a Mandatory Code of Practice for an Occupational Health Programme (Occupational Hygiene and Medical Surveillance) on Thermal Stress, states that a code of practice must be prepared when the wet-bulb temperature exceeds 25°C.

Primary ventilation infrastructure

Shafts

Downcast (men and material)	Φ 8.5m
Upcast ventilation	Φ 6.0m

Fan stations

Operating flow and pressure	650m ^{3} /s and 4.5kPa

Power estimates

Main fans (650m ^{3} /s & 4.5kPa)	3,600kW
Capital development	2,000kW
Total	**5,600kW**

Note: Maximum power demand will depend on the ultimate scheduling.

Engineering Considerations

The following report describes the engineering of the proposed mining shaft system for the WBJV Project 1 mine pre-feasibility study. Conventional mining techniques have been used in the study to ensure a reasonable approach. The report includes the mining operations but excludes the plant, waste and tailings dumps, environmental and ventilation operations.

The shaft system

The deposit will be accessed via a twin vertical shaft system. The system geometry is typical of shafts designed for conventional mining at depths below 600 meters.

The main shaft will have a lined diameter of 8.5 meter and sunk to a depth of 712 meter. This down cast shaft is sized to ensure adequate ventilation, rock, materials, personnel and services flow for the 140 ktpm platinum bearing reef production requirement.

The up cast ventilation shaft will have a lined diameter of 6m and be sunk to a depth of 662m. This shaft will also be used as a second outlet and equipped with an emergency winder.

Surface infrastructure

The surface infrastructure supporting the mining operations will include access roads, parking areas, offices, change houses, workshops, capital and consumable stores, lay down areas, electrical sub-station and sewerage system.

The shaft bank area will include the winders, compressors, ventilation fans, ore handling systems and the headgear.

An area of 400 x 300 meter will be excavated and filled with suitable founding material to act as the base for the surface infrastructure. The area will be fenced and secured. The area will also be drained. Run off water will be contained to meet EMPR requirements and prevent flash flooding.

Compressed air supply

Compressed air is required for drilling operations (440 drills), service requirements (air pumps) and control purposes (air cylinders). The compressors will be designed to allow efficient operation during low consumption periods and peak consumption during day shift. The station will be sized for peak consumption. One standby compressor will be provided to ensure that unplanned breakdowns and planned maintenance shuts do not interrupt production activities.

Compressed air will be generated from a surface station located adjacent to the shaft. Air will be cooled and dried before being piped (350 mm) down the shaft to the working levels.
A water cooling system will be situated at the compressor station.

Workshops
Maintenance of the mining machinery would be undertaken on the following basis:

- daily and weekly maintenance programs would be completed on production machinery at or near the workface in maintenance bays or workshops;
- minor refurbishment will be completed in the surface workshops;
- major refurbishment of machinery will be outsourced to independent industry.

The engineering workshops will be right-sized for the shaft machinery. The workshops will include fitter (surface and production), diesel, boiler (surface and underground) and electrical workshops as well as wash and paint bays. The workshops will also include consumable, tool and lubrication stores. Secure lay down areas have been provided for production equipment, machinery, cabling, etc.

Underground maintenance bays would be located on each level.

Water supply
Magalies Water (MWB), the local water authority, has indicated that a water supply project inclusive of pipeline and dams is planned to supply water in the direct vicinity of the mine. MWB have indicated that the infrastructure could be expanded to include the mine's supply requirement. The supply point would be at the mine's surface boundary. The mine would pipe the water from this point to its plant approximately three kilometres away.

The shaft service water feeder pipe, to provide top-up water, will tee-off from this pipeline and feed the shaft header tank. Repayment of the infrastructure could be included in the tariff. MWM would require timeous notification of the mine's water requirement to allow them to be incorporated in the expansion plan.

It is envisaged the mining operations (excluding the plant) will operate on a neutral or marginally negative water balance.

Electricity supply
Eskom have confirmed in a meeting that power would be available for the planned mining and plant operations. The regional electrical supply infrastructure would be extended to supply the mine. The main

supply substation would be located close to the plant to optimise the infrastructure cost. The shaft substation will be supplied from the main substation.

A lump sum provision has been allowed for the electrical supply infrastructure in the study. The mine will be fed from a ring supply from two nearby Eskom switching yards at 88kV. A total of 30km of 88kV line will be required from Ararat and Boschkoppie switching yards.

In the main substation, metering of consumption and maximum demand will be effected by Eskom. Provision for these costs has been allowed in the study as per the relevant tariff structure.

Provision is also made for the use of 22kV temporary power to be supplied from Sun City as well as emergency power generation at the start of the project.

Main shaft

The main shaft will be equipped with one rock winder and two man winders. This was based on extensive investigation into the shaft duty required to service the mine. It will also be equipped with service piping, electrical feeders and communication cables, supported on brackets on the shaft sidewall. A steel headgear is planned with steel ore bins.

Winders

Rock winder

The double drum rock winder will be used to hoist the mined reef and waste. It has been sized as follows:

- Drum diameter 4,880mm
- Drum width 1,500mm
- Payload 13.5ton
- Speed 12m/s
- Motor rating 2530kW

Man Winder

Two double drum man winders are planned. Each will be configured with a cage and counterweight. They are sized as follows:

- Drum diameter 4,880mm
- Drum width 1,500mm
- Payload 10.0ton
- Speed 10.0m/s
- Motor rating 1305kW

Ore handling

Reef (Merensky and UG2) and waste will be transferred from the stope boxes to the shaft via battery locomotive trains. The hoppers will discharge their load into the shaft tips. The ore will pass through a sizing grizzly and into the correct ore pass system. Three ore pass systems (Merensky, UG2 and waste) will convey and store the reef/waste between production levels, i.e. from -262 meter to the transfer level (-662 meter). These systems operate separately such that each ore type and the waste are not mixed.

The ore will be fed via vibratory feeders onto the transfer level conveyors and into the measuring flasks. These flasks will deposit the ore into the skips for transport to the headgear ore bins. The waste will be transported by truck to the waste dump. A conveyor will transport the reef to the shaft surge silo and onto the plant feed conveyor to the plant.

Merensky and UG2 will be transported at different times to prevent mixing. The material handling systems will be purged (by running empty) as part of the ore change-over procedure.

Water reticulation

Service water will be fed from the 500 cubic meter surface header tank down the shaft in the 400 mm pipe service water feeder. The header tank will be topped up from the Magalies Water supply infrastructure.

Pressure reducing stations located on each level will adequately control the pressure of the water supplied to the mining operations.

Water will drain from the mining stopes on each working level and, after containment, will be pumped to the shaft. This water will drain via annex holes down to the water settlers located below the mine working levels.

Flocculent dosing will ensure an efficient settling process. The clean water will be stored in a dam prior to recirculation. Recirculation pumps will return the water to the upper level for reuse.

Excess water, possibly resulting from fissure water ingress, will be pumped to the surface header tank. This can then be used for mining purposes at a later date or forwarded to the plant. These main pumps will also be used for dewatering purposes should a major leak occur underground.

Dam systems will be sized and designed to accommodate water surge resulting from mining operations or other causes, such as power failures.

Shaft bottom pumps will pump the water collecting in the main shaft to the shaft settlers.

Ventilation Shaft

The ventilation shaft will be sunk conventionally using a stage and kibble winder. Thereafter the main ventilation fans will be installed at the head of the shaft.

A small emergency winder will be installed in the ventilation shaft as a second outlet as required by law. It will run on rope guides thereby preventing the need to equip the shaft with shaft steelwork and guides. The winder will be powered by ESKOM power or a generator set. This will allow evacuation of the mine in the rare event of a total power failure. The emergency winder will be sized as follows:

- Drum diameter 1,500mm
- Drum width 1,500mm
- Payload 1.5ton
- Speed 3.9m/s
- Motor rating 300kW

Item 25 (b): Recovery efficiency

Concentrator recovery

The expected recovery of platinum-group metals and conversion from ore to concentrate is 87.5% for Merensky Reef and 82.5% for UG2 Reef. The figures are supported by the metallurgical test work and correspond to the experience of neighbouring operations. These recoveries will be achieved with a penalty concentrate grade of 150g/t 4E; the mine will produce concentrate at better than 150g/t. The recovery of copper and nickel across the concentrator will be 60% and 50% respectively.

Concentrator plant and process description

The project team has concluded an initial process plant scoping study including estimates of capital costs and operating costs factored in from a project cost database. Operating costs for similar plants are well established.

The conceptual flow sheet includes a single-stream module containing primary and secondary milling units with rougher flotation after the first and second milling stages. Primary and secondary rougher concentrates are cleaned and recleaned as necessary to provide a combined low-mass final concentrate at an acceptable grade and recovery.

It should be noted that the same plant stream will be used to treat both ore types (Merensky and UG2); equipment is designed to accommodate the slower kinetics of the UG2 ore. The mining plan is for the first few years to involve the treatment of Merensky ore only, with gradual phasing into UG2-rich material. To facilitate smooth transition, use will be made of a number of surface stockpiles.

The plant will treat 140 kilo-tons/month of fresh ore on a dry basis.

For the purposes of this study it has been assumed that the plant will be operated under contract by an established operations contractor and the concentrate will be treated at a local smelter facility in the Rustenburg area.

Tailings disposal and the costs entailed in all associated aspects thereof have been included elsewhere in the study. Also excluded here are mining, geology, survey, waste rock disposal, statutory taxes and levies, owner's budget costs, head office, insurance, payroll, and all royalties, commissions, lease payments to other parties, capital for bulk power and water supply, environmental obligations, feasibility studies, etc.

Total plant and infrastructure capital costs are estimated at R427 million which includes a 10% contingency. Operating costs of R48.50 per ton milled are indicated, excluding power and water consumption. The capital cost allowances include all the necessary equipment and facilities to allow the plant to be operated correctly once commissioned. The capital cost is based on a "lump sum turnkey" proposal in respect of a project similar in size to WBJV Project 1, and thus additional contingency fees have not been included in the capital estimate.

The process description is as follows:

- Ore is delivered from the mining area to the shaft bin, and then conveyed to the crushing plant and surface stockpile. The plant design assumes that the ore will be effectively blended in the mining and stockpiling operation. Crushed ore is conveyed to the mill feed silo.
- Ore is withdrawn from the silo by apron feeders and conveyed to the primary mill. The mill is a steel-lined, grate-discharge ball mill. Water is added to the mill inlet with the recirculating load. Milled slurry discharges into a sump and is pumped to the circuit cyclones. When processing Merensky ore, cyclone underflow reports in the usual manner to the mill feed hopper to join the incoming feed. Overflow reports to a trash/woodchip removal screen, and the trash screen underflow to the primary rougher flotation conditioning tank.
- When processing UG2 ore, cyclone underflow passes over a vibrating screen. The cyclone overflow passes over the trash screen before rejoining the cyclone underflow that passes to the circuit vibrating screen. Vibrating screen oversize reports to mill feed again, while the undersize reports to the primary rougher flotation tank. The use of vibrating screens in modern UG2 primary milling circuits is normal and prevents the over-grinding of chromite that would occur if only cyclones were used.
- Primary rougher flotation is carried out in tank-type cells in series with gravity flow and sufficient residence time to accommodate the slower-floating kinetics of UG2 ore. The cells have level measurement, air-flow measurement and control. Primary rougher concentrate is pumped to the primary cleaner circuit for two stages of cleaning. Cleaner tails are pumped to the primary mill discharge sump.

- Primary rougher tailings are pumped to classifying cyclones ahead of the secondary mill, which operates in open circuit. Cyclone overflow reports to a sump from which it is pumped to a second stage of cyclones. Underflow from both stages reports to the mill feed hopper. Overflow from the second stage reports to the mill discharge sump, joining the milled slurry. This combined slurry is pumped to the secondary rougher flotation conditioner; secondary flotation is performed in tank cells in series with gravity flow.
- Secondary rougher concentrate is pumped to the secondary cleaner circuit. Again two stages of cleaning are used. Secondary cleaner tails are pumped out to join plant tails.
- Secondary rougher tails are pumped to a dewatering area which has a dedicated guard cyclone, thickener tailings disposal tank, pumps and a pipeline to the tailings dam.
- The proposed plant has a fully automated reagent makeup and distribution facility for activator, depressants, frother, collectors, and flocculant. The plant will also utilise up-to-date process control capability, but the control envisaged will be fit for purpose.
- Other features include high-pressure plant-compressed air, instrument-quality air, high-pressure filter air and low-pressure flotation cell air. There will be a small sample preparation facility from where samples will be sent to a commercial laboratory in Rustenburg for analysis.
- No Dense Media Separation plant has been included in the process design, as initial test work did not warrant this inclusion to upgrade the ore.
- No chromite recovery plant is envisaged until only UG2 is being processed through the concentrator. This will be a considerable period beyond production year 10, and is excluded at this time.

Tailings Dam

The tailings dam site has not been selected for WBJV Project 1, but as the area around the shaft is fairly flat, there is no restriction on the possible siting of this facility. The dam will be construction using upstream self raising construction techniques, which are very common in the platinum mining industry. The anticipated rate of rise will be approximately 2.0m/annum at a maximum deposition rate of 140ktpm. The tailings dam will cover an area of 100 hectares.

There will be return water facilities to retain the industrial water for mine use as well as the containment use of rainwater.
The tailings dam is expected to cost R45 million in capital plus return and slurry lines from the plant. The operating cost is expected to be R1.20 per ton milled.

Item 25 (c): Metal markets

It is envisaged that the Project 1 metal output will be shipped in the form of precious metals-bearing concentrates for processing by a smelter in South Africa. Saleable metals from the project concentrate include

platinum, palladium, rhodium, gold, copper and nickel as well as iridium and ruthenium. Osmium is present but this is not a pay element. Iridium, ruthenium and osmium are affectionately known as minor elements (the other platinum metals). Platinum is by far the most important metal in the project, with rhodium (in the UG2 concentrate) making a secondary contribution to revenue.

There are well-established global markets for all of the major metals and off-take for these concentrates as well as those of associated recoverable metals is assured. A smelter sale or ore sale agreement will be required; a pro forma agreement is included in the WBJV Agreement.

Markets for minor elements and rhodium are relatively thin and the metals are difficult to assay for and therefore are calculated in a resource model. Compared with the major metals there is considerably more risk attached to assumptions about price for the minor elements, as well as to their occurrence as estimated by the resource expert; but they contribute only about 3% of overall revenues. The fact that there are known and established recoveries of all minor elements in other parts of the BIC reduces some of the risk in assessing their financial returns.

Metal price assumptions suggested by PTM for this technical report are derived from consensus views as published by independent commodity and mining analysts. Platinum prices, which comprise approximately 60% of the value of the project have risen strongly over the past five years to over $1,000 per troy ounce.

The following long-term market prices and exchange rate were used for this technical report:

Platinum	US$	900 per ounce
Palladium	US$	330 per ounce
Rhodium	US$	2,000 per ounce
Gold	US$	500 per ounce
Ruthenium	US$	100 per ounce
Iridium	US$	250 per ounce
Copper	US$	1.31 per pound
Nickel	US$	4.65 per pound
Exchange Rate: R/US$		7.50

In general, these metal prices are reasonable and conservative in the context of current market conditions.

It is important to note that long-range metal price forecasting for any commodity is by nature difficult. Given the long lead times between the current report and the completion date of the project, Turnberry Projects has no basis to disagree with the suggested market-based consensus forecast selected by PTM.

In order to arrive at the indicated prices suggested by PTM, as operator of the WBJV, for this report, surveys of investment bank analysts were completed. Anglo Platinum and PTM conducted surveys of investment bank analysts' views independent of each other. The averages of each market survey were compared and the selected prices are within the range of the averages derived by the two groups of analysts. Three of the analysts were counted in both the PTM and Anglo Platinum survey group. The analysts' views cover long-term forecasts made by thirteen different analysts from South Africa, the UK and the USA in the second and third quarter of 2006.

The analysts' views were generally consistent with the overall view of the platinum, palladium, rhodium and other PGM markets as published by Johnson Matthey. Johnson Matthey publishes a quarterly and annual overview of the platinum, palladium, rhodium, and gold markets. Johnson Matthey is generally acknowledged as an authority on these markets."

Platinum *– Source: Johnson Matthey* – Demand for platinum grew by 2% to 6.7 million ounces in 2005. Its use in autocatalysts continued to increase whereas purchases of platinum jewellery declined. Demand for purposes of light-duty diesel vehicles was responsible for the largest growth in autocatalyst consumption. Purchases of platinum jewellery dropped 9% in reaction to higher metal prices. New supplies of platinum increased by 2%, bringing the world production figure for 2005 to 6.63 million ounces. South African output rose less than expected, while supplies from North America and Russia fell. Johnson Matthey in the report for 2005 expected the platinum market to remain moderately undersupplied in 2006 and that there would be further growth in demand from the light-duty diesel sector; in addition per-vehicle loadings are expected to rise. Autocatalyst demand for platinum in China, and the rest of the world, is also expected to continue growing in line with the high rate of vehicle production and tightening controls on emission. The outlook for platinum jewellery is mixed. Overall, say Johnson Matthey, the platinum market is likely to remain strong.

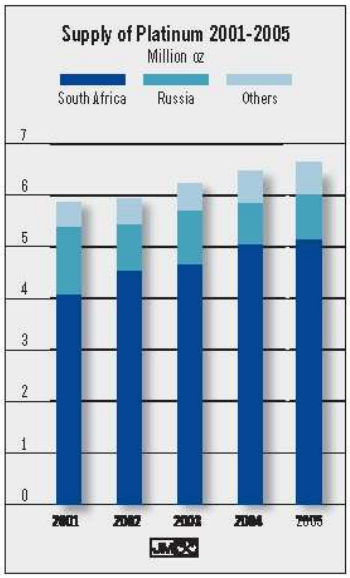
Palladium *– Source: Johnson Matthey* **-** Demand for palladium climbed to more than 7 million ounces for the first time in five years in 2005. The Chinese jewellery sector and autocatalyst demand are largely responsible for the growth in demand. Supplies in palladium were lower at 8.39 million ounces but remained well in excess of demand. In Johnson Matthey's view the prospects for palladium over 2006 are less good than for platinum.

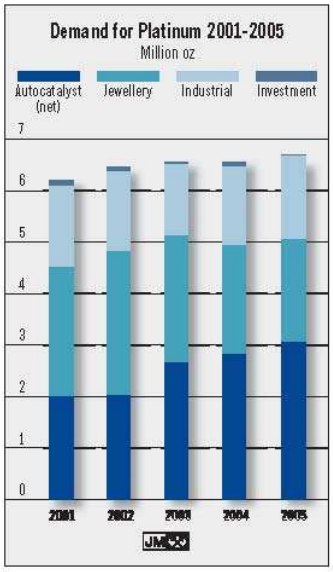
Rhodium *– Source: Johnson Matthey* – Purchases expanded by 11% in 2005 to 812 000 ounces, equalling the high that was set in 2000. Demand for rhodium in autocatalysts in China and the rest of the world increased by 11%, as a function of strong growth in the production of light vehicles in Asia and South America.

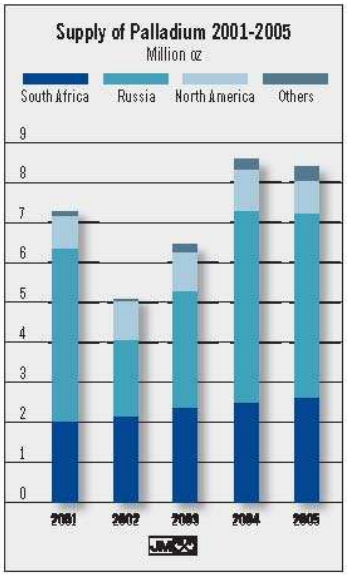
Ruthenium and iridium *– Source: Johnson Matthey* – The electronics sector was the driving force, substantially increasing the demand for ruthenium (17% higher in 2005), and iridium demand was up by more

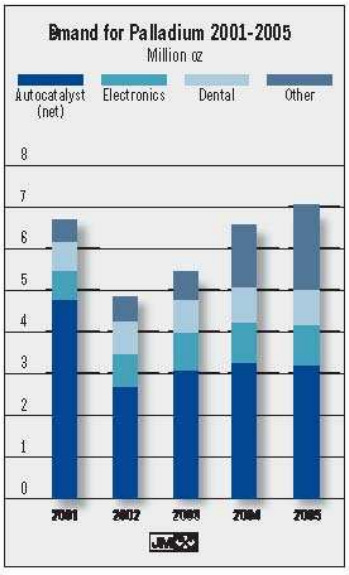
than 3% with growth coming from a range of applications. Foreseen tight supplies of rhodium, ruthenium and iridium are sighted in the Johnson Matthey 2005 report.

Market supply and demand – The following supply and demand data for platinum and palladium shows the historical trend s for the last 5 years. There is a continuing demand for both metals.









Nickel and copper are minor components of the valuable minerals included in the concentrate to be produced from Project 1. There is a ready market for these metals, which are sold by the major smelting facilities in South Africa. The actual forecasted world market prices for amounts actually to be received by the joint venture are discussed in Item 25(d) – Smelting.

A market consensus on long-term copper and nickel prices was determined by PTM, as operator of the WBJV project, by examining independent investment bank analysts' published positions for the third quarter of 2006. A total of 15 investment banks were surveyed; nine of the analysts provided long-term forecasts for nickel and eight gave long-term views for copper. The investment analysts concerned are based in Canada, the UK, USA and South Africa.

Forecasts of nickel prices range from US$4.00 per pound to US$5.89 per pound and the average is US$4.65 per pound. The range for copper is US$1.00 to US$1.58 per pound and the average is US$1.31 per pound.

Item 25 (d): Smelter contract

A draft pro forma smelter agreement is in place for the WBJV and contains the terms and conditions precedent for the sale of concentrate to the Anglo Platinum smelter operations at Rustenburg. The terms and conditions have not been fully negotiated between the parties but the following have been proposed and used in the project financials on the understanding that such negotiations will in fact take place:

- Smelter payment terms for Pt, Pd, Rh, Au, Ru, Ir, Cu, Ni will be 86%.
- Treatment charges will be R500 per dry ton of concentrate delivered to the smelter.
- Sampling charges will be R2,500 per sample lot of concentrate to determine the quantity and quality of concentrate delivered to the smelter.
- A concentrate moisture content of 17% will be acceptable but penalties will apply to moisture in excess of 15%. The rate is not specified but for purposes of this financial exercise it is assumed to be R30 per ton of water content in excess of 15%.
- A chromite penalty shall be applied on a sliding scale subject to no penalty below 1% Cr_2O_3. It is assumed that the Merensky concentrate will contain 1% chromite whilst UG2 concentrate will contain up to 4%.
- A grade penalty shall apply if the grade of 4E concentrate is below an as yet unspecified value – for this financial exercise, the penalty grade is assumed to be 150g/t and the mine will produce concentrate at better than 150g/t.
- A tonnage supply penalty may be applied once a production profile has been established, but for now it is assumed that the production schedule will be met or exceeded.
- Payment will be effected in month 4 following receipt of concentrate at the smelter.

In addition, there is an option for the toll-treatment of ore from the mine if desired or expedient for cash-flow reasons, subject to availability of processing capacity.

There is no financing aspect associated with this contract currently with all payments effectively being made four months after delivery of concentrate to the smelter. Currently this aspect is treated as a working capital situation with revenue delayed by 4 months.

In addition, the mining royalty bill effects are to be incorporated into the smelter contract.

Item 25 (e): Anticipated environmental considerations

The holistic environment that will be affected by the project is considered, which includes a combination of social, cultural, historical, economic, political and ecological aspects. In some instances the impact will be cumulative and can be viewed as the total effect on a resource, ecosystem, or human community of that particular action and all other activities affecting that resource no matter what the entity. Cumulative impact cannot be fully assessed during the Pre-feasibility phase owing to the lack of detailed information. On completion of the specialist studies, the cumulative effects will be evaluated and incorporated into the Feasibility study.

Several anticipated environmental issues and impacts were identified of which numerous impacts cannot be fully quantified until further investigation is undertaken. Hence, the mitigation measures and costs of mitigation can only be estimated on what is known at this stage, and these will need to be re-assessed in the Feasibility study.

Geology

Disturbance of the natural geology will occur during the constructional, operational, decommissioning and closure phases of the project. During the construction, there will be a site-specific impact on the geology as material will be removed. The material will either be stockpiled or used for construction of roads and storm water diversion structures. In addition, minerals will be removed when mining takes place. This impact is unavoidable.

Climate

There is unlikely to be any impact on the climate at or in the vicinity of the proposed project site.

Topography

Changes to the topography of the area will result from mining operations, such as stockpiles and eventual rehabilitation. The topographical changes cannot be prevented and will occur during the construction, operational and decommissioning phases of the project. During the decommissioning and closure phases the affected areas will be rehabilitated and contoured to represent pre-mining topography as closely as possible. The impact on the topography will decrease significantly during this phase of the project.

Land use

The entire site comprises grazing land and arable land for certain crops only. Currently the main land use in the surrounding area is grazing. The development of the project will result in a loss of cultivated and grazing land.

Fauna

No endangered species or signs of activity of endangered species were observed on site. The potential impact on the animal life related to the loss of habitat and encroachment of human activity is low. However, the riverine areas are suitable habitat for the Giant Bullfrog and should be assessed in more detail during the environmental impact assessement (EIA).

Flora

Vegetation on the project area comprises mainly Acacia-grassveld. Large parts of the area have been disturbed by grazing and to a lesser extent by cultivation and as a result none of the areas to be disturbed are considered to be of high ecological quality.

Surface water

There are no recognised rivers or streams crossing the project area. The nearest major river to the site is the Elands River and care should be taken to avoid runoff to the river.

Groundwater

It is expected that dewatering of the aquifers surrounding the proposed mining area will occur due to dewatering of the mine workings. The extent of the drawdown cone and the impact on the surrounding private groundwater users has to be quantified during the detailed feasibility investigation. Recovery of the groundwater levels in the post-mining environment can cause decanting and contaminant migration away from the site. It is expected that contamination originating from the mining area, the tailings stroring facility (TSF) and the waste rock dump could occur.

Visual

The visual impact associated with the proposed mining operations will continue for the construction, operational and decommissioning phases of the project and the impact does not differ significantly between these phases. The visual impacts will be observable on a local scale only.

The scale and bulk of stockpiles will result in a slight change to the landscape characteristics. The magnitude of intrusion will diminish during the decommissioning and closure phases when rehabilitation and re-vegetation operations take effect. In the long term after decommissioning and closure, assuming that

landscape rehabilitation and other mitigation measures have been effective, the negative visual impact will probably be reduced to very little.

Item 25 (f): Taxes

Mining royalties

The mining royalties bill has not yet been passed by the South African parliament but recent discussion papers have been released indicating that the following royalties will apply in the platinum mining industry:

- Platinum-group metals – 6% of revenue if the project is not producing metals and 3% if the project can confirm that added-value products are produced from the concentrate. A value of 3% of revenue has been used in this evaluation.
- Base metals – 1% of revenue.

These royalties are possibly being introduced as from 2009, and will thus apply to the full production from this project.

This royalty aspect is still to be incorporated into the smelting contract as recommended by smelter contract lawyers.

Tax situation

The Net Present Value model, which forms part of this document, is based on the effective tax rates applicable to companies at the time of writing. The rate applicable to normal pre-tax profit was 29%. The tax rate applicable to dividends declared and paid is 12.5%, as a secondary tax on companies. The assumption in the NPV calculation is that all profits will be taxable at 29% and the entire after-tax profit will be distributed to shareholders in the form of dividends.

Item 25 (g): Capital and operating cost estimates

Capital expenditure – vertical shaft

The estimated capital cost of developing the vertical shaft option for the WBJV Project 1 is based on the following information:

- Shaft-sinking estimates by a reputable sinking contractor factored for changed arrangements.
- Concentrator costs factored from database information for a recently priced platinum concentrator of similar size.
- Database cost estimates for other major equipment required to operate a mine.
- Development metres based on the production schedule and costed at industry rates.

The sinking costs have been developed from first principles by a reputable sinking contractor, on the basis of a schematic diagram prepared by the project team. The schematic has subsequently been modified and the costs have been factored from the sinking estimate to suit the new arrangement. Equipment supply prices have been estimated and factored in from database information.

The concentrator costs include all required provisions within the confines of the plant, to operate the plant at the required throughput, appropriate recovery and necessary availabilities for processing. These costs are based on a recent “lump sum turnkey” contract price for a similar-sized plant in the platinum industry.

Owner’s costs are to be accumulated from July 2006 and are allocated to the project from the current financial year – i.e. Year -2 in the capital schedule. Currently, the sunk costs prior to June 2006 are R53 million and these have not been incorporated into the capital costs model. It is anticipated that a total additional cost of R49.7 million will have accumulated by the end of June 2008. These secondary costs may be recouped from the individual shareholders from the capital provision instituted by the mine operators.

There is a preliminary & general provision of 15% included in a number of items in respect of purchase costs and design work. This figure is regarded as a reasonable and appropriate provision for such additional work by any engineering company.

The spread of the capital is such as to enable the shaft sinking to be complete within four years and for primary development to commence Month 1 of Year Five. First production is to commence in Month 8 of Year Five. The concentrator and all surface infrastructures are to be complete by this time to accept the production. Portions of the mining equipment are to be available from Month 1 of Year Five with a ramp-up in production until Month 5 of Year Eight.

All costs are effectively in second quarter 2006 money terms.

A working capital provision equal to four months of production is intended to cover the delay in cash flow from sales by the contract smelter.

A sustaining capital provision equal to 3% of the total mine working costs is included to allow for ongoing capital expenditure. This is in addition to the requirements for mining fleet and surface vehicle replacements, and amounts to approximately R17 million per annum for major repairs and new capital items.

A capital contingency of 15% is provided to cover any inaccuracies that have been introduced into capital estimates, apart from those associated with the concentrator.

Where exchange rates are applicable, the rate is R7.50 : $US1.00 as at second quarter 2006.

The capital cost of the project excluding working capital and primary development is R2,154.6 million, the primary development being R302.5 million. The working capital increases to R375 million once the mine has reached steady-state production in Year Seven and then decreases – the working capital will be recouped at the end of the mine's life; thus, over the life of the mine the accumulated working capital is nil, as indicated below. Peak funding requirement for the project will be R2,143 million.

The total capital cost to get the mine into initial production will be R2,092 million to the end of Year Six. From then on there will be adequate production to cover ongoing capital requirements and to make a working profit.

The accuracy of this estimate is expected to be roughly 25% and with the introduction of the 15% contingency, it is expected that the reporting accuracy will be between 10 and 15%.

Project Philosophy

As the operators of the WBJV Project 1 mine, PTM do not have large "in house" engineering capabilities, it is proposed that the shaft sinking contract will be based on the design and build concept with a reputable contractor and the contractor providing all design services and handing over a completed shaft to the operators.

A reputable mineral processing contractor will build the concentrator on a lump sum turn key (LSTK) basis.

Capital cost schedule

The capital cost schedule as developed within this study and used in the financial model is summarised in the following table:

TOTAL CAPITAL	**Rands**	**2,457,107,404**
Owners Costs	Rands	7,500,000
Twin Decline System	Rands	-
Shaft Access with In-Circle Development	Rands	878,599,093
Internal Incline and Decline Systems	Rands	13,552,463
Primary & Secondary Development	Rands	302,502,576
Ventilation	Rands	26,657,234
Supply Portion	Rands	271,460,375
Concentrator Plant	Rands	427,000,000
Tailings Dam	Rands	45,000,000
On Going Capital Requirement	Rands	242,065,554
Contingency Provision	Rands	242,770,109
Working capital (4 months of Revenue)	Rands	0

The timing of this capital expenditure during the first eight years of the mine's life is indicated in the following table:

	Year	1	2	3	4	5	6	7	8
TOTAL CAPITAL	Rand s	286,002,279	341,747,031	626,367,326	620,339,648	318,011,131	166,822,966	483,868,293	116,360,407
Owners Costs	Rands	1,500,000	1,500,000	1,500,000	1,500,000	1,500,000	-	-	-
Twin Decline System	Rands	-	-	-	-	-	-	-	-
Shaft Access with In-Cliride Development	Rands	193,741,960	231,622,853	270,774,507	160,011,267	22,448,506	-	-	-
Internal Incline and Decline Systems	Rands	-	-	-	-	-	-	8,710,000	2,212,463
Primary & Secondary Development	Rands	-	-	-	-	138,697,066	19,316,558	14,754,432	8,693,990
Ventilation	Rands	-	-	-	23,040,000	1,250,000	-	800,000	800,000
Supply Portion	Rands	9,303,500	38,755,000	57,307,950	73,361,950	54,781,975	28,750,000	9,200,000	-
Concentrator Plant	Rands	-	28,750,000	143,750,000	208,500,000	46,000,000	-	-	-
Tailings Dam	Rands	-	-	-	9,900,000	11,550,000	5,750,000	-	10,800,000
On Going Capital Requirement	Rands	-	-	-	-	1,574,452	6,846,890	20,368,083	17,913,859
Contingency Provision	Rands	30,456,819	41,119,178	52,024,869	44,026,330	35,209,132	8,072,484	5,394,665	3,375,968
Working capital (4 months of Revenue)	Rands	-	-	-	-	-	88,087,024	374,631,114	71,554,128

Item 25 (h): Economic analysis

Financial evaluation

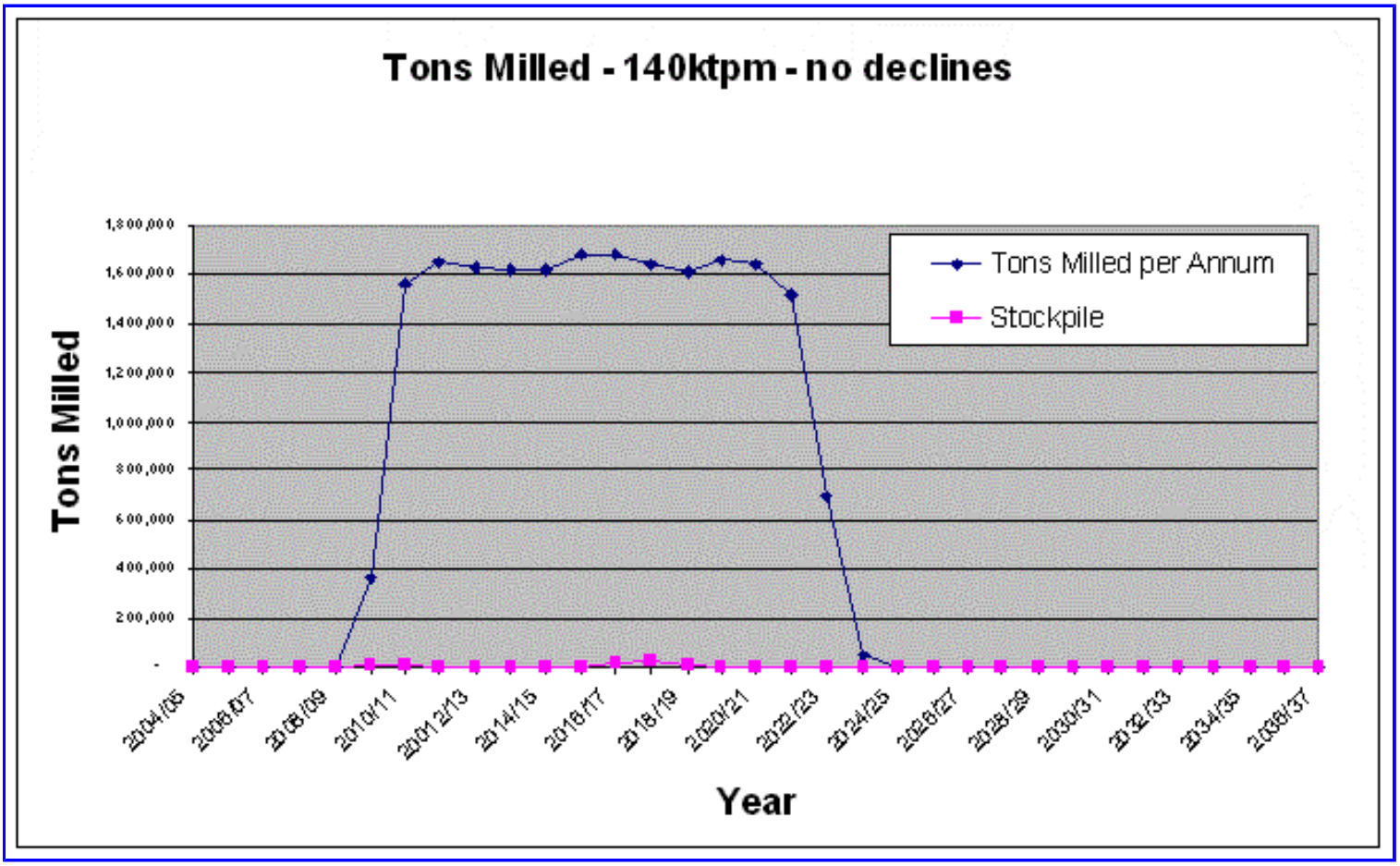
Production schedule

The tonnage and production schedules have been developed from the mining plan – as described in Item 25(a) – with a number of relevant flow-of-ore factors applied to allow for achievement of a reasonable production target. The mining production will be stored if it exceeds the milling capacity and consumed when capacity if available. The concentrator has a nominal capacity of 1,680,000 tons per annum or 140,000 tons per month.

The mine will be mining both Merensky and UG2 Reefs during its life, with the higher-grade Merensky Reef being produced initially and as production declines from this reserve the UG2 Reef will supplement the tonnage milled. The Merensky Reef produces a higher yield than the UG2 Reef and thus the metal production diminishes in later years which are clearly seen in the following schedule:

	Merensky			UG2			Total		
Production Year	Tons to Mill	Mill Feed Kg's	Mill Feed grade g/t	Tons to Mill	Mill Feed Kg's	Mill Feed grade g/t	Tons Milled	Mill Feed Kg's	Mill Feed grade g/t
5	9,357	26.29	2.81				9,357	26	2.81
6	356,142	1624.70	4.56				356,142	1,625	4.56
7	1,560,581	8686.56	5.57				1,560,581	8,687	5.57
8	1,649,108	10138.65	6.15				1,649,108	10,139	6.15
9	1,631,130	10245.19	6.28				1,631,130	10,245	6.28
10	1,618,580	9469.42	5.85				1,618,580	9,469	5.85
11	1,620,053	9162.02	5.66				1,620,053	9,162	5.66
12	1,696,724	9431.05	5.56				1,696,724	9,431	5.56
13	1,635,225	8944.41	5.47	49,467	175.33	3.54	1,684,692	9,120	5.41
14	995,298	5333.51	5.36	627,168	2378.50	3.79	1,622,466	7,712	4.75
15	334,603	1856.87	5.55	1,277,052	4536.26	3.55	1,611,655	6,393	3.97
16	48,905	276.60	5.66	1,609,883	5790.16	3.60	1,658,788	6,067	3.66
17				1,647,359	5963.85	3.62	1,647,359	5,964	3.62
18				1,513,266	5430.82	3.59	1,513,266	5,431	3.59
19				692,200	2469.41	3.57	692,200	2,469	3.57
20				50,488	174.37	3.45	50,488	174	3.45
Grand Total	13,155,707	77,123	5.86	7,466,883	27,609	3.70	20,622,589	102,114	4.95

These production tons made available to the concentrator are manipulated to provide constant feed to the concentrator with a capacity of 140,000 tons per month or 1,680,000 tons per annum. In the event of excess production from the mine, the material is stockpiled. The production profile for the concentrator is depicted in the following graph, with the stockpile also indicated.



Graph 19: Production profile.

The mine will produce a total of 21 million tons of Merensky Reef and UG2 Reef during its life and present 102 tons of 4E to the concentrator. The concentrator will deliver to the smelter some 88 tons of 4E contained in 508,602 tons of concentrate.

Metal Prices

There has been much debate about which metal prices should be used in the financial model, the options being

- current metal prices and exchange rates;
- historical 24-month averages of metal prices and exchange rates;
- long-term metal prices and exchange rates as predicted by individual banking and investment institutions; and
- long-term metal prices and exchange rates as predicted by major mining companies.

Metal price assumptions suggested by PTML for this Technical Report are derived from consensus views as published by independent commodity and mining analysts. The platinum price, which comprises approximately 60% of the value of the project, has risen strongly over the past five years to over $1,000 per troy ounce.

For Project 1, it was decided that the base case would be evaluated on a metal price of R185,800 for Merensky Reef and R208,800 for UG2 Reef. Sensitivity analyses around these values may be performed to understand the effect of the differing forecasts.

The base prices (not including discounts to be applied at smelter payment for concentrate) used are:

Platinum	US$	900 per ounce
Palladium	US$	330 per ounce
Rhodium	US$	2,000 per ounce
Gold	US$	500 per ounce
Ruthenium	US$	100 per ounce
Iridium	US$	250 per ounce
Copper	US$	1.31 per pound
Nickel	US$	4.65 per pound
Exchange Rate: R/US$		7.50

It is assumed that no value is generated from Osmium.

There is to be a base metal discount applied at the rate of $US150 per ton for copper and $US70 per ton for nickel as per the WBJV Agreement.

The metal prices are summarised in the following table:

WBJV PROJECT 1

Merensky Reef - PGM Basket Calculation

		Base Price	Exchange Rate	Discount	Metal Price	Split	Head Grade	Assumed Price
		$US/oz	R/$US		R/kg	%	g/t	R/kg
Basket Metal	Pt	$900	7.50		R 217,018	65.3%	4.15	R 202,500
	Pd	$330	7.50		R 79,573	26.3%	1.67	R 78,800
	Rh	$2,000	7.50		R 482,261	3.6%	0.23	R 450,000
	Au	$500	7.50		R 120,565	4.9%	0.31	R 112,500
	3PGE's & Au	$771	Basket Price		R 185,818	100.0%	6.36	R 185,800
	Ir	$250	7.50		R 60,283		0.33	R 60,000
	Ru	$100	7.50		R 24,113		1.46	R 24,000
	Os	$750	7.50		R 180,848			R 180,800
		$US/ton			R/ton			
	Cu	$2,887	7.50	$150.00	R 20,529	Prices Consensus		R 20,500
	Ni	$10,249	7.50	$70.00	R 76,340	Date 31-Oct-06		R 76,300

UG2 Reef - PGM Basket Calculation

		Base Price	Exchange Rate	Discount	Metal Price	Split	Head Grade	Assumed Price
		$US/oz	R/$US		R/kg	%	g/t	R/kg
Basket Metal	Pt	$900	7.50		R 217,018	63.0%	2.47	R 202,500
	Pd	$350	7.50		R 84,396	26.0%	1.02	R 78,800
	Rh	$2,000	7.50		R 482,261	10.2%	0.40	R 450,000
	Au	$500	7.50		R 120,565	0.8%	0.03	R 112,500
	3PGE's & Au	$866	Basket Price		R 208,762	100.0%	3.92	R 208,800
	Ir	$250	7.50		R 60,283			R 60,000
	Ru	$100	7.50		R 24,113			R 24,000
	Os	$750	7.50		R 180,848			R 180,800
		$US/ton			R/ton			
	Cu	$2,887	7.50	$150.00	R 20,529	Prices Consensus		R 20,500
	Ni	$10,249	7.50	$70.00	R 76,340	Date 31-Oct-06		R 76,300

PGM metal splits

The platinum-group metal splits – into Pt, Pd, Rh, Au, Ru and Ir – are based on a large number of samples analysed for these six elements. The splits as determined for the mining zones have been averaged and applied to the concentrate production, assuming that there is no preferential loss of individual elements across the concentrator. The metal splits for each ore type are:

	Merensky Reef	UG2 Reef
Pt	65.25	63.01
Pd	26.26	26.04
Rh	3.62	10.19
Au	4.87	0.77
Sub-total	**100.00**	**100.00**
Ru	22.90	8.96
Ir	5.18	2.20
Grand-total	**128.08**	**111.16**

These splits have been used in the financial model to determine the basket metal price for 4Es for the expected head grade to be processed.

If the above metal prices and exchange rates are applied to these splits the average basket price for 4Es contained in Merensky is R185,800/kg and for the UG2 basket it is R208,800/kg.

Escalation and inflation

Neither escalation nor inflation factors have been applied within the financial model. The assumed capital and operating costs are as at the second quarter of 2006. During this time the mining industry has been under extreme inflationary pressure and substantial increases in prices of steel and capital equipment, being primarily due to the scarcity in the marketplace. Delivery time of items such as milling and winding plant has moved from approximately 52 weeks in late 2005 to 78 weeks-plus and this is having a major influence on costs.

Operating costs

The cost centres are: Stoping, Development, Concentrator, Concentrate Transport, Treatment Charges, Sampling Charges, Services (power and water), Administration Costs and Overheads. Provision is made for Penalties, which are deducted from Revenue. The following allocations are based on benchmarking and database information:

Stoping: R200/t, corresponding to approximately R632/m ^{3} for Merensky Reef and R720/m ^{3} for UG2 Reef.

Development: R230/t, taking into account all pipes, cables and trackwork. This corresponds to approximately R830/m ^{3} in hoisted rock, which is the approximate cost as determined by neighbouring mines.

Concentrator: a processing cost of R48.50/t, excluding services.

Tailings dam : tailings placement costs of R1.20 per ton milled have been included.

Other costs are as follows:

- *Concentrate transport* costs are calculated on the basis of 17% moisture in the concentrate, a distance of 30km to the nearest smelter and a transport rate of R0.85 per ton kilometre. This amounts to R26 per ton shipped or about R0.88 per ton milled.
- *Treatment charges and sampling* charges have been discussed above and form part of the overall concentrate treatment contract. This amounts to R15.09 per ton milled.
- *Services – electric power and water* : The electric power provision is based on 65 kilowatt hours per ton mined for the mining operations, and 55 kilowatt hours per ton milled for the concentrator. These consumption figures are based on database information and not on a detailed equipment list. The power cost is R0.18 per kilowatt hour, fully inclusive of all other charges. Water is supplied from the local water board at an estimated cost of R4.50/m^3 . The mine is expected to consume 0.4m^3 per ton mined and the plant will consume 0.5m^3 per ton milled. The total cost for both services will be R27 per ton milled.

Administration: The general administration of the mine is expected to cost approximately R11.50 per ton milled. This includes the general activities required to operate the mine including the following:

- transport of staff from local housing to the mine
- managing mine accounts and finances
- operating the mine warehouse and consumables purchase
- managing the purchase of large or capital items
- external consultants
- general mine administration including communications, office requisites, etc.
- auditing fees
- insurances
- operation of company vehicles
- maintenance of offices and buildings
- security personnel
- human resources management
- safety and training departments
- conferences and ongoing professional training for staff.

The allowance of R11.50 per ton milled is a provision and has not been developed from first principles but is regarded as fair and reasonable.

Overheads associated with a mine of this size are expected to be approximately R7.50 per ton milled. These will be incurred at corporate level and will include

- funding requirements
- corporate costs and Head Office fees
- marketing fees where appropriate
- interaction with government officials and departments
- advertising
- financial statements
- Board meetings and social events
- social investment programme

The amount of R7.50 per ton is provisional and has not been developed from first principles but is regarded as fair and reasonable.

Rehabilitation fund and closure costs: The expected cost of rehabilitation and closure of the mine is between R75 and R100 million. For the fund there is an ongoing provision of R4 per ton milled, which will result in R82.4 million at closure.

WBJV Project 1 operating cost: The life-of-mine operating cost is expected to be approximately R352 per ton milled. The costs comprise the following aspects, based on the parameters detailed above.

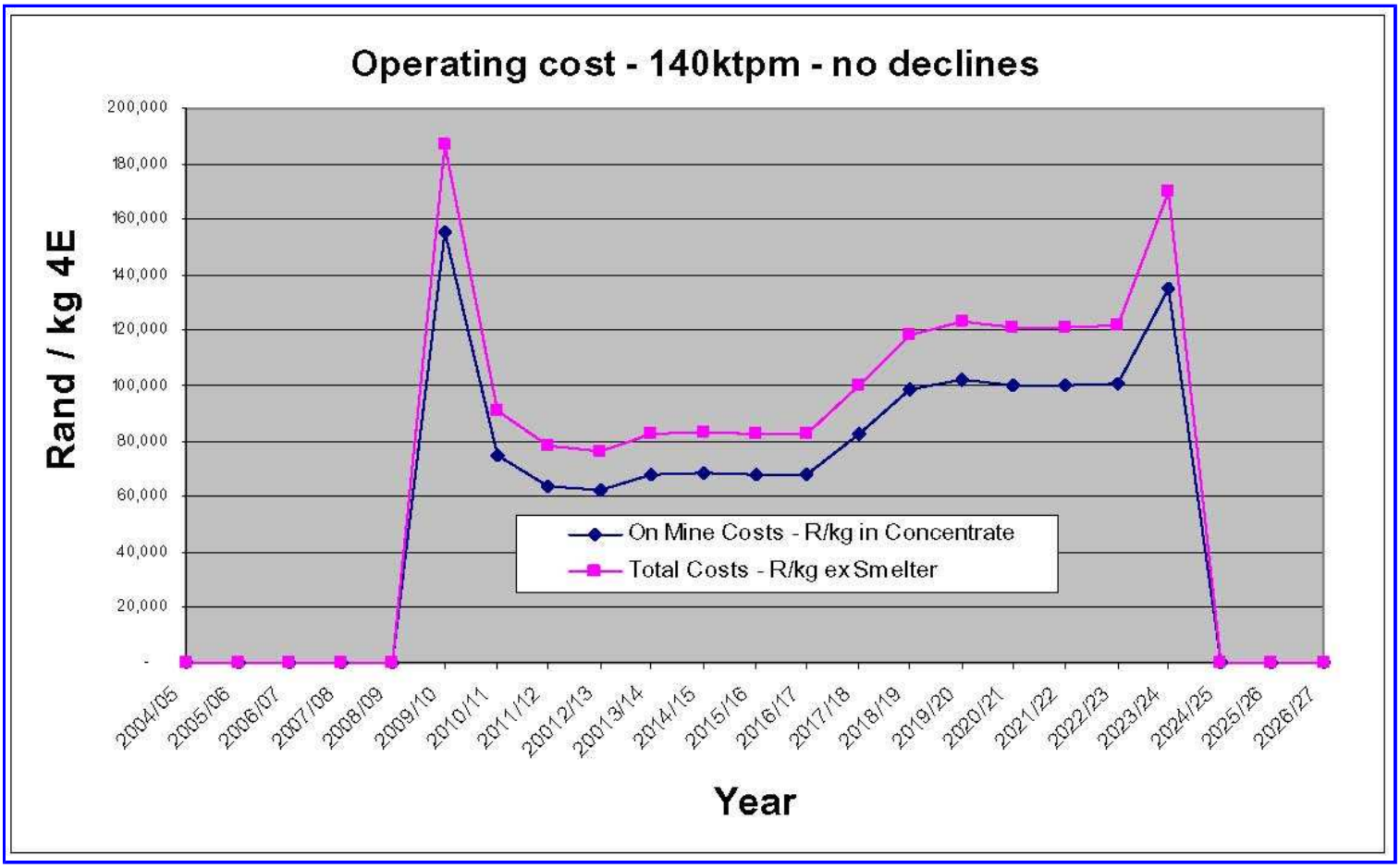
TOTAL	R('000)	7,255,518,474
Mining	R('000)	4,867,859,976
Processing	R('000)	1,000,195,567
Tailings Disposal	R('000)	24,747,107
Conc. Transport	R('000)	18,083,559
Conc. Treatment	R('000)	311,176,058
Services	R('000)	559,176,434
Admin.& Overheads	R('000)	391,789,418
Rehabilitation Fund	R('000)	82,490,356

TOTAL	R/ton milled	351.82
Mining	R/ton milled	236.05
Processing	R/ton milled	48.50
Tailings Disposal	R/ton milled	1.20
Conc. Transport	R/ton milled	0.88
Conc. Treatment	R/ton milled	15.09
Services	R/ton milled	27.11
Admin.& Overheads	R/ton milled	19.00
Rehabilitation Fund	R/ton milled	4.00

The off-mine costs – those associated with the concentrate sales and transport – account for less than R16 per ton milled. The expected on-mine cost up to and including concentrate production is R336 per ton milled.

If the smelter discount and treatment charges are modelled as an operating cost rather than as a discount to revenue, the operating costs in terms of platinum-group metals production is R95,942 per kilogram of 4E paid for as per the pro-forma concentrate off-take agreement.

The on-mine operating costs – excluding any concentrate treatment, sampling or transport – are based on kilograms of 4E in concentrate as depicted in the following graph for the life of the mine. The on-mine cost will be R78,972 per kg 4E in concentrate. The estimated operating cost is accurate within 25%.



The cost-trend as shown in the above graph will be similar for the vertical shaft and decline option.

Reasonableness check on operating costs: On the basis of an average of R7,000 per employee listed in the Human Resources section, the total labour costs for the mine will be R23.78 million per month. The cost of labour within the platinum industry as given in annual reports by the major platinum producers is approximately 50% of the total mine cost up to and including concentrate production. Applying this industry norm, the operating cost is expected to be R47.56 million, i.e. R339.70 per ton milled at 140kt/m.

Benchmarking four similar operations/projects within the public domain indicated a range of operating costs of between R275 and R372 per ton.

If the costs of transporting and treating the final concentrate are split from the overall amount of R351 per ton milled, the operating cost is reduced to approximately R335 per ton milled. This is regarded as being within the reported range for platinum producers and is within 5% of the reasonableness check depicted above.

These benchmarking exercises support confidence in the operating costs as detailed in the study.

Economic evaluation – vertical shaft

The inclusion of the production profile, the capital expenditure profile, the operating cost estimates and the revenue generated by the metals sold, results in the economic and financial model shown in Table 14 – Appendix A.

The internal rate of return (IRR), the net present value (NPV) as at December 2006 and the net future value (NFV) as at July 2008 as calculated in a pre-tax scenario are detailed as follows:

Financial analysis:

Pre-Tax Net Present Value @ December 2006	5.0%	10.0%	12.5%
R'000	2,544,530	**977,390**	533,799
Pre-Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	2,671,756	**1,075,128**	600,524
Pre-Tax IRR	**17.7%**		

The after-tax scenario is depicted in the following table.

After Tax Net Present Value @ December 2006	5.0%	10.0%	12.5%
R'000	1,333,571	**347,793**	72,350
After Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	1,400,250	**382,573**	81,394
After-Tax IRR	**13.4%**		

The peak funding requirement will be R2,143 million. The project cash flow is depicted in item 25(i).

Human resources

Labour : The mine labour force has been estimated from zero base as follows:

	Workers	Supervisors	Management	Leave Relief	TOTAL
Stoping	1680	98		178	1956
Development	350	14		36	400
Tramming	210	42		25	277
Engineering	140	28		17	185
Shaft Operations	48	16		6	70
Technical Services	100	20	17	14	151
Re-Establishment	100	10		11	121
TOTAL UNDERGROUND	**2628**	**228**	**17**	**287**	**3160**
Engineering	20	5		3	28
Stores / Administration	40	5		5	50
Concentrator	120	12		13	145
Senior Staff			13	1	14
TOTAL SURFACE	**180**	**22**	**13**	**22**	**237**
TOTAL MINE	**2808**	**250**	**30**	**309**	**3397**

This table indicates the potential number of people that will be employed once steady-state production on Project 1 is achieved.

Gate wage salary : The financials for Project 1 are based on a gate wage to all employees.

Safety and environment : The safety of all employees will be paramount and an effective safety system, managed professionally, will be implemented. In addition, the environment will be actively monitored to ensure that water quality, dust loadings, noise abatement and other sensitive issues are managed professionally. External auditors in the field of safety and the working environment will conduct regular audits of the mine to ensure compliance with national, regional and local standards.

Economics evaluation – decline system

The option of a surface decline system for mining the shallow portion of the Merensky Reef has been evaluated to the same degree of accuracy as the vertical shaft option detailed above. A reputable sinking contractor with considerable experience in sinking decline shafts cost the sinking of twin declines. The estimate for the twin declines is R190 million. There will be some savings associated with the reduced infrastructure on the upper level. However, additional capital will be required estimated at R205.7 million. Peak funding of R2,040 million will be required for this project, assuming that all early decline production can be treated at a neighbouring concentrator.

The decline option can be summarised as below by comparison with the vertical shaft. The mining costs will be lower than those for the vertical shaft because trackless equipment will be used in the work environment. The projected operating costs will be R150 per ton mined in the stopes rather than R200. The development costs will be lower as well: approximately R200 per ton mined as against R230.

Since the concentrator construction project will not be brought forward to allow processing of the low tonnage from the decline system, it is expected that the reef produced would have to be treated at a neighbouring concentrator; for this it is deemed necessary to transport the reef to a concentrator in Rustenburg rather than to Bafokeng Rasimone Platinum Mine. Toll-treatment of the ore is expected to attract a premium of R10 per ton treated; in addition there is a transport cost of R30 per ton treated. This increases the concentrator costs to R88.50 per ton treated.

The total amount of revenue will be unaffected by the decline system. There will additional capital costs with marginally reduced operating costs and some production in earlier years.

Table 13 – Appendix A – shows the resultant financial model if the economics of including the decline system are considered.

The financial results of this model in pre-tax money are:

Financial analysis:

Pre-Tax Net Present Value @ December 2006	5.0%	10.0%	12.5%
R'000	2,441,366	**910,921**	476,122
Pre-Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	2,563,435	**1,002,013**	535,637
Pre-Tax IRR	**17.0%**		

In after-tax terms the results are as follows:

After Tax Net Present Value @ December 2006	5.0%	10.0%	12.5%
R'000	1,268,921	**300,156**	27,876
After Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	1,332,367	**330,172**	31,360
After-Tax IRR	**12.8%**		

The above financial evaluation indicates that the decline system may detract from the overall project economics with the pre-tax IRR decreasing from 17.7% to 17.0% and a commensurate drop in the NPV figures.

Whilst the option of using a decline system for early access to the Merensky Reef may seem attractive, there is insufficient tonnage, albeit at a higher grade, to adequately fund the project. There is limited financial benefit in pursuing this option. There may be some political or corporate reason to reconsider the decline option at a future time, perhaps during the feasibility study.

The financials indicate that both options are equally viable and therefore should be considered in the BFS.

There are significant risks associated with the decline system, the major one being uncertainty as to whether a neighbouring mine will have spare capacity to treat the ore produced. As at November 2006, all processing capacity is being utilised at neighbouring operations and if this situation continues the ore produced from the declines will be stockpiled and only incur costs without any revenue.

It is the recommendation of the independent mining engineers that the decline system may be considered for further work, unless there is a corporate reason for not pursuing this option.

Sensitivity to metal prices

The recent significant upswing in the metal prices for PGMs and Base-metals is a sensitivity to the base case long-term metal pricing indicated above. Metal prices (not including discounts to be applied at smelter payment for concentrate) in mid November 2006 were as indicated:

Platinum	US$	1,204 per ounce
Palladium	US$	322 per ounce
Rhodium	US$	4,800 per ounce
Gold	US$	627 per ounce
Ruthenium	US$	100 per ounce
Iridium	US$	250 per ounce
Copper	US$	3.36 per pound
Nickel	US$	14.88 per pound
Exchange Rate: R/US$		7.25
Merensky Reef	R/kg 4E	250,000
UG2 Reef	R/kg 4E	311,400

Inserting these prices into the vertical shaft financial model increases the economic valuation considerably for pre-tax and post-tax valuations:

Financial analysis:

Pre-Tax Net Present Value @ Dec 2006	5.0%	10.0%	12.5%
R'000	6,630,687	**3,246,723**	2,257,354
Pre-Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	6,962,221	**3,571,396**	2,539,523
Pre-Tax IRR	**28.9%**		

After Tax Net Present Value @ Dec 2006	5.0%	10.0%	12.5%
R'000	3,745,423	**1,702,847**	1,108,358
After Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	3,932,694	**1,873,131**	1,246,903
After-Tax IRR	**22.7%**		

There is a considerable improvement in the economic returns using current metal prices. When these metal prices are inserted into the decline-vertical shaft model, the discrepancy between the two options is negated as indicated in the following tables.

Financial analysis:

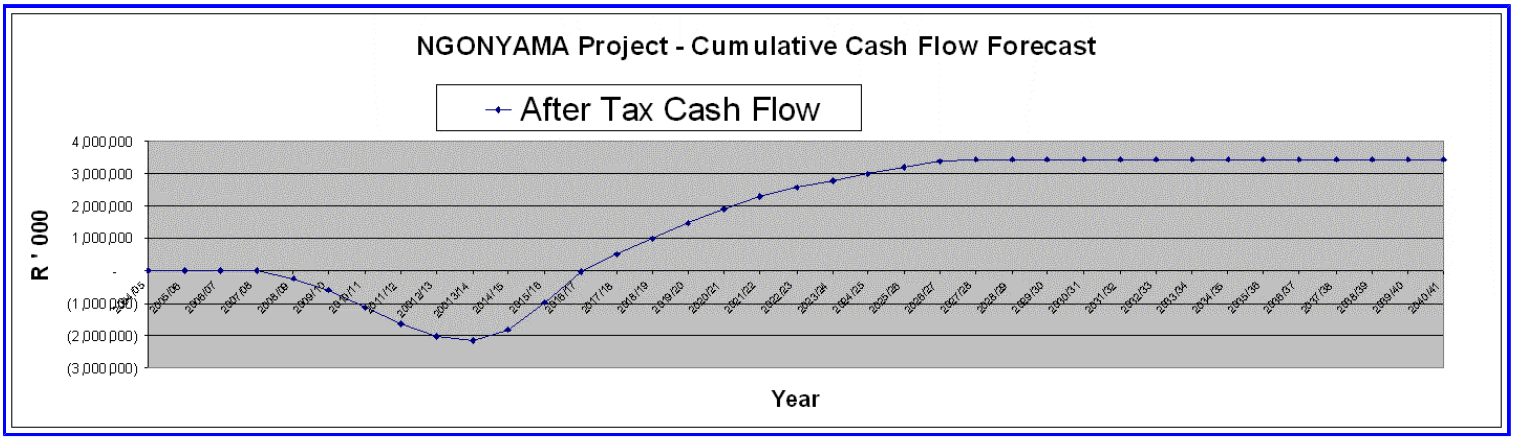
Pre-Tax Net Present Value @ Dec 2006	5.0%	10.0%	12.5%
R'000	6,609,284	**3,263,862**	2,280,390
Pre-Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	6,939,749	**3,590,248**	2,565,439
Pre-Tax IRR	**29.0%**		

After Tax Net Present Value @ Dec 2006	5.0%	10.0%	12.5%
R'000	3,737,771	**1,717,364**	1,125,248
After Tax Net Future Value @ July 2008	5.0%	10.0%	12.5%
R'000	3,924,659	**1,889,100**	1,265,904
After-Tax IRR	**22.9%**		

The improved metal price scenario indicates that the decline option which produces tons earlier, is to be re-assessed during the next phase of the study.

Item 25 (i): Payback

Given the above capital and operating costs for the vertical shaft option, with a revenue stream from the payable metals as indicated, the joint-venture partners expect payback 9 years and six months from the date of approval of the project. The following graph indicates the cash-flow position for the mine.



This graph also clearly indicates that the peak funding requirement will be R2,143 million.

Item 25 (j): Project risks and opportunities

The following table summarises the preliminary risks identified:

Geology	Grade	Project viability based on high-grade Merensky Reef to compensate for small deposit. If this grade is not there, there is no economy of scale to rescue the project.
	Structure	If the local structure is even worse than predicted, planned output will not be achieved and the mine infrastructure will have been over-designed.
	Losses	Greater then expected losses will have an abnormally large effect on this mine because of its already short life.

Mining	Stoping width	Inability to mine at the predicted width will cause extra dilution. Not a serious problem as the mine infrastructure can handle the tonnage.
	Support cost	If hanging wall conditions are worse than expected and support other than elongates is necessary the effects on costs and productivity will be significant.
	MCF	Inability to operate at the high MCF selected will have the same effect as lower than expected grade.
	Face availability	If face availability is lower than expected (mainly due to Geological complexity) then individual units and therefore the whole mine will under produce despite having paid for the greater infrastructure.
	Suitable labour	This proposal uses a labour intensive mining method as dictated by the deposit. It assumes that the necessary labour will be available, either contract or mine labour. If not, the outcome is catastrophic as the reef does not lend itself to mechanized methods.
	Development rates	Achieving development rates as planned is crucial to the face availability issue above. If not achieved the mine will under produce.
	Ventilation	Higher than anticipated virgin rock temperatures may require the use of refrigerated systems
	Busy stations/levels	Due to the complexity of the geology and the duplication of reef on certain levels, the mine plans now include certain shaft levels that must service up to 40,000 reef tons per month. These logistics have not been assessed.
	Ramp up in tonnage	Inability to achieve the desired ramp up in tonnage from the mine.
	Structure	Complex structure model.
Shaft systems	Over designed	Limited risk of non-achieving.

Surface	Water supply	The area where the mine is located is relatively dry and poorly drained. The ground water resource is limited. All water for the project is to be piped in from a long distance. This is always a risk for such a project. The mine design is to collect as much water as possible and retain it on the property for future use. Grey water supply may be considered.
	Power supply	South Africa is in a power crisis and whilst ESKOM are confident that they will be able to meet all future demands, the growth in the mining and other industries is putting a significant strain on the power network in the country. This is regarded as a sustained risk to the project.
	Housing	Gate wage principle applied. Must be assessed to appreciate the full implication.
	Change houses	Large labour force to pass through modern industrialized change house system
	Shaft position	Hotel, etc are located very near by and there could be conflict and the site may need to change from the current ideal position for the deposit.
	Fans	Noise factor.
	Waste dumps	Location and dust.
	Slimes dams	Location and potential dust.
Legal	BEE	BEE partner.
	Local community	Local community involvement.
	Licences	Granting of licences/permits.
Metallurgical	Tonnage	Inability to process the required tonnage.
	Grade	Inability to achieve the desired concentrate grade.
	Recovery	Inability to achieve the desired recovery.
	Penalty grades	Inability to achieve the desired penalty element grades.
	Amplats smelting contract	Amplats contract not beyond very early discussions and needs to be negotiated.
	Ramp up in tonnage	Ramping up of tonnage throughput not achieved at desired recovery and grade.
Project Scheduling	Learning period	Learning period (as per Amplats) i.e. a slower production builds up.
	Sinking	Sinking delays due to factors outside of the control of the project.
	Ramp up of tonnage	The tonnage ramp up is not achieved.

	Equipment	Availability of equipment.
	Long lead items	Delivery and delays of long lead items.
Capex	Equipment	Major equipment price increases.
	Products	Escalation of all products.
	Steel	Steel price increases.
Opex	Wages	Wage increases.
	Consumables	Consumables price increases – escalation.
	Consumption	Increased consumptions of items – increased costs.
Revenue	Metal price	Conservative metal price estimation has been used and as such the risk is low.
	Penalty	Penalty escalation.
	Exchange rates	Volatility in markets may result in strengthening or weakening of the exchange rates with adverse effects on the financials.
	Penalty condition	Penalty condition changed.
	Penalty achievement	Non-achievement of levels.

NB - As this will be a low-cost producer of platinum-group metals, many risks will be ameliorated. These risks could also become future oppertunies if proved incorrect.

There are a significant number of opportunities that can be considered for the WBJV Project 1 including:

- metal price improvements and exchange rate weakening
- potential for recovery of nearby resources once the shaft is in position and in operation
- improvement in the structural complexity with reduced mining costs
- more rapid sinking of the vertical shaft than anticipated
- fast track access to the shallow ore body

These strengths and weaknesses will be further reviewed during the Bankable Feasibility study.

Item 25 (k): Life of mine and project schedule – vertical shaft

The life of the mine will be 12 years at peak production with a two- to three-year ramp-up and a two-year tail, resulting in maximum production life of 15 years. The Merensky Reef profile is at the peak for seven years, followed by a two-year period of Merensky mixed with UG2 and then three years of UG2 only.

If the decision to proceed is given in July 2008, the mine will start production in 2013 with milling commencing in 2014. The mine will cease production in 2027 with an exhausted resource.

Schedule

The shaft-sinking schedule involves 10 months of preparatory work with regard to design and site establishment, followed by 38 months on site.

Following the detailed design work, the concentrator will take between 18 and 24 months to construct. The milling plant will have the longest lead-time, with deliveries currently at approximately 78 weeks. The plant needs to be operational by Month 1 of Year Six and thus the plant construction contract is likely to have been awarded by Month 6 of Year Three – preferably six months sooner.

The only remaining long-delivery item is the supply of electric power from the ESKOM grid. Currently this process can take in excess of 24 months to complete, but the matter is not seen as critical since temporary power will be installed ahead of time for the shaft-sinking activities.

The decline option assumes that there is spare processing capacity within road transport distance and that the ore-processing contract could be finalised. Currently there is limited processing capacity within economical road transport distance.

Item 25 (l): Conclusions and recommendations

After considering a number of access options, the QP and other experts involved in this study recommend the shaft project outlined above as having the best IRR and NPV at long-term metal prices. This study can be assumed accurate to the following degree for the mentioned factors:

- Capital Cost Estimate – ±25%
- Operating Cost Estimate – ±25%
- Project Timing Estimate – ±20%
- Project Output Estimate – ±20%

It is the recommendation of Turnberry Projects, the QP and experts involved in this study, that a Bankable Feasibility study be commissioned to improve the accuracy of the above estimates and progress the project to the next phase with the agreement of all WBJV partners. As part of the BFS, the decline option will be re-investigated to reassess its contribution to the project finances.

To the understanding of Turnberry Projects, the detailed scope of work and budget for the BFS would be developed in consultation with a committee of the WBJV, formed specifically for the development of a final Feasibility study as specified in the agreement. This committee will consider the risks and opportunities of the Pre-feasibility study and the objectives of the partners including, but not limited to the

- timing;
- profile of production;
- return hurdles; and
- partners' own short- and long-term planning.

Turnberry Projects welcomes the opportunity to assist in developing this scope of work.

ITEM 26: ILLUSTRATIONS

Diagram 1: Setting of the Bushveld Igneo

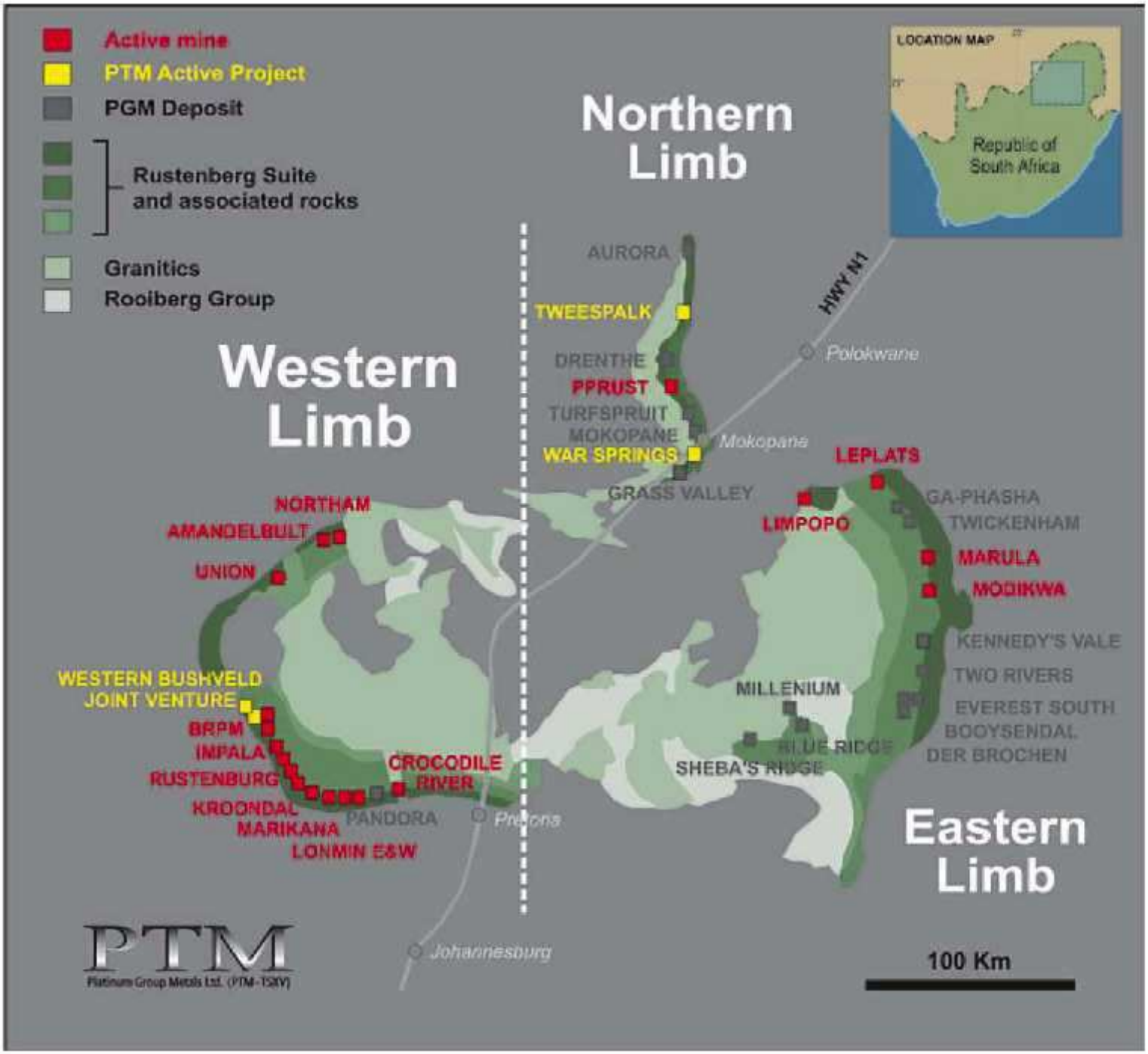
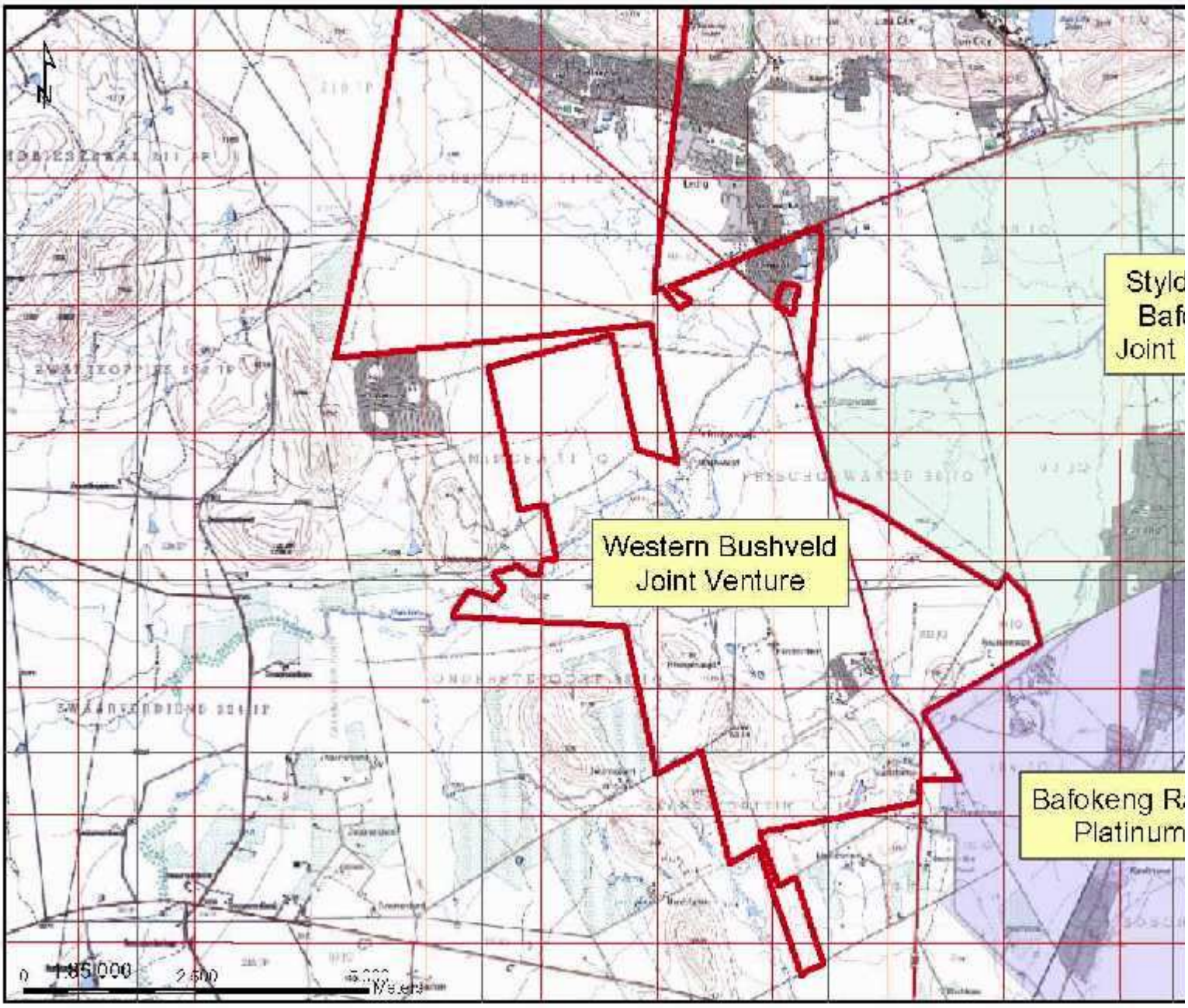


Diagram 2: Locality Plan of the



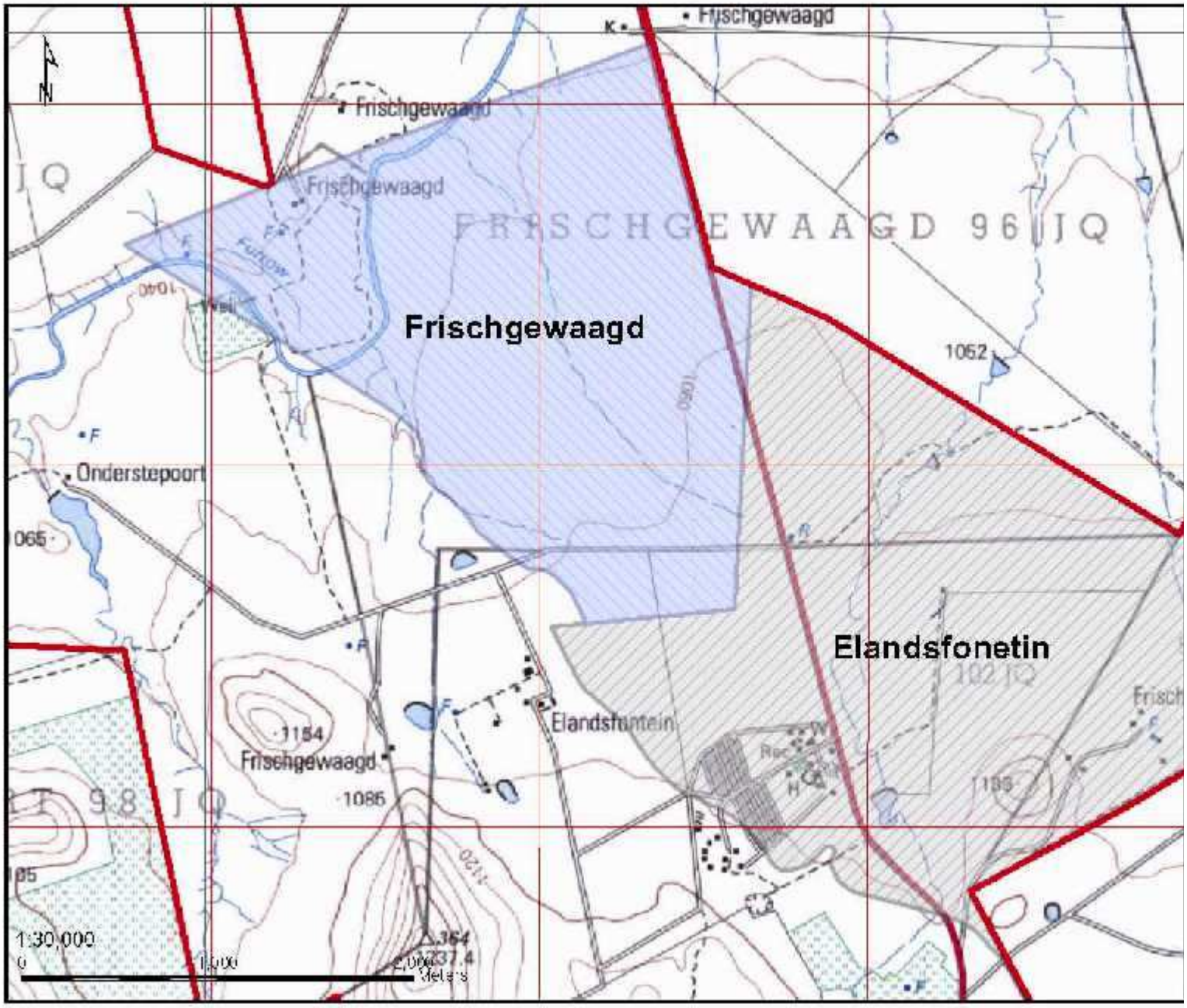

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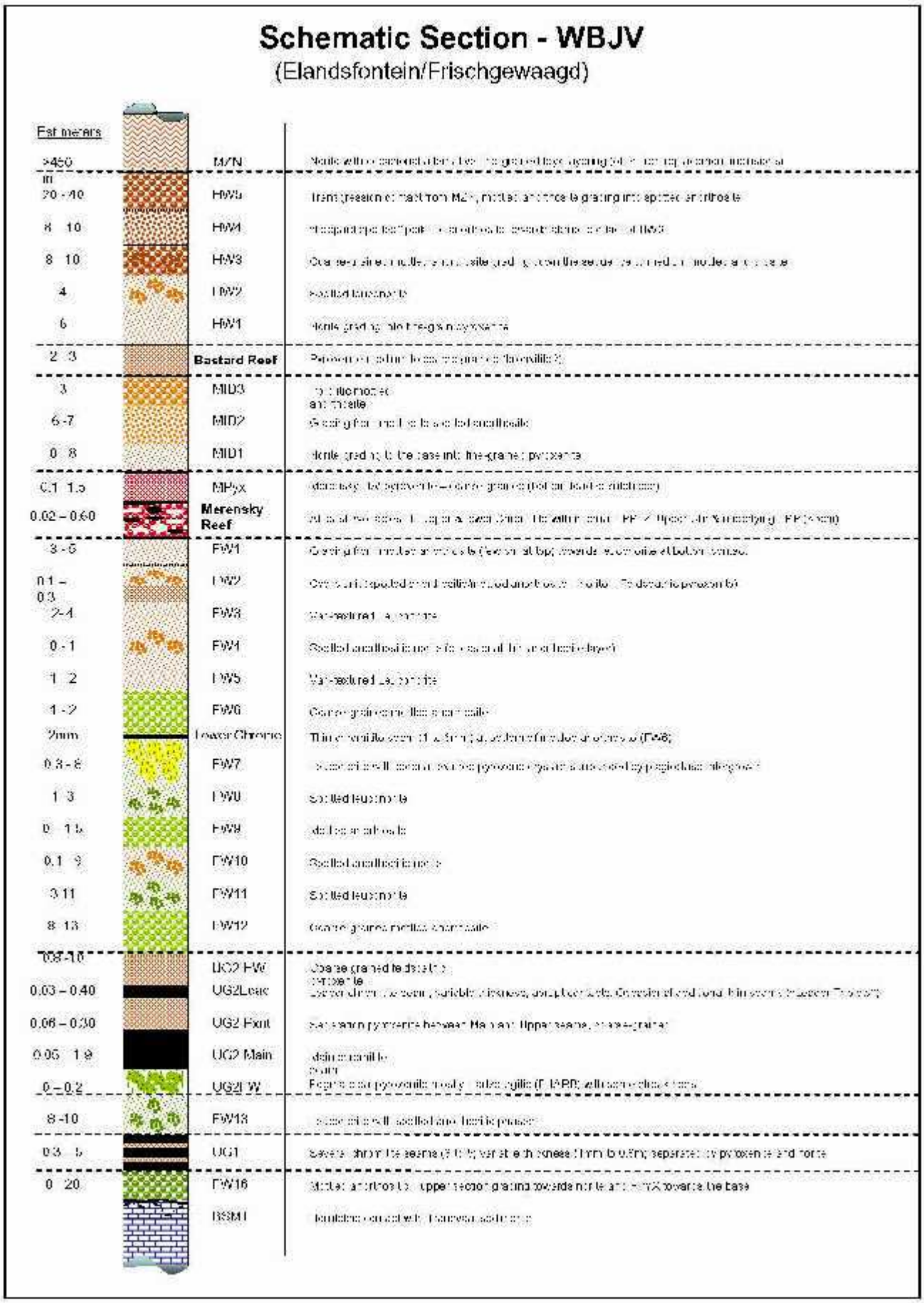
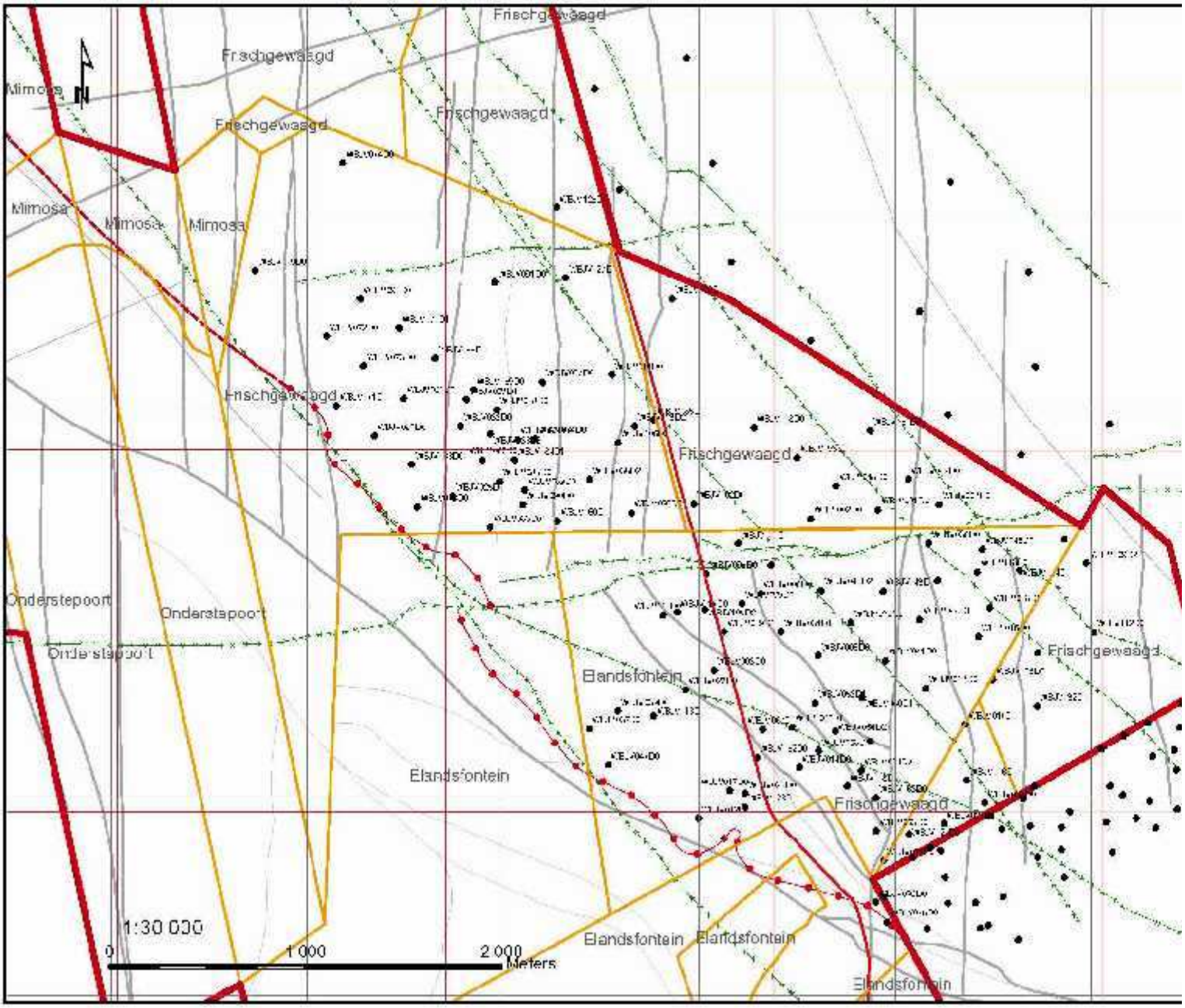
Diagram 4: Detailed Stratigraphy of the Western Bushveld Sequence.

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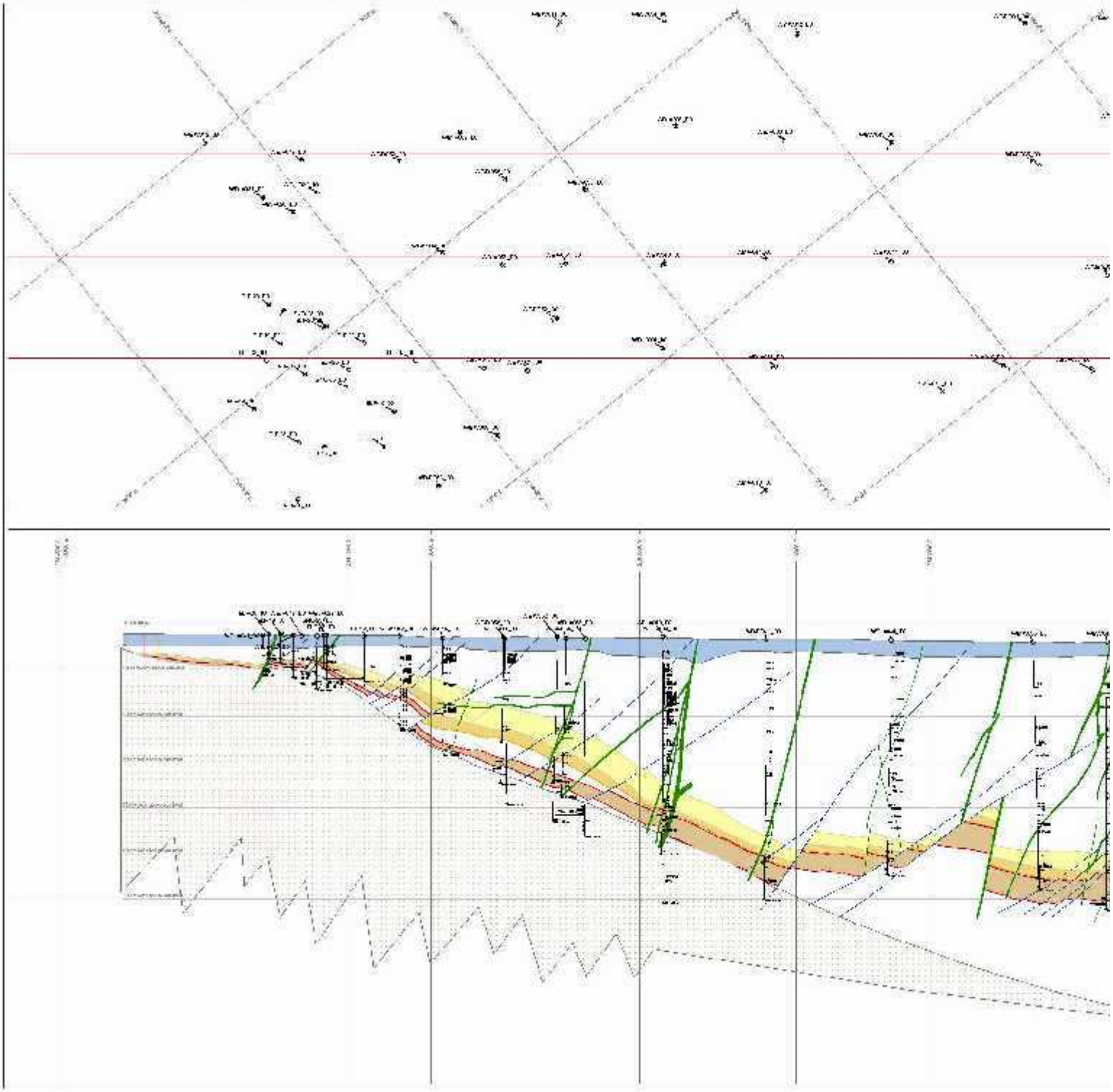

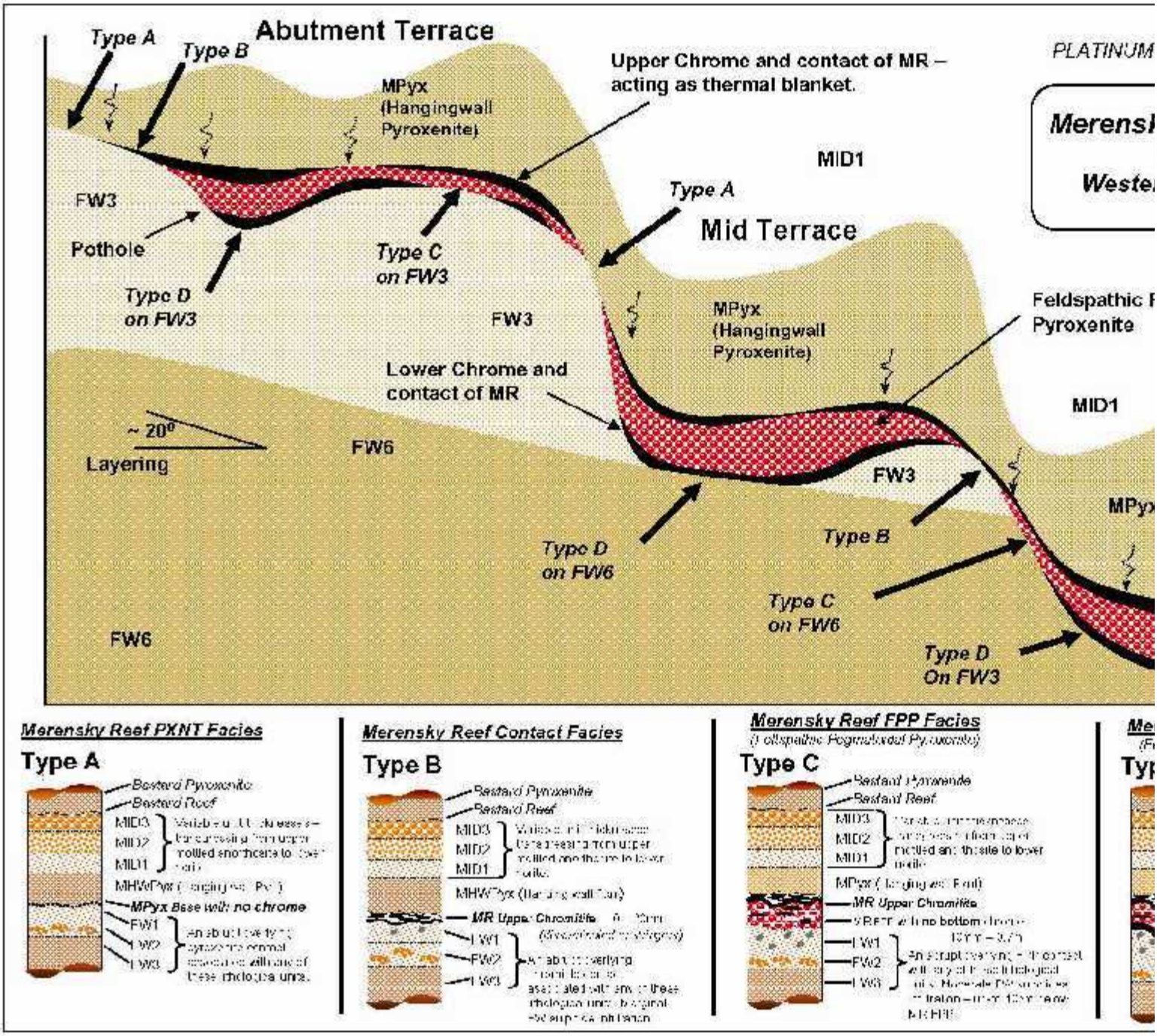
Abutment Terrace
Type A
Type B
MPyx
(Hangingwall
Pyroxenite)
Upper Chrome and contact of MR –
acting as thermal blanket.
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MID1
FW3
Pothole
Type A
Mid Terrace
Type C
on FW3
Type D
on FW3
FW3
MPyx
(Hangingwall
Pyroxenite)
Feldspathic
Pyroxenite
Lower Chrome and
contact of MR
MID1
~ 20°
Layering
FW6
FW3
Type D
on FW6
Type B
Type C
on FW6
FW6
Type D
On FW3
Merensky Reef PXNT Facies
Type A
Bastard Pyroxenite
Bastard Reef
MID3
MID2
MID1
MHWPyx
MPyx Base with no chrome
FW1
FW2
FW3
Merensky Reef Contact Facies
Type B
Bastard Pyroxenite
Bastard Reef
MID3
MID2
MID1
MHWPyx
MR Upper Chromitite
FW1
FW2
FW3
Merensky Reef FPP Facies
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MPyx
MR Upper Chromitite
FW1
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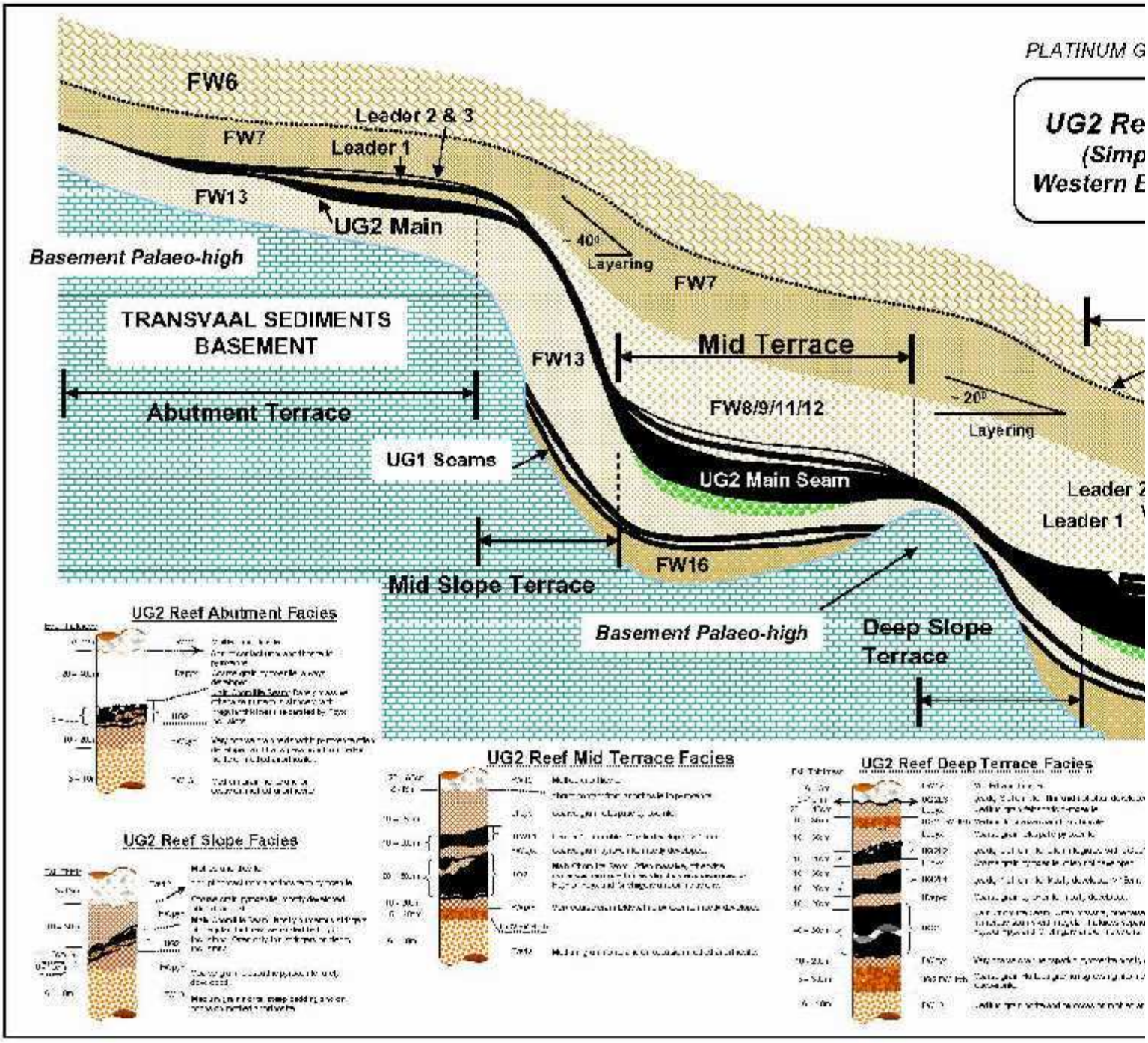
FW3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FW7
TRANSVAAL SEDIMENTS
BASEMENT
Mid Terrace
FW13
Abutment Terrace
FW8/9/11/12
~ 20°
Layering
UG1 Seams
UG2 Main Seam
Leader 2
Leader 1
FW16
Mid Slope Terrace
Basement Palaeo-high
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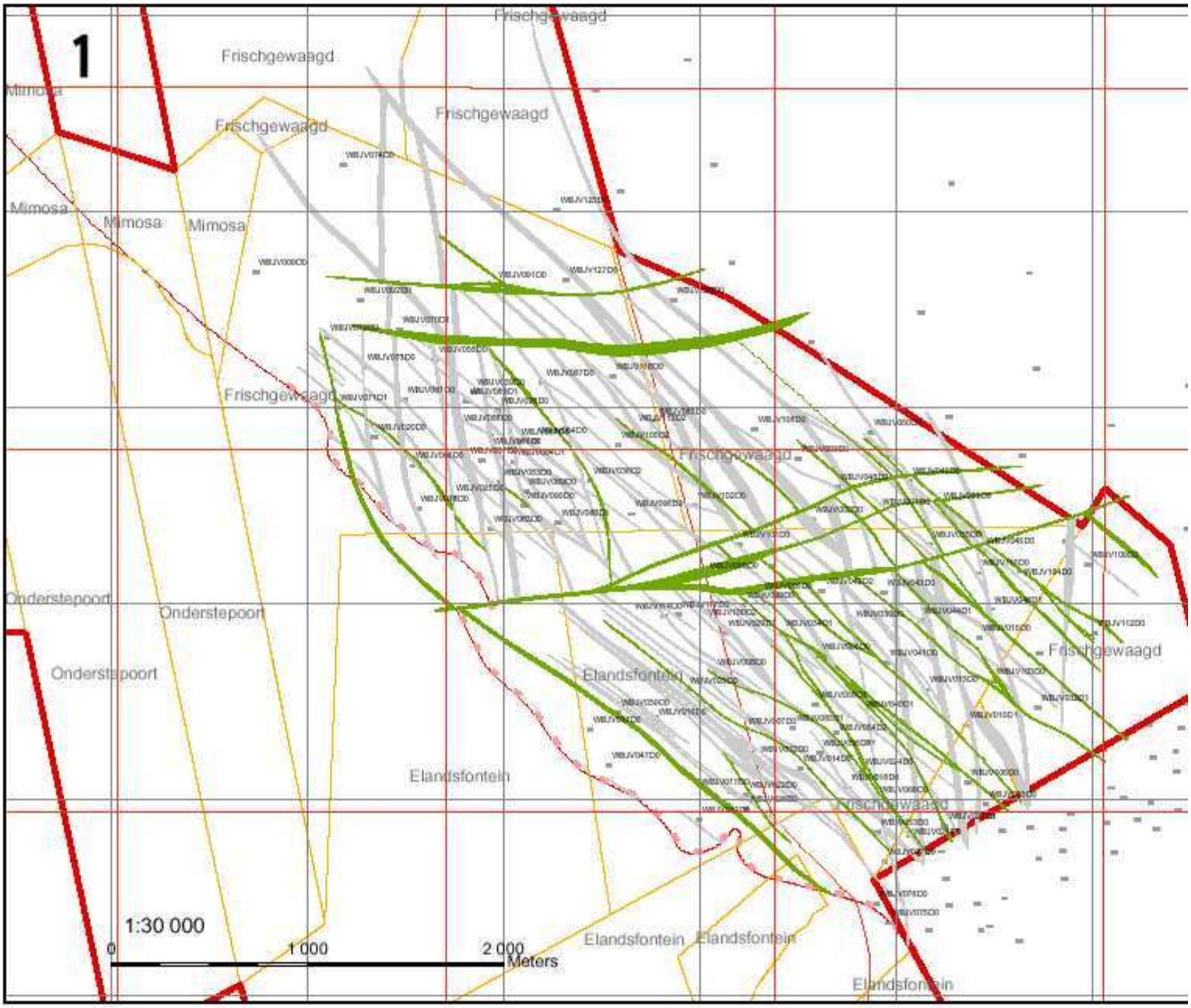
UG2 Reef Abutment Facies
UG2 Reef Slope Facies
UG2 Reef Mid Terrace Facies
UG2 Reef Deep Terrace Facies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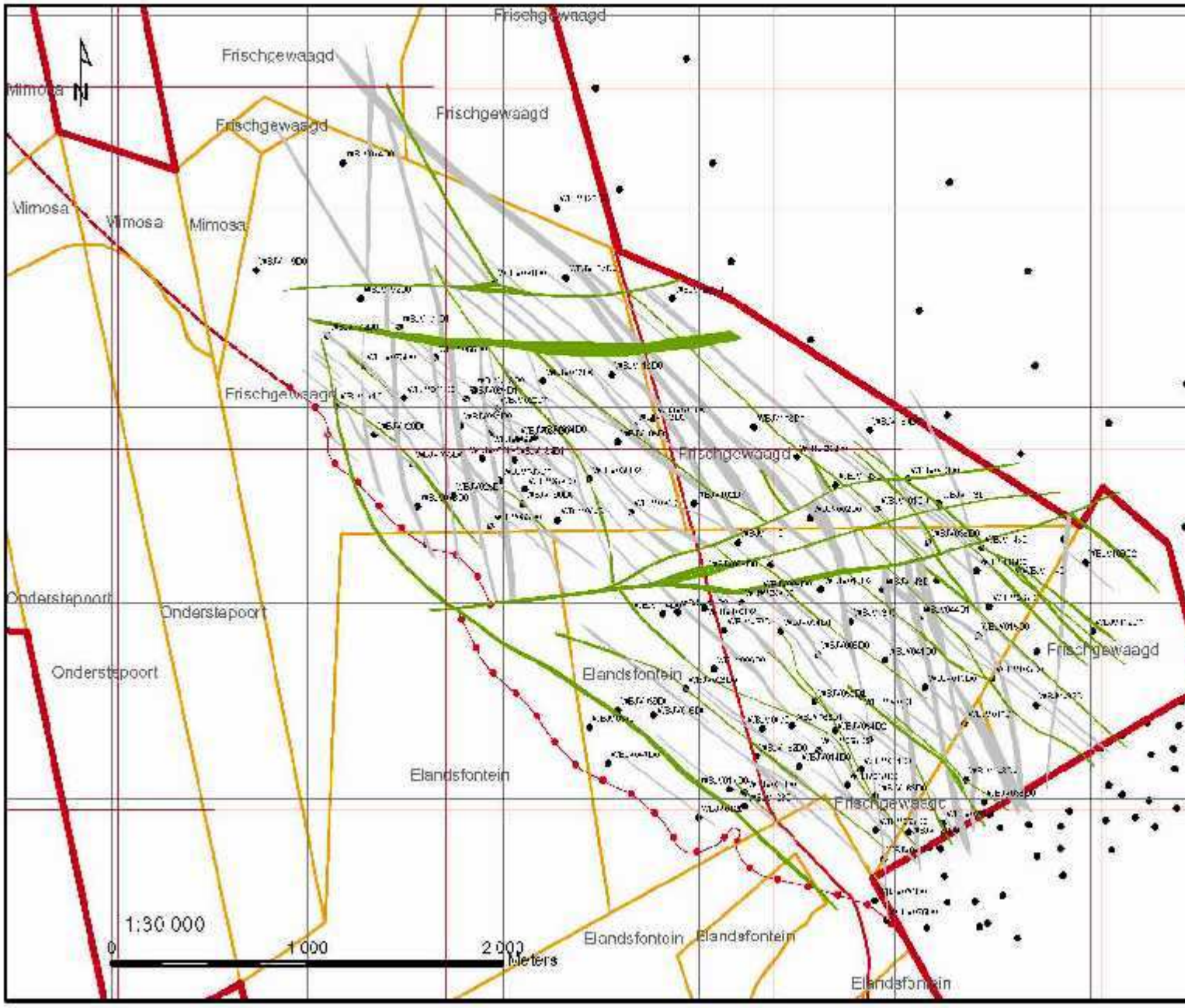
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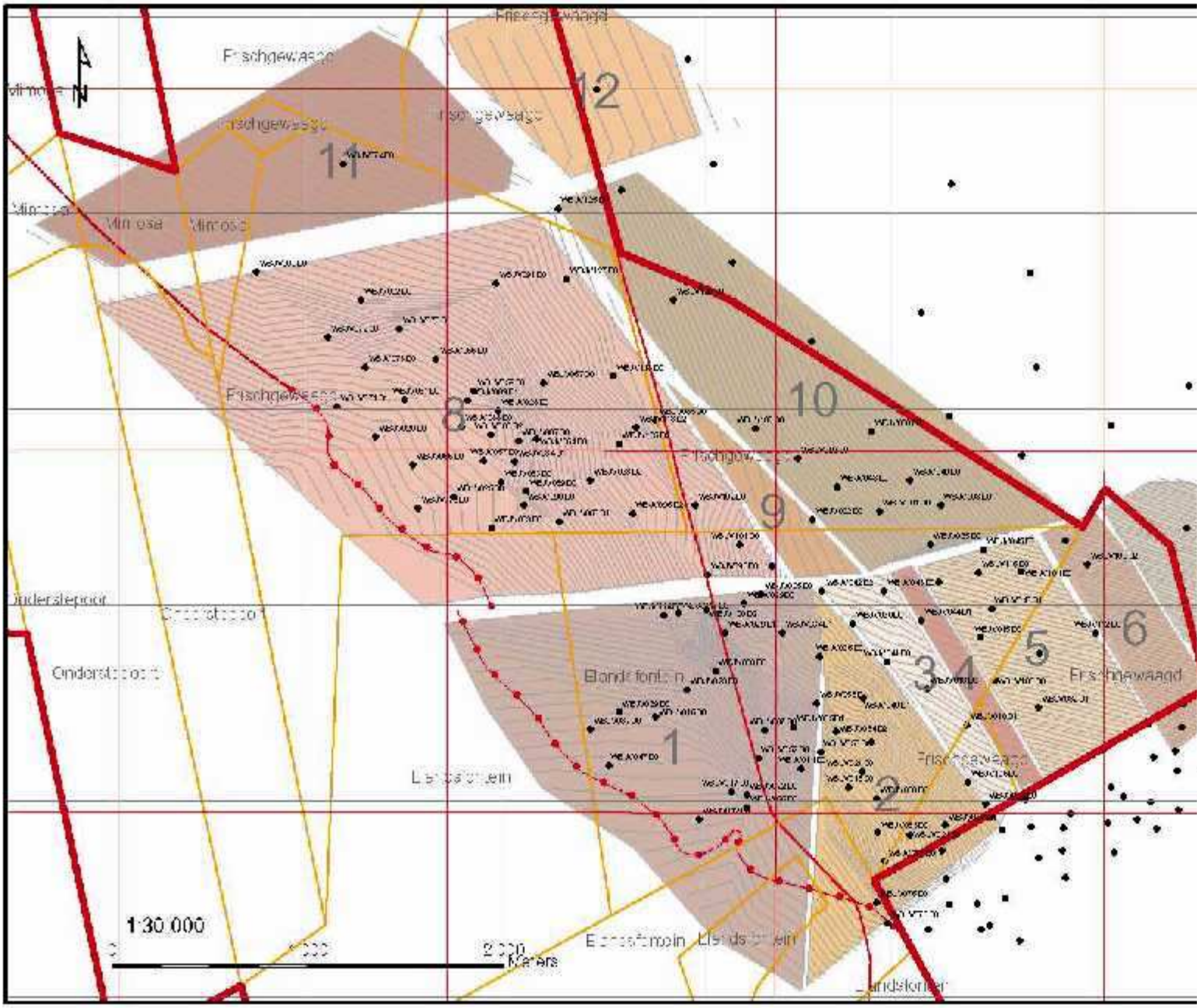
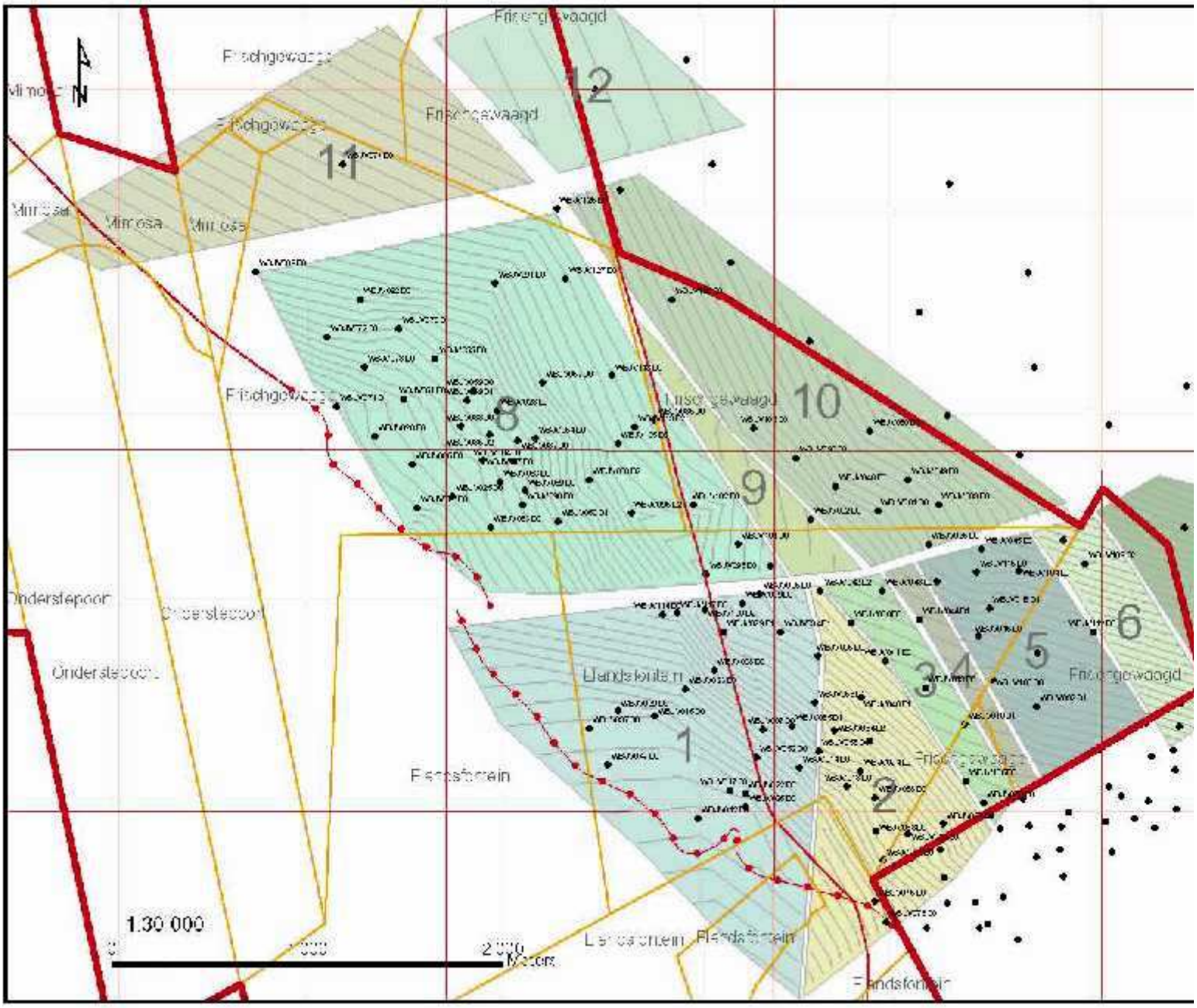
Diagram 10a: M

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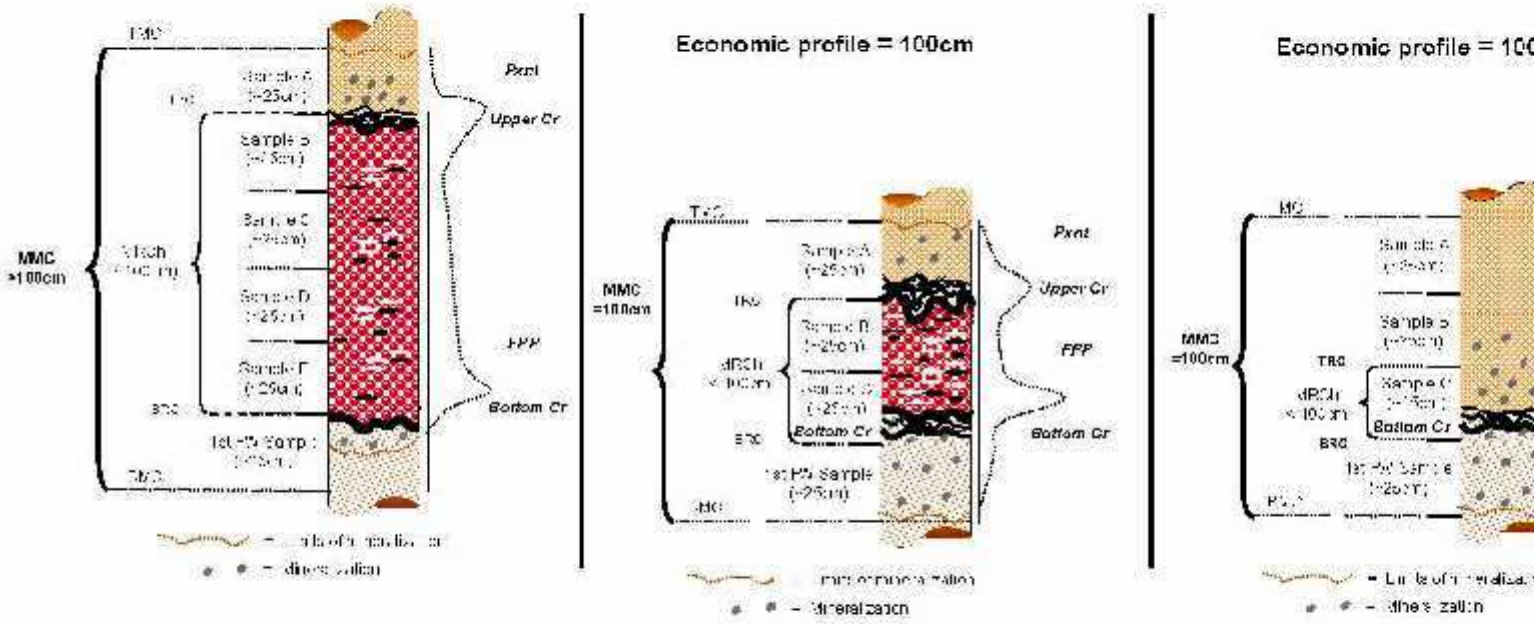
Merensky Reef Mining Cut Model



Mining Cut Selection

1. Bottom chrome or bottom of FPP (in case of bottom chrome not developed) is taken as Merensky Reef Bottom.
2. Select 1st FW sample, whether visible sulphide infiltration is present or not, as the BMC sample. The bottom part of sample may not be mineralized. Only the first FW sample is included even if there are others mineralised.

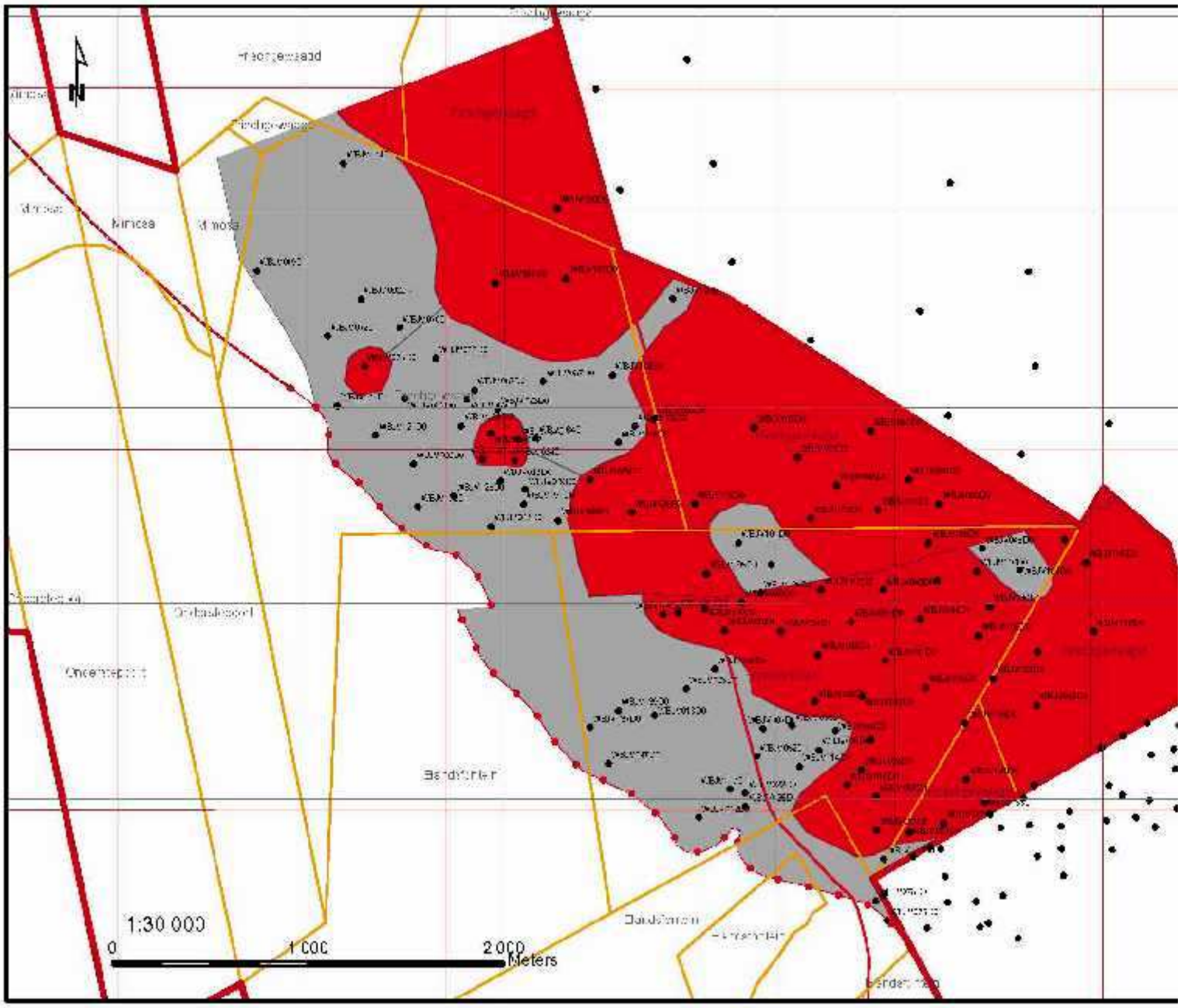
- TRC = the top of the MRCh within the Merensky HW Pxnt and top limit of mineralization.
- BRC = The bottom of the MRCh within Merensky FW and the bottom limit of mineralization.
- MRCh = selection between TRC and BRC

3. In the case of:

- MRCh > 100cm: select MRCh (including 1st FW sample and last sample that includes the TRC).
- MRCh < 100cm: select MRCh (including 1st FW sample and additional HW samples to 100cm).
- MRCh = 100cm: select MRCh

Terminology

- MRCh = Merensky Re
- TRC = Merensky Top
- BRC = Merensky Bott
- TMC = Top Mining Cu
- BMC = Bottom Mining
- MMC = Merensky Min
- FPP = Feldspathic Pe
- Pxnt = Pyroxenite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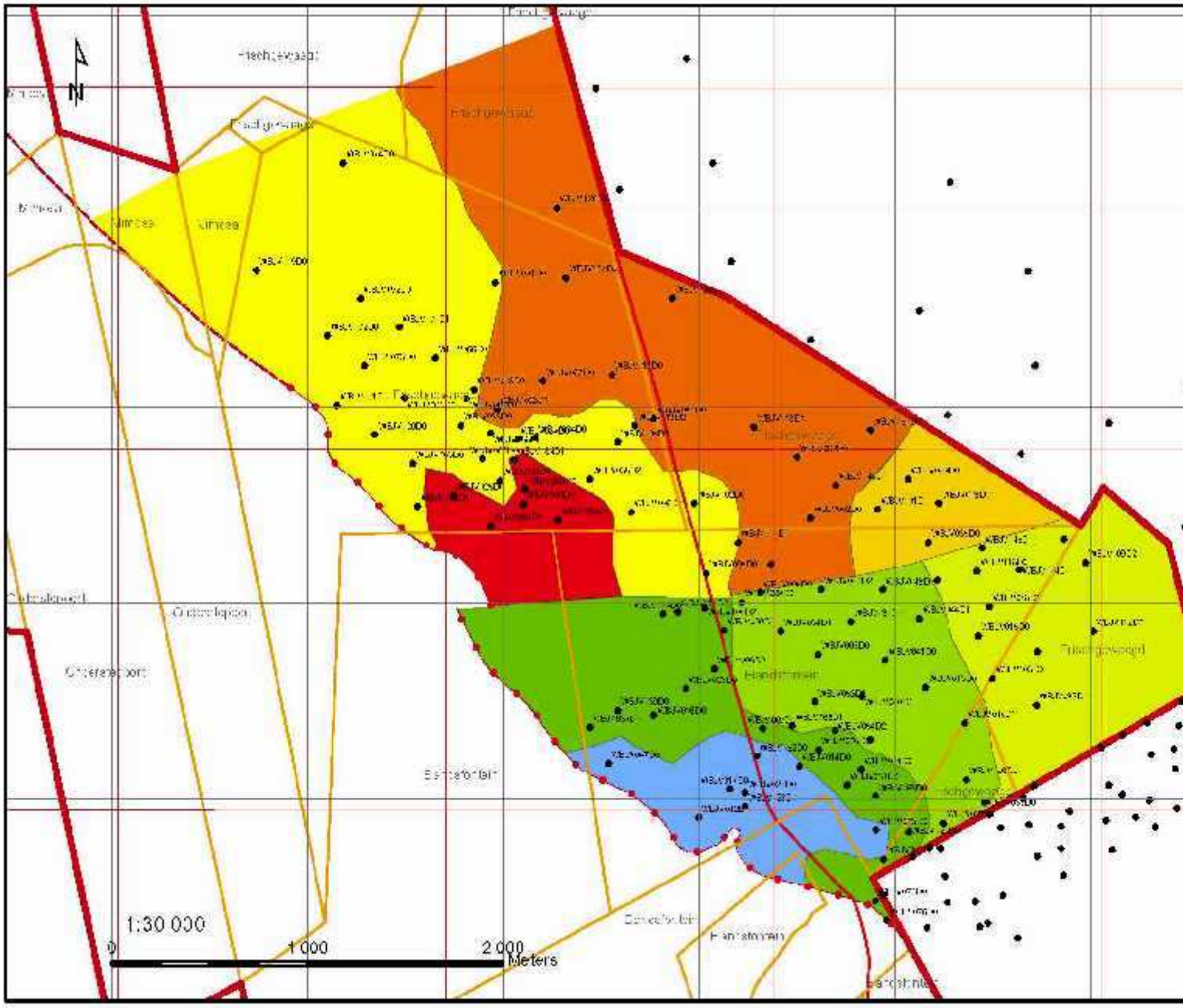
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Diagram 17: Meren

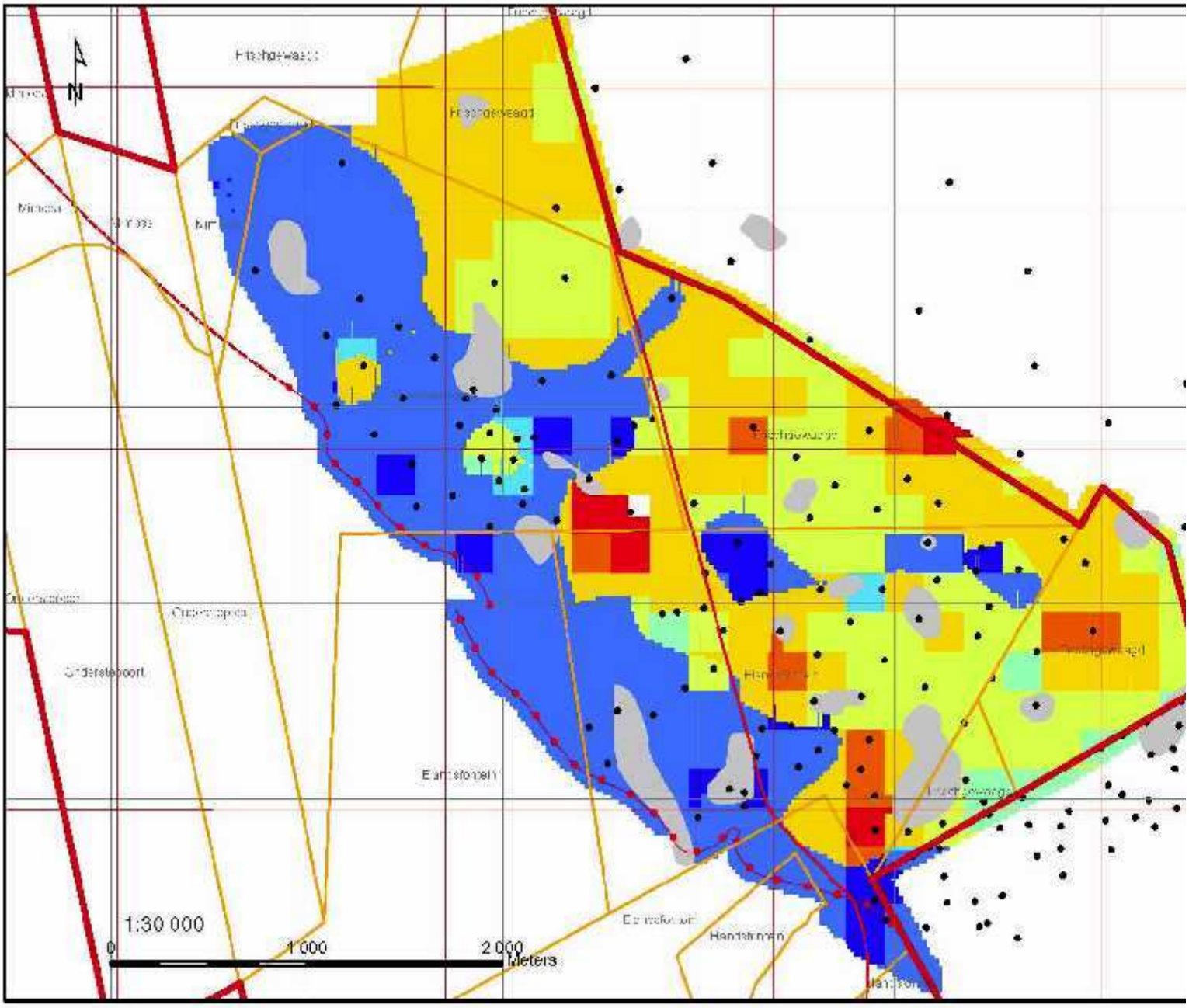


Diagram 18

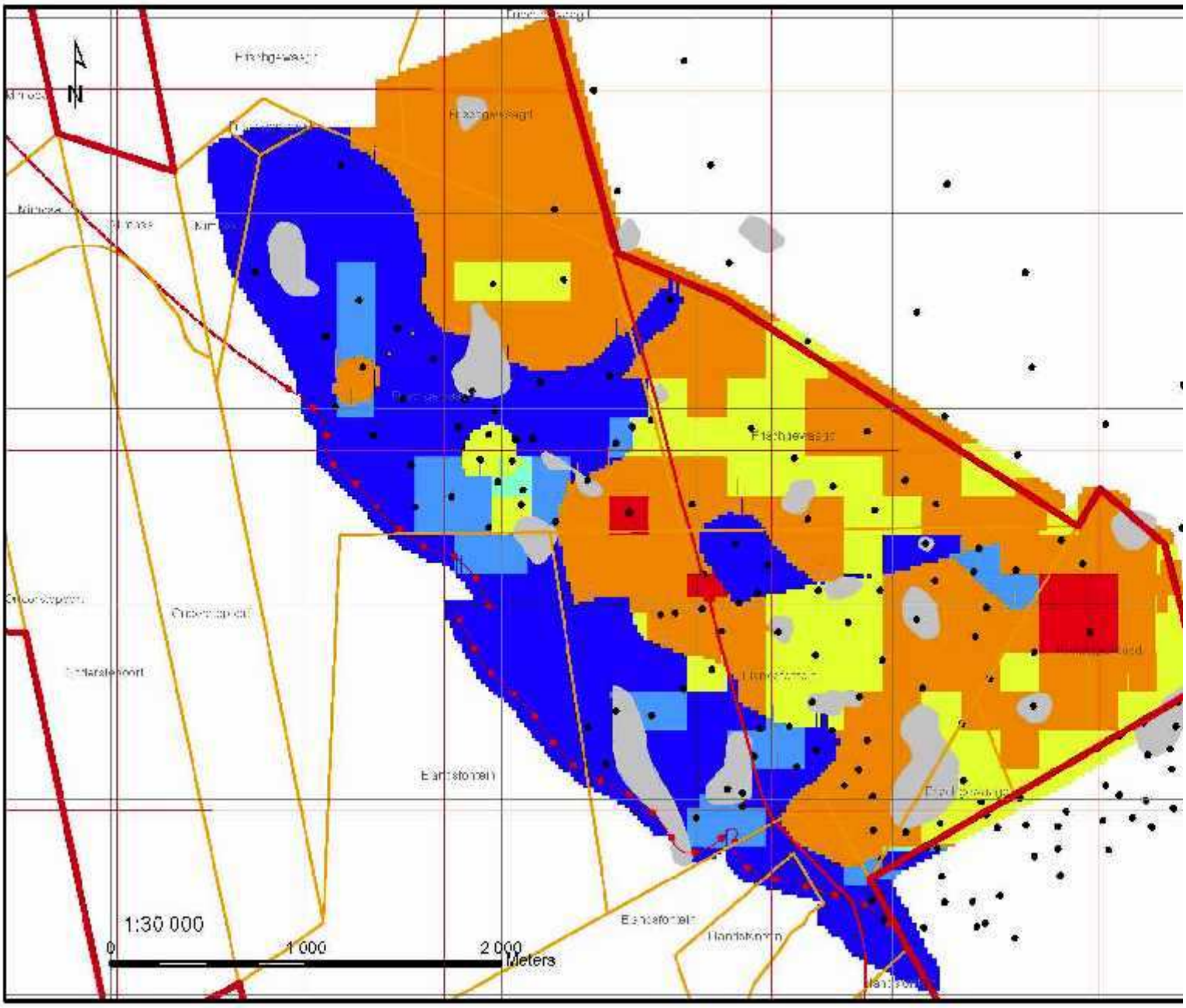


Diagram 19: M

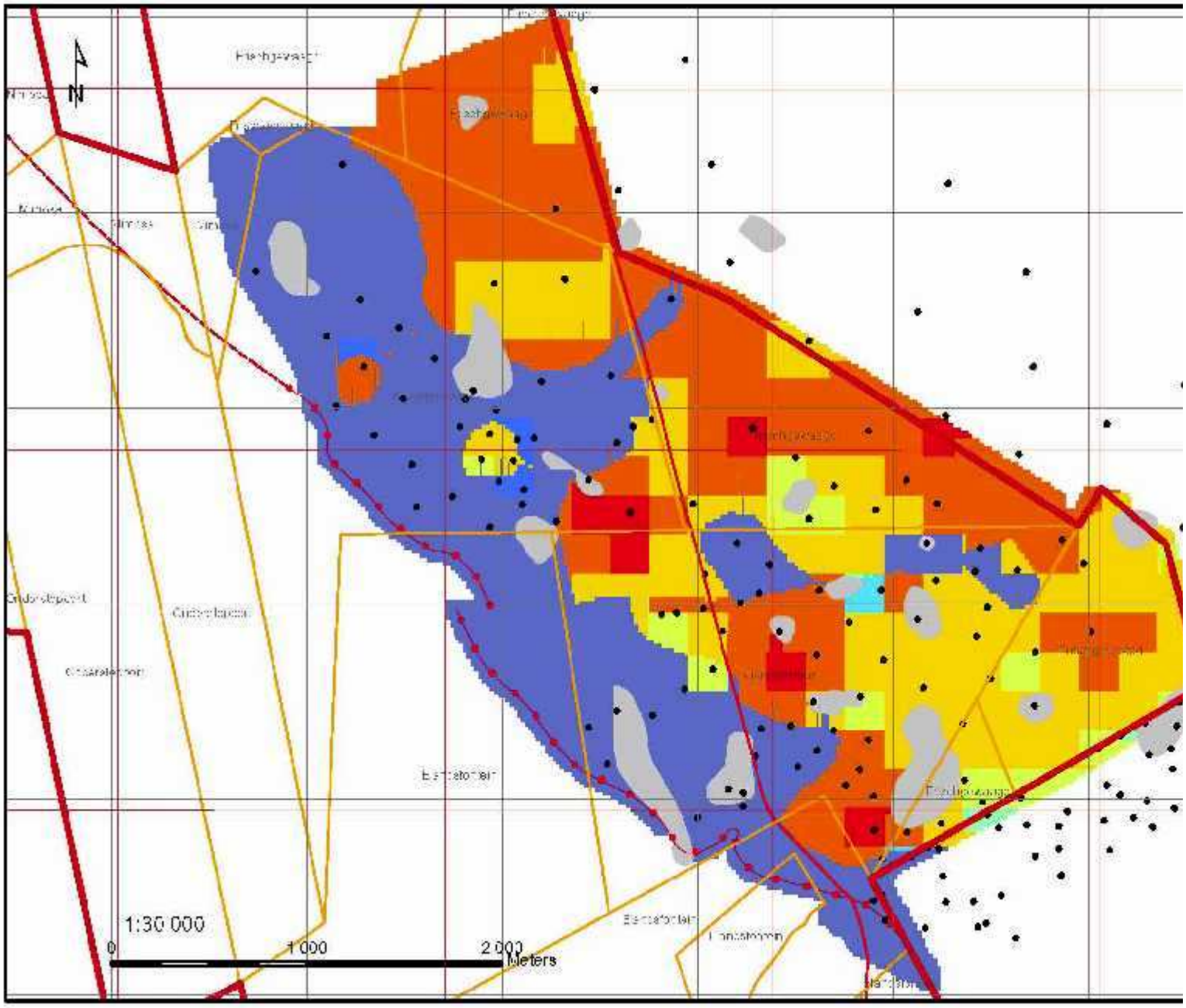
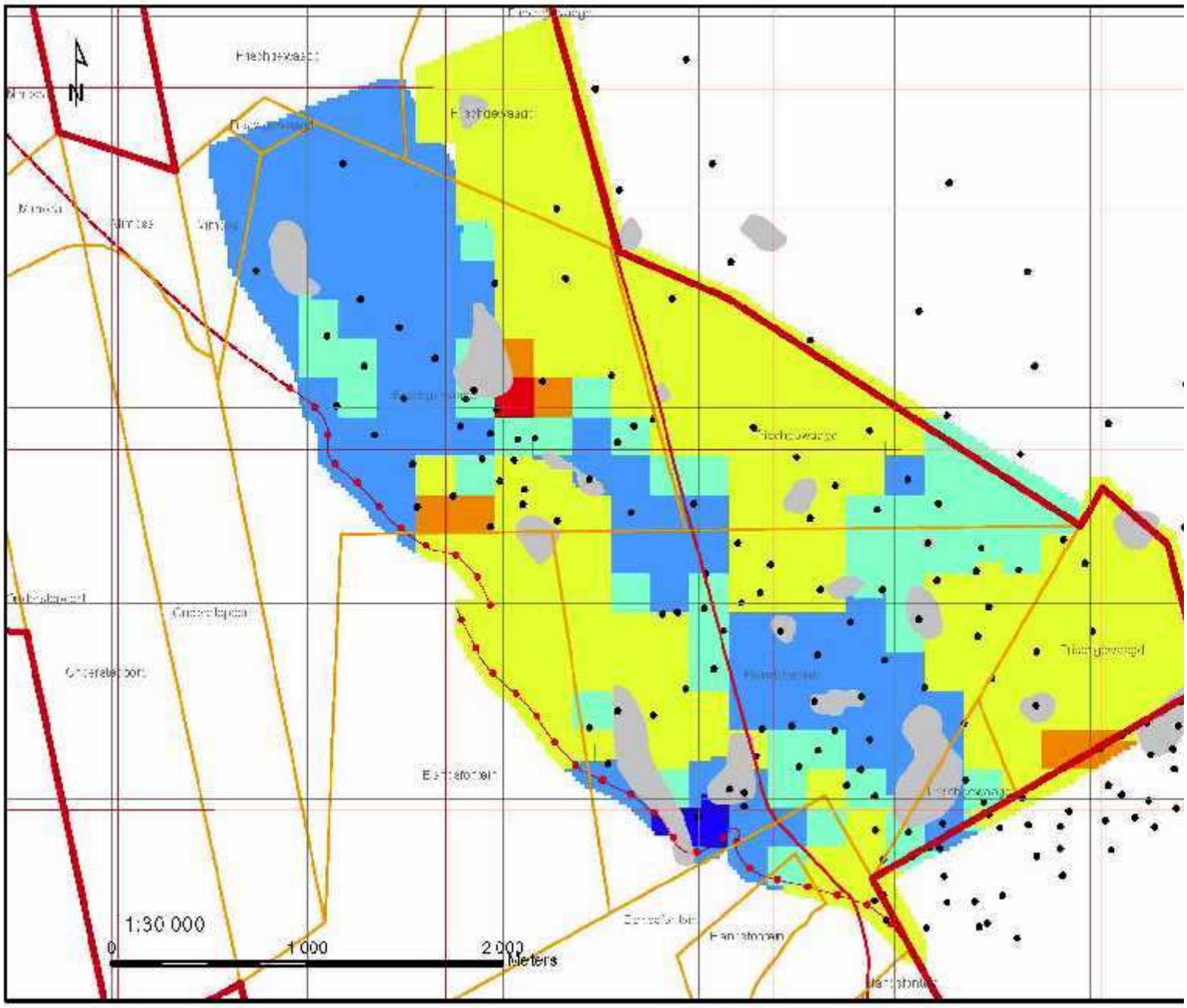


Diagram 20:



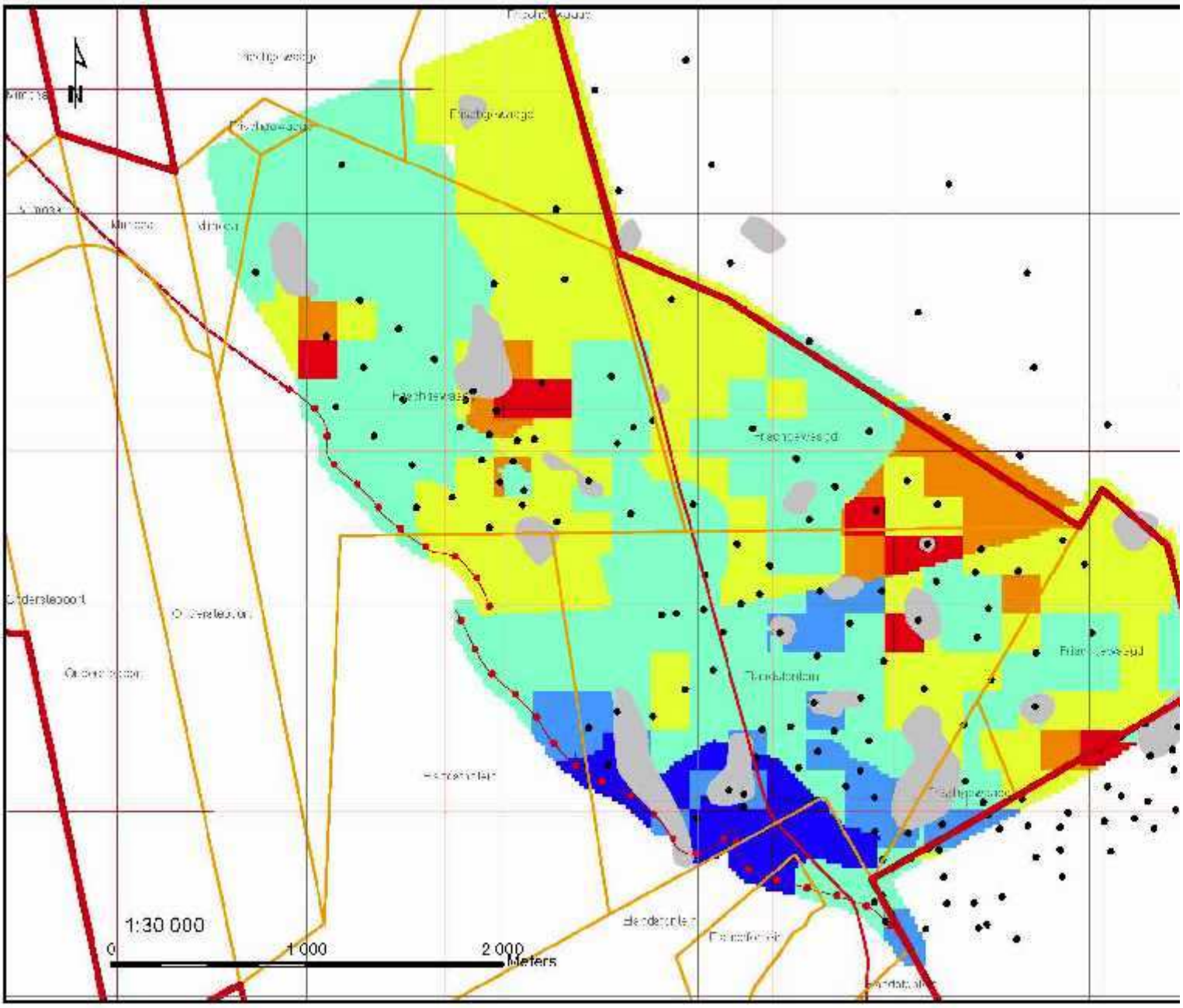

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Diagram

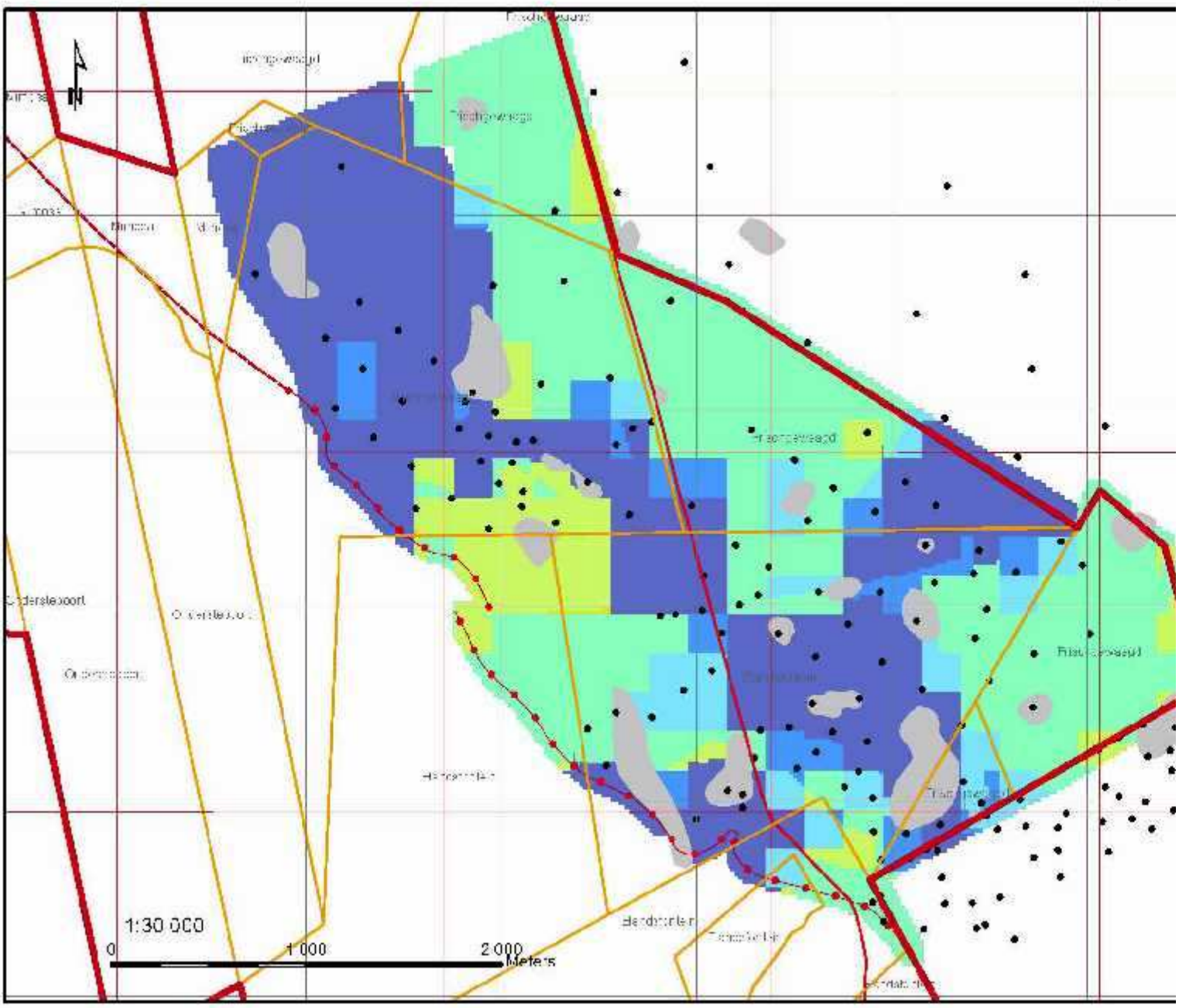

Diagram 23: Merens

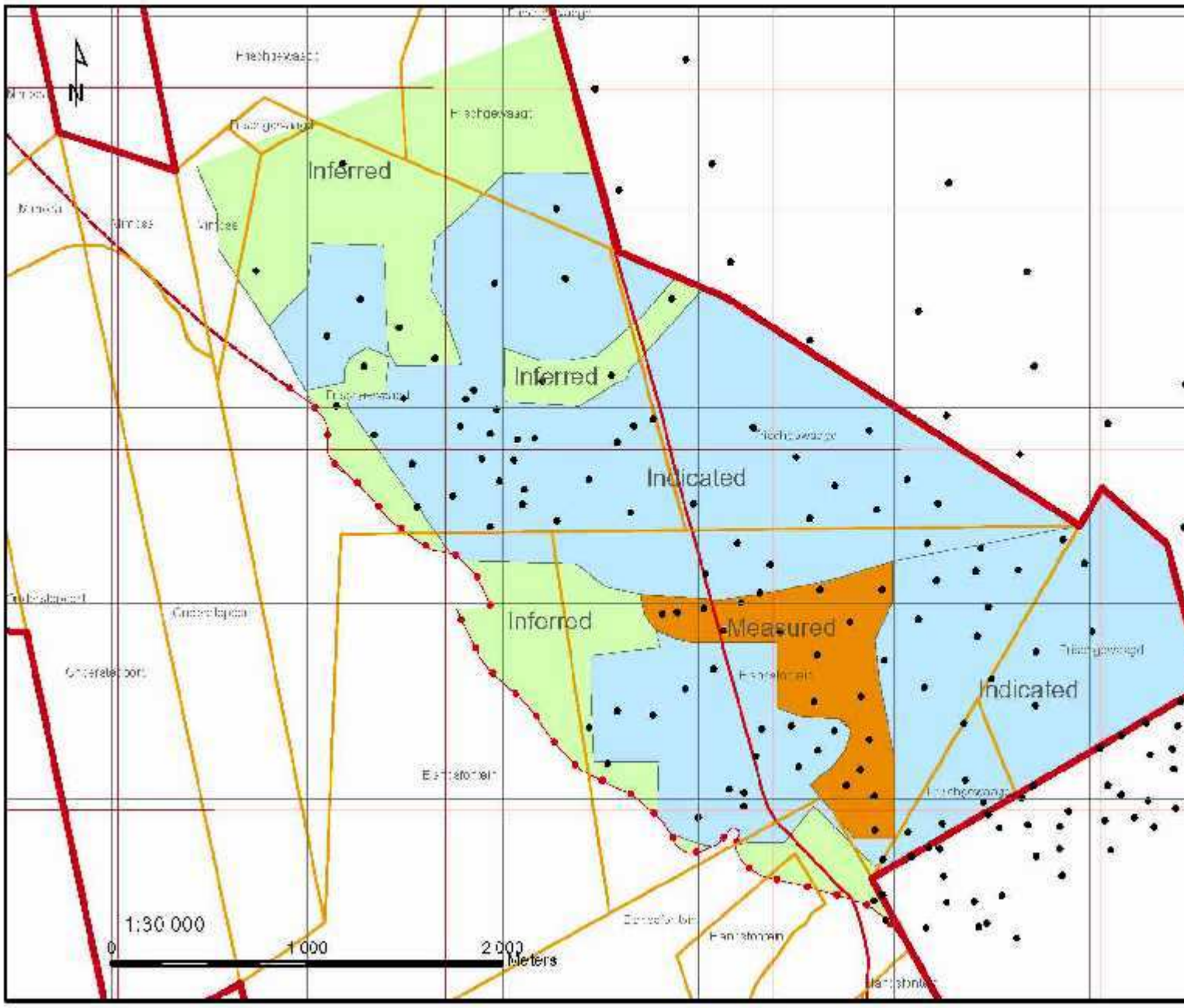

Diagram 24: U

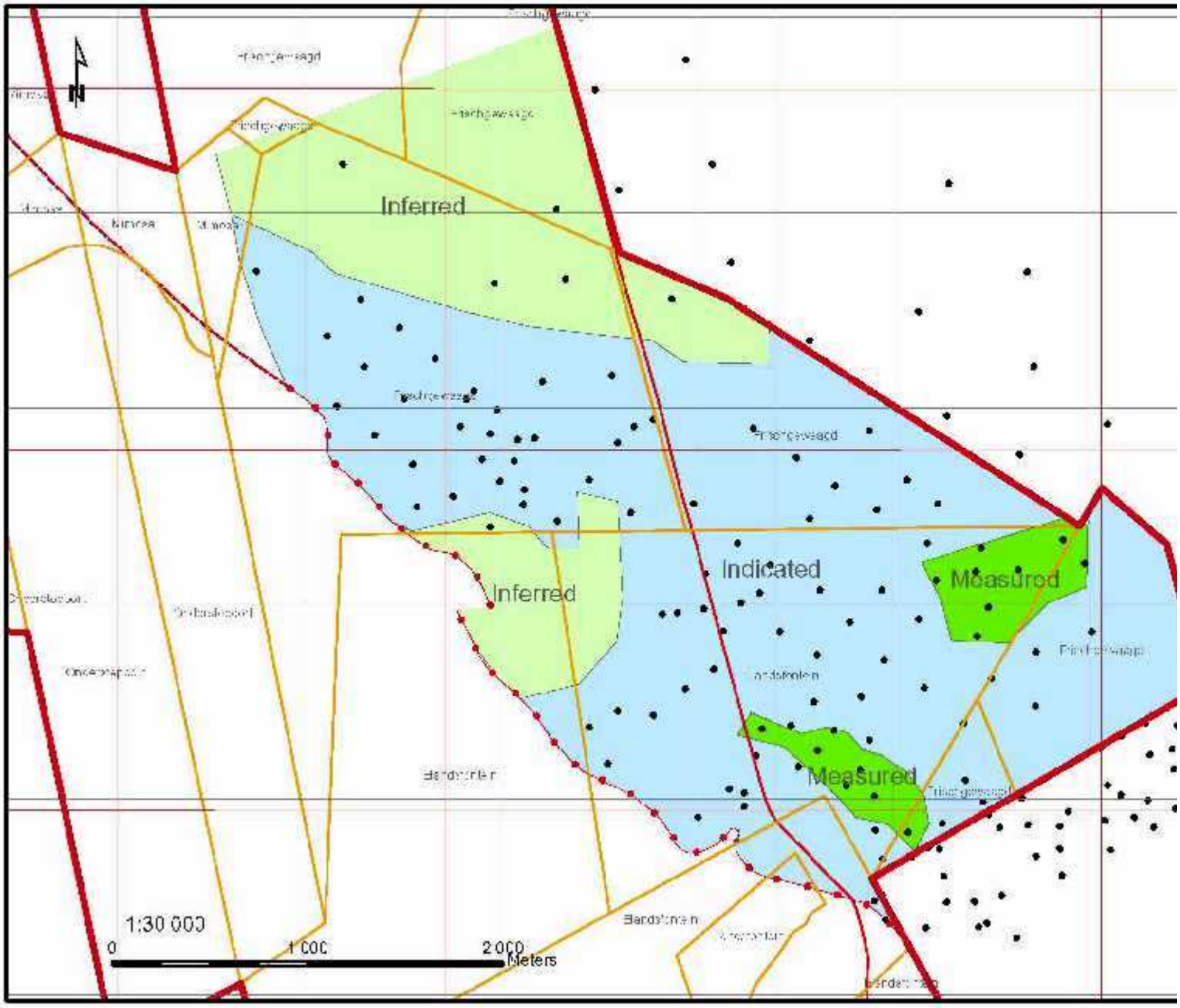
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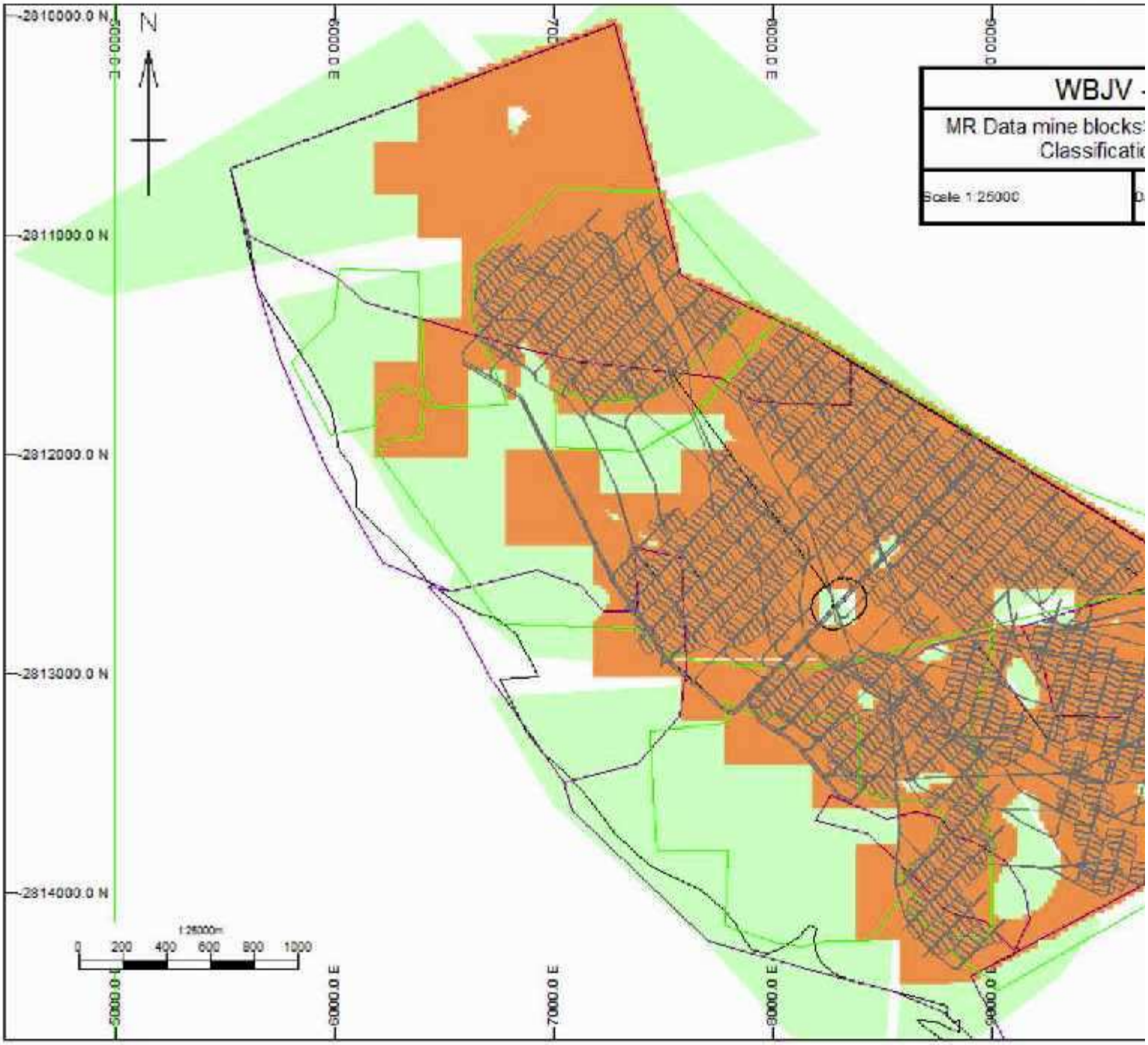
Diagram 2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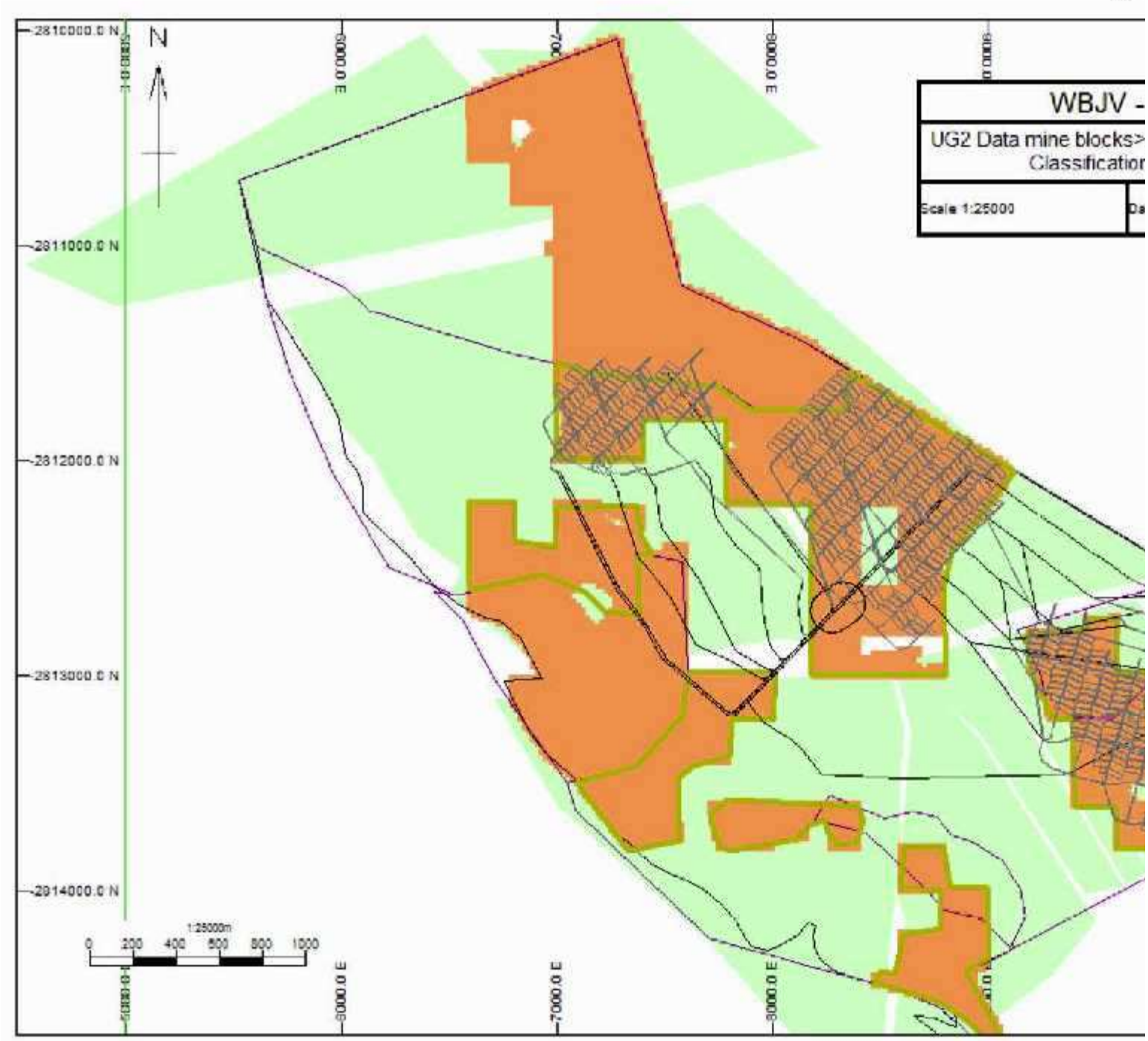

APPENDIX A

Table 1A: Merensky Reef Mineralised Intersections.

BHID	FROM(m)	TO(m)	LENGTH	CBA	PT(g/t)	PD(g/t)	RH(g/t)	AU(g/t)	4E(g/t)	REEF	VALID
WBJV001D0	447.60	448.65	1.05	15	2.74	1.26	0.17	0.16	4.33	MRMC	Pass
WBJV001D2	27.94	28.93	0.99	15	3.09	1.41	0.18	0.28	4.96	MRMC	Pass
WBJV002D0	464.62	465.91	1.29	15	3.38	1.66	0.20	0.36	5.60	MRMC	Pass
WBJV002D1	14.63	16.15	1.52	15	1.72	0.98	0.11	0.18	2.99	MRMC	Pass
WBJV006D0	459.98	460.98	1.00	15	10.05	4.53	0.56	0.45	15.59	MRMC	Pass
WBJV006D1	96.69	97.69	1.00	15	9.52	4.95	0.53	0.74	15.73	MRMC	Pass
WBJV008D0	243.00	244.23	1.23	15	1.23	0.58	0.09	0.11	2.00	MRMC	Pass

WBJV008D1	19.48	20.52	1.04	15	0.49	0.28	0.01	0.10	0.88	MRMC	Pass
WBJV009D1	23.99	25.07	1.08	15	0.90	0.47	0.02	0.03	1.41	MRMC	Pass
WBJV009D3	26.70	27.70	1.00	15	0.14	0.01	0.01	0.01	0.18	MRMC	Pass
WBJV010D1	51.42	52.43	1.01	15	1.37	0.57	0.22	0.01	2.18	MRMC	Pass
WBJV012D0	64.16	65.22	1.06	15	0.32	0.12	0.04	0.01	0.50	MRMC	Pass
WBJV014D1	37.82	38.82	1.00	15	0.29	0.14	0.04	0.01	0.48	MRMC	Pass
WBJV015D0	389.67	390.73	1.06	15	6.29	2.29	0.28	0.37	9.23	MRMC	Pass
WBJV015D1	31.76	33.22	1.46	15	2.94	1.24	0.14	0.19	4.51	MRMC	Pass
WBJV016D0	117.60	118.73	1.13	15	0.64	0.36	0.03	0.12	1.15	MRMC	Pass
WBJV016D1	27.12	28.20	1.08	15	0.30	0.14	0.02	0.09	0.55	MRMC	Pass
WBJV017D0	77.15	78.15	1.00	15	0.04	0.02	0.01	0.01	0.08	MRMC	Pass
WBJV017D1	16.65	17.65	1.00	15	0.04	0.02	0.01	0.01	0.08	MRMC	Pass
WBJV018D1	30.95	32.12	1.17	15	5.87	2.61	0.21	0.58	9.27	MRMC	Pass
WBJV022D0	81.16	82.16	1.00	15	0.17	0.08	0.03	0.01	0.28	MRMC	Pass
WBJV022D1	22.08	23.21	1.13	15	0.05	0.06	0.02	0.01	0.13	MRMC	Pass
WBJV022D2	11.50	12.54	1.04	15	0.06	0.03	0.01	0.01	0.12	MRMC	Pass
WBJV025D0	113.63	114.90	1.27	15	0.28	0.19	0.02	0.03	0.52	MRMC	Pass
WBJV025D1	33.36	34.46	1.10	15	0.35	0.24	0.01	0.04	0.65	MRMC	Pass
WBJV026D0	61.36	62.56	1.20	15	0.18	0.24	0.03	0.14	0.59	MRMC	Pass
WBJV026D1	11.51	12.51	1.00	15	0.98	0.24	0.03	0.11	1.36	MRMC	Pass
WBJV029D1	56.01	57.58	1.57	15	3.71	2.22	0.26	0.40	6.58	MRMC	Pass
WBJV030D0	475.89	477.12	1.23	15	4.98	2.06	0.27	0.43	7.74	MRMC	Pass
WBJV030D1	21.03	22.21	1.18	15	3.21	1.57	0.15	0.34	5.26	MRMC	Pass
WBJV030D2	27.77	28.81	1.04	15	0.09	0.04	0.01	0.09	0.23	MRMC	Pass
WBJV033D0	338.61	339.80	1.19	15	1.98	0.99	0.10	0.29	3.36	MRMC	Pass
WBJV033D1	19.42	20.41	0.99	15	3.02	1.05	0.17	0.22	4.46	MRMC	Pass
WBJV033D2	24.31	25.67	1.36	15	1.38	0.62	0.08	0.11	2.20	MRMC	Pass
WBJV039D0	124.06	125.06	1.00	15	0.33	0.13	0.05	0.01	0.52	MRMC	Pass
WBJV040D0	384.84	385.84	1.00	15	0.02	0.02	0.01	0.03	0.08	MRMC	Pass
WBJV040D1	14.74	15.97	1.23	15	1.55	0.77	0.07	0.27	2.65	MRMC	Pass
WBJV042D0	503.38	504.45	1.07	15	7.78	2.96	0.37	0.74	11.85	MRMC	Pass
WBJV042D1	7.74	8.89	1.15	15	3.54	1.75	0.21	0.45	5.95	MRMC	Pass
WBJV042D2	14.80	15.84	1.04	15	4.80	2.34	0.27	0.45	7.86	MRMC	Pass
WBJV043D0	529.37	530.76	1.39	15	4.78	1.78	0.23	0.36	7.14	MRMC	Pass
WBJV043D1	14.75	15.74	0.99	15	4.75	1.25	0.13	0.22	6.35	MRMC	Pass
WBJV043D2	9.65	10.75	1.10	15	4.19	1.56	0.24	0.30	6.28	MRMC	Pass
WBJV045D1	62.00	63.19	1.19	15	0.01	0.01	0.01	0.01	0.04	MRMC	Pass
WBJV048D0	423.17	424.37	1.20	15	0.62	0.59	0.06	0.10	1.37	MRMC	Pass
WBJV048D1	44.36	45.59	1.23	15	5.20	1.88	0.22	0.31	7.62	MRMC	Pass
WBJV050D0	530.63	531.75	1.12	15	4.42	2.01	0.24	0.30	6.97	MRMC	Pass
WBJV050D1	35.51	36.93	1.42	15	4.84	2.28	0.27	0.33	7.71	MRMC	Pass
WBJV053D0	220.50	222.54	2.04	15	7.48	2.39	0.47	0.39	10.73	MRMC	Pass
WBJV054D0	312.60	313.60	1.00	15	0.01	0.01	0.01	0.02	0.05	MRMC	Pass
WBJV054D2	27.64	28.67	1.03	15	0.01	0.01	0.01	0.01	0.04	MRMC	Pass
WBJV056D1	36.30	37.35	1.05	15	0.80	0.59	0.08	0.25	1.72	MRMC	Pass
WBJV057D0	145.72	146.77	1.05	15	2.90	1.09	0.16	0.13	4.28	MRMC	Pass
WBJV057D1	55.36	56.43	1.07	15	1.36	0.48	0.09	0.05	1.97	MRMC	Pass
WBJV058D0	384.49	385.67	1.18	15	4.67	1.63	0.30	0.33	6.92	MRMC	Pass
WBJV058D1	3.68	4.80	1.12	15	7.22	1.45	0.30	0.29	9.26	MRMC	Pass
WBJV059D0	184.20	185.20	1.00	15	0.91	0.24	0.13	0.01	1.30	MRMC	Pass
WBJV059D1	34.15	35.34	1.19	15	0.51	0.24	0.09	0.01	0.85	MRMC	Pass
WBJV063D0	139.64	140.88	1.24	15	0.05	0.02	0.01	0.01	0.09	MRMC	Pass

WBJV063D1	19.87	20.87	1.00	15	0.04	0.02	0.01	0.01	0.08	MRMC	Pass
WBJV064D0	228.76	229.86	1.10	15	0.04	0.02	0.01	0.01	0.08	MRMC	Pass
WBJV064D1	18.25	19.26	1.01	15	0.12	0.05	0.02	0.01	0.21	MRMC	Pass
WBJV065D1	8.26	9.61	1.35	15	0.05	0.02	0.01	0.01	0.09	MRMC	Pass
WBJV066D0	107.96	109.09	1.13	15	0.03	0.02	0.01	0.01	0.08	MRMC	Pass
WBJV066D1	27.67	28.48	0.81	15	0.01	0.01	0.01	0.01	0.04	MRMC	Pass
WBJV069D0	199.50	200.93	1.43	15	0.02	0.01	0.01	0.01	0.05	MRMC	Pass
WBJV072D0	172.38	173.40	1.02	15	0.02	0.01	0.01	0.01	0.05	MRMC	Pass
WBJV073D0	146.37	147.58	1.21	15	5.13	2.10	0.32	0.38	7.93	MRMC	Pass
WBJV075D0	87.00	88.10	1.10	15	0.07	0.03	0.01	0.01	0.12	MRMC	Pass
WBJV076D0	105.15	106.18	1.03	15	0.11	0.06	0.01	0.10	0.29	MRMC	Pass
WBJV077D0	219.70	220.84	1.14	15	0.18	0.09	0.02	0.02	0.30	MRMC	Pass
WBJV083D0	143.09	144.13	1.04	15	0.29	0.17	0.04	0.02	0.52	MRMC	Pass
WBJV083D1	12.71	13.86	1.15	15	0.11	0.05	0.01	0.01	0.18	MRMC	Pass
WBJV083D1	51.31	53.14	1.83	15	0.42	0.37	0.13	0.01	0.93	MRMC	Pass
WBJV083D2	18.07	19.11	1.04	15	1.14	0.48	0.17	0.02	1.81	MRMC	Pass
WBJV084D0	160.64	161.93	1.29	15	3.73	1.35	0.17	0.30	5.54	MRMC	Pass
WBJV085D0	467.14	468.16	1.02	15	3.35	1.15	0.15	0.20	4.85	MRMC	Pass
WBJV085D1	16.84	17.84	1.00	15	3.31	0.80	0.15	0.21	4.48	MRMC	Pass
WBJV087D0	192.49	193.59	1.10	15	3.41	1.57	0.21	0.46	5.65	MRMC	Pass
WBJV087D2	7.29	8.31	1.02	15	4.71	1.69	0.24	0.38	7.02	MRMC	Pass
WBJV087D3	12.37	13.50	1.13	15	2.72	0.69	0.13	0.15	3.70	MRMC	Pass
WBJV090D0	151.48	153.49	2.01	15	0.73	0.56	0.05	0.18	1.51	MRMC	Pass
WBJV090D1	12.37	13.51	1.14	15	0.33	0.22	0.02	0.05	0.62	MRMC	Pass
WBJV090D2	17.76	18.77	1.01	15	0.81	0.75	0.06	0.09	1.70	MRMC	Pass
WBJV091D0	346.44	347.55	1.11	15	4.06	0.82	0.30	0.15	5.34	MRMC	Pass
WBJV092D0	279.50	280.72	1.22	15	1.04	0.51	0.13	0.05	1.73	MRMC	Pass
WBJV092D1	19.07	20.14	1.07	15	0.46	0.21	0.04	0.05	0.75	MRMC	Pass
WBJV092D2	24.54	25.62	1.08	15	0.79	0.31	0.05	0.12	1.27	MRMC	Pass
WBJV093D0	399.96	401.17	1.21	15	1.42	0.72	0.07	0.28	2.48	MRMC	Pass
WBJV095D0	417.40	419.40	2.00	15	2.55	1.13	0.11	0.28	4.07	MRMC	Pass
WBJV095D1	13.03	15.28	2.25	15	3.16	1.51	0.15	0.40	5.21	MRMC	Pass
WBJV096D0	337.74	339.05	1.31	15	5.37	1.75	0.30	0.35	7.77	MRMC	Pass
WBJV096D1	60.85	63.35	2.50	15	15.57	7.38	1.05	0.67	24.67	MRMC	Pass
WBJV096D2	71.03	73.20	2.17	15	10.09	3.86	0.43	0.73	15.10	MRMC	Pass
WBJV100D2	26.12	27.32	1.20	15	3.26	1.44	0.20	0.26	5.16	MRMC	Pass
WBJV101D0	498.12	499.16	1.04	15	0.06	0.04	0.01	0.05	0.16	MRMC	Pass
WBJV102D0	408.86	410.20	1.34	15	2.34	1.00	0.11	0.21	3.66	MRMC	Pass
WBJV104D0	535.67	536.75	1.08	15	0.12	0.04	0.01	0.01	0.19	MRMC	Pass
WBJV104D1	60.46	61.64	1.18	15	1.01	0.55	0.07	0.07	1.70	MRMC	Pass
WBJV104D2	66.18	67.22	1.04	15	0.59	0.31	0.03	0.09	1.02	MRMC	Pass
WBJV106D0	398.08	399.07	0.99	15	4.76	1.96	0.32	0.33	7.36	MRMC	Pass
WBJV106D2	28.75	29.88	1.13	15	4.28	1.74	0.24	0.34	6.61	MRMC	Pass
WBJV108D1	42.24	43.25	1.01	15	11.85	4.07	0.52	1.17	17.61	MRMC	Pass
WBJV109D1	27.98	29.25	1.27	15	3.91	1.56	0.27	0.29	6.03	MRMC	Pass
WBJV109D2	31.29	34.67	3.38	15	2.13	0.80	0.13	0.23	3.29	MRMC	Pass
WBJV112D0	452.43	453.64	1.21	15	1.40	0.57	0.09	0.24	4.46	MRMC	Pass
WBJV112D1	19.88	23.08	3.20	15	3.84	1.41	0.29	0.06	5.59	MRMC	Pass
WBJV112D2	24.25	27.83	3.58	15	6.42	2.91	0.45	0.28	10.06	MRMC	Pass
WBJV113D0	411.96	412.99	1.03	15	0.04	0.03	0.01	0.02	0.11	MRMC	Pass
WBJV113D1	10.40	11.48	1.08	15	0.17	0.11	0.02	0.05	0.35	MRMC	Pass
WBJV113D2	18.19	19.44	1.25	15	0.07	0.06	0.01	0.02	0.15	MRMC	Pass
WBJV114D0	355.90	357.07	1.17	15	4.17	2.09	0.22	0.61	7.09	MRMC	Pass

WBJV116D0	506.30	507.52	1.22	15	2.02	0.88	0.10	0.19	3.18	MRMC	Pass
WBJV116D1	16.19	17.41	1.22	15	2.67	1.18	0.13	0.23	4.21	MRMC	Pass
WBJV117D0	345.78	347.03	1.25	15	0.02	0.01	0.01	0.01	0.05	MRMC	Pass
WBJV120D0	330.66	331.67	1.01	15	0.22	0.11	0.02	0.06	0.41	MRMC	Pass
WBJV125D0	457.71	458.88	1.17	15	4.95	1.59	0.20	0.24	6.97	MRMC	Pass
WBJV127D0	446.28	447.44	1.16	15	2.32	0.83	0.10	0.20	3.46	MRMC	Pass

Footnote: Additional “Pass” intersections from the adjoining mine were used in the resource calculations as they have a sphere of influence relative to the property. Although deemed to be reliable, these are not publicly available.

Table 1B: UG2 Reef Mineralised Intersections.

BHID	FROM(m)	TO(m)	LENGTH	CBA	PT(g/t)	PD(g/t)	RH(g/t)	AU(g/t)	4E(g/t)	REEF	VALID
WBJV001D0	473.20	475.60	2.40	15	0.50	0.17	0.11	0.00	0.78	UG2MC	Pass
WBJV001D1	25.29	27.82	2.53	15	0.30	0.11	0.08	0.00	0.50	UG2MC	Pass
WBJV001D2	53.52	55.45	1.93	15	0.51	0.18	0.11	0.00	0.80	UG2MC	Pass
WBJV002D0	555.92	557.62	1.70	15	2.03	0.73	0.29	0.01	3.06	UG2MC	Pass
WBJV002D1	105.11	106.11	1.00	15	2.12	0.75	0.30	0.01	3.17	UG2MC	Pass
WBJV002D2	16.67	17.86	1.19	15	2.22	0.73	0.31	0.01	3.27	UG2MC	Pass
WBJV003D0	536.61	537.68	1.07	15	2.56	0.86	0.35	0.02	3.80	UG2MC	Pass
WBJV003D1	83.10	84.58	1.48	15	1.99	1.17	0.28	0.03	3.48	UG2MC	Pass
WBJV003D2	186.26	187.32	1.06	15	0.38	0.12	0.09	0.00	0.60	UG2MC	Pass
WBJV005D0	483.89	485.66	1.77	15	0.51	0.19	0.11	0.00	0.81	UG2MC	Pass
WBJV007D0	255.66	256.78	1.12	15	2.25	0.65	0.22	0.03	3.14	UG2MC	Pass
WBJV008D0	324.14	325.32	1.18	15	0.60	0.25	0.12	0.01	0.97	UG2MC	Pass
WBJV008D1	102.36	103.51	1.15	15	1.55	0.62	0.19	0.01	2.38	UG2MC	Pass
WBJV009D0	279.72	281.14	1.42	15	0.38	0.09	0.08	0.01	0.57	UG2MC	Pass
WBJV009D3	46.14	47.50	1.36	15	0.62	0.18	0.13	0.01	0.94	UG2MC	Pass
WBJV010D1	84.50	86.46	1.96	15	0.50	0.24	0.11	0.02	0.87	UG2MC	Pass
WBJV012D0	69.85	70.97	1.12	15	0.12	0.05	0.02	0.01	0.20	UG2MC	Pass
WBJV013D0	471.99	475.20	3.21	15	0.43	0.16	0.11	0.01	0.70	UG2MC	Pass
WBJV013D1	124.12	125.22	1.10	15	0.26	0.07	0.06	0.01	0.41	UG2MC	Pass
WBJV014D0	247.35	248.36	1.01	15	0.32	0.10	0.08	0.01	0.51	UG2MC	Pass
WBJV014D1	47.17	48.17	1.00	15	0.17	0.05	0.03	0.01	0.26	UG2MC	Pass
WBJV015D0	433.97	435.62	1.65	15	2.43	0.98	0.34	0.03	3.79	UG2MC	Pass
WBJV015D1	77.13	78.31	1.18	15	2.98	0.98	0.36	0.02	4.34	UG2MC	Pass
WBJV016D0	133.01	134.18	1.17	15	2.87	1.05	0.36	0.03	4.32	UG2MC	Pass
WBJV016D1	41.94	43.30	1.36	15	2.19	0.57	0.29	0.02	3.07	UG2MC	Pass
WBJV018D0	243.35	244.96	1.61	15	2.77	1.36	0.39	0.03	4.55	UG2MC	Pass
WBJV018D1	45.20	46.42	1.22	15	1.71	0.71	0.27	0.02	2.71	UG2MC	Pass
WBJV020D0	96.34	97.53	1.19	15	0.55	0.04	0.10	0.01	0.71	UG2MC	Pass
WBJV020D1	26.50	27.63	1.13	15	1.30	0.12	0.29	0.01	1.72	UG2MC	Pass
WBJV021D0	280.54	281.65	1.11	15	4.02	1.75	0.43	0.05	6.25	UG2MC	Pass
WBJV021D1	89.85	90.85	1.00	15	2.15	0.74	0.26	0.03	3.18	UG2MC	Pass
WBJV022D0	99.13	100.95	1.82	15	0.14	0.06	0.04	0.01	0.25	UG2MC	Pass
WBJV022D1	38.26	39.42	1.16	15	0.33	0.11	0.08	0.01	0.52	UG2MC	Pass
WBJV022D2	28.59	29.59	1.00	15	0.16	0.18	0.04	0.01	0.39	UG2MC	Pass
WBJV023D0	201.75	204.50	2.75	15	1.86	0.61	0.28	0.02	2.77	UG2MC	Pass
WBJV024D0	282.96	283.96	1.00	15	0.75	0.42	0.09	0.02	1.28	UG2MC	Pass
WBJV024D1	63.00	64.00	1.00	15	0.94	0.54	0.10	0.03	1.60	UG2MC	Pass

WBJV025D0	121.48	123.17	1.69	15	2.76	0.85	0.33	0.02	3.96	UG2MC	Pass
WBJV025D1	40.20	42.68	2.48	15	3.40	2.62	0.37	0.08	6.47	UG2MC	Pass
WBJV026D0	70.03	71.03	1.00	15	0.28	0.21	0.03	0.02	0.53	UG2MC	Pass
WBJV026D1	19.70	20.70	1.00	15	0.32	0.12	0.03	0.01	0.49	UG2MC	Pass
WBJV027D1	58.03	59.03	1.00	15	0.22	0.08	0.05	0.01	0.37	UG2MC	Pass
WBJV027D2	99.33	100.35	1.02	15	0.29	0.11	0.05	0.01	0.46	UG2MC	Pass
WBJV028D0	221.91	224.65	2.74	15	3.10	1.58	0.37	0.05	5.10	UG2MC	Pass
WBJV028D1	71.64	74.21	2.57	15	4.37	2.46	0.38	0.07	7.28	UG2MC	Pass
WBJV030D0	516.56	517.65	1.09	15	0.60	0.21	0.03	0.02	0.86	UG2MC	Pass
WBJV032D0	360.95	362.11	1.16	15	2.55	1.65	0.30	0.37	4.87	UG2MC	Pass
WBJV032D1	113.25	114.45	1.20	15	3.57	1.26	0.42	0.02	5.27	UG2MC	Pass
WBJV033D1	55.50	56.54	1.04	15	0.81	0.26	0.04	0.01	1.13	UG2MC	Pass
WBJV033D2	59.43	60.42	0.99	15	0.99	0.61	0.12	0.01	1.73	UG2MC	Pass
WBJV034D0	478.44	479.60	1.16	15	0.17	0.10	0.04	0.01	0.32	UG2MC	Pass
WBJV034D1	48.34	49.34	1.00	15	0.15	0.04	0.04	0.01	0.25	UG2MC	Pass
WBJV035D0	517.04	519.11	2.07	15	1.01	0.22	0.14	0.01	1.38	UG2MC	Pass
WBJV035D1	46.38	50.71	4.33	15	0.57	0.21	0.14	0.01	0.93	UG2MC	Pass
WBJV037D0	46.06	47.06	1.00	15	2.87	1.15	0.35	0.05	4.43	UG2MC	Pass
WBJV038D2	67.00	69.10	2.10	15	0.18	0.07	0.02	0.01	0.28	UG2MC	Pass

WBJV039D0	136.99	137.99	1.00	15	0.48	0.14	0.06	0.01	0.69	UG2MC	Pass
WBJV040D0	433.12	434.14	1.02	15	0.04	0.02	0.01	0.01	0.07	UG2MC	Pass
WBJV040D1	61.75	62.75	1.00	15	0.20	0.10	0.05	0.01	0.36	UG2MC	Pass
WBJV041D0	537.70	539.14	1.44	15	0.02	0.02	0.01	0.01	0.06	UG2MC	Pass
WBJV041D1	60.37	61.58	1.21	15	0.06	0.01	0.01	0.01	0.10	UG2MC	Pass
WBJV042D0	524.54	525.54	1.00	15	2.36	0.79	0.30	0.01	3.47	UG2MC	Pass
WBJV042D1	29.15	30.14	0.99	15	2.48	1.55	0.30	0.09	4.42	UG2MC	Pass
WBJV043D0	574.50	575.50	1.00	15	0.65	0.28	0.07	0.01	1.02	UG2MC	Pass
WBJV043D1	63.93	65.18	1.25	15	0.48	0.43	0.11	0.01	1.03	UG2MC	Pass
WBJV044D0	500.47	503.45	2.98	15	0.52	0.20	0.13	0.01	0.87	UG2MC	Pass
WBJV044D1	30.16	33.25	3.09	15	0.46	0.31	0.08	0.01	0.86	UG2MC	Pass
WBJV045D0	573.68	575.41	1.73	15	3.16	1.46	0.48	0.01	5.11	UG2MC	Pass
WBJV045D1	73.67	74.94	1.27	15	2.92	1.13	0.41	0.01	4.47	UG2MC	Pass
WBJV046D0	544.48	545.78	1.30	15	2.73	1.06	0.32	0.02	4.12	UG2MC	Pass
WBJV046D1	64.41	65.75	1.34	15	3.03	1.39	0.44	0.06	4.92	UG2MC	Pass
WBJV047D0	47.52	48.52	1.00	15	0.41	0.26	0.06	0.01	0.74	UG2MC	Pass
WBJV048D0	478.21	479.94	1.73	15	2.07	0.40	0.36	0.01	2.84	UG2MC	Pass
WBJV048D1	97.30	98.40	1.10	15	3.02	1.41	0.41	0.04	4.88	UG2MC	Pass
WBJV049D0	550.64	551.85	1.21	15	0.24	0.07	0.06	0.01	0.37	UG2MC	Pass
WBJV050D0	591.47	592.67	1.20	15	3.33	1.42	0.56	0.04	5.34	UG2MC	Pass
WBJV050D1	96.56	97.67	1.11	15	3.17	1.58	0.51	0.05	5.31	UG2MC	Pass
WBJV052D0	190.63	191.72	1.09	15	0.60	0.19	0.05	0.01	0.86	UG2MC	Pass
WBJV053D0	249.17	250.26	1.09	15	0.31	0.02	0.05	0.01	0.40	UG2MC	Pass
WBJV053D1	45.90	46.90	1.00	15	0.25	0.10	0.07	0.01	0.43	UG2MC	Pass
WBJV053D2	53.49	54.71	1.22	15	0.36	0.30	0.10	0.02	0.77	UG2MC	Pass
WBJV054D0	337.40	338.47	1.07	15	1.00	0.34	0.13	0.02	1.49	UG2MC	Pass
WBJV054D1	17.38	18.50	1.12	15	1.44	0.64	0.22	0.02	2.32	UG2MC	Pass
WBJV055D0	218.88	220.26	1.38	15	0.45	0.07	0.11	0.01	0.64	UG2MC	Pass
WBJV055D1	28.85	30.00	1.15	15	0.50	0.05	0.13	0.01	0.69	UG2MC	Pass
WBJV056D0	286.33	287.87	1.54	15	3.29	2.37	0.40	0.03	6.09	UG2MC	Pass
WBJV056D1	92.59	93.82	1.23	15	1.84	0.56	0.30	0.01	2.72	UG2MC	Pass
WBJV057D0	162.18	163.28	1.10	15	0.80	0.39	0.17	0.01	1.38	UG2MC	Pass
WBJV057D1	71.70	72.80	1.10	15	0.88	0.29	0.14	0.01	1.32	UG2MC	Pass

WBJV058D0	418.03	419.11	1.08	15	0.23	0.10	0.07	0.01	0.41	UG2MC	Pass
WBJV058D1	37.95	39.12	1.17	15	0.36	0.13	0.11	0.01	0.62	UG2MC	Pass
WBJV059D0	200.62	203.22	2.60	15	0.46	0.13	0.14	0.01	0.74	UG2MC	Pass
WBJV060D0	248.46	249.78	1.32	15	2.29	0.81	0.37	0.02	3.49	UG2MC	Pass
WBJV060D1	49.37	51.72	2.35	15	3.23	1.05	0.45	0.03	4.75	UG2MC	Pass
WBJV061D0	137.35	138.61	1.26	15	0.39	0.05	0.10	0.01	0.56	UG2MC	Pass
WBJV061D1	78.78	80.14	1.36	15	0.60	0.06	0.15	0.01	0.83	UG2MC	Pass
WBJV064D0	242.53	245.02	2.49	15	0.56	0.11	0.14	0.01	0.82	UG2MC	Pass
WBJV064D1	33.84	35.79	1.95	15	0.53	0.13	0.14	0.01	0.81	UG2MC	Pass
WBJV065D0	315.77	316.91	1.14	15	0.14	0.09	0.04	0.01	0.28	UG2MC	Pass
WBJV065D1	31.19	32.37	1.18	15	0.07	0.03	0.02	0.01	0.13	UG2MC	Pass
WBJV067D0	375.25	378.34	3.09	15	3.47	1.36	0.50	0.02	5.35	UG2MC	Pass
WBJV068D0	267.45	268.51	1.06	15	1.98	0.92	0.36	0.01	3.28	UG2MC	Pass
WBJV068D1	26.38	27.74	1.36	15	2.74	1.00	0.45	0.01	4.19	UG2MC	Pass
WBJV070D1	66.41	68.12	1.71	15	0.34	0.15	0.08	0.01	0.58	UG2MC	Pass
WBJV070RD0	52.72	53.73	1.01	15	0.24	0.04	0.07	0.01	0.36	UG2MC	Pass
WBJV071D0	56.50	58.64	2.14	15	0.41	0.13	0.10	0.01	0.65	UG2MC	Pass
WBJV071D1	27.81	28.93	1.12	15	0.37	0.30	0.12	0.01	0.81	UG2MC	Pass
WBJV072D1	54.65	58.50	3.85	15	0.36	0.12	0.08	0.01	0.57	UG2MC	Pass
WBJV073D0	159.02	160.17	1.15	15	2.98	1.32	0.53	0.03	4.86	UG2MC	Pass
WBJV074D0	530.44	531.50	1.06	15	0.13	0.03	0.03	0.01	0.20	UG2MC	Pass
WBJV078D0	72.25	73.66	1.41	15	0.37	0.26	0.08	0.01	0.72	UG2MC	Pass
WBJV083D0	178.15	182.37	4.22	15	0.51	0.43	0.15	0.01	1.10	UG2MC	Pass
WBJV083D2	53.15	54.49	1.34	15	0.83	0.16	0.25	0.01	1.24	UG2MC	Pass
WBJV084D1	69.97	71.09	1.12	15	2.92	0.98	0.38	0.03	4.31	UG2MC	Pass
WBJV085D0	508.65	510.16	1.51	15	2.75	0.86	0.42	0.01	4.04	UG2MC	Pass
WBJV085D1	57.62	59.16	1.54	15	3.15	1.20	0.46	0.01	4.82	UG2MC	Pass
WBJV086D0	202.94	204.35	1.41	15	0.48	0.24	0.14	0.01	0.87	UG2MC	Pass
WBJV086D1	30.70	33.43	2.73	15	0.46	0.23	0.14	0.01	0.84	UG2MC	Pass
WBJV086D2	37.33	38.58	1.25	15	0.04	0.02	0.01	0.01	0.07	UG2MC	Pass
WBJV088D0	184.81	185.83	1.02	15	0.18	0.07	0.04	0.01	0.31	UG2MC	Pass
WBJV089D1	121.72	122.72	1.00	15	1.99	0.22	0.09	0.01	2.31	UG2MC	Pass
WBJV093D0	439.26	440.43	1.17	15	1.33	0.21	0.15	0.01	1.71	UG2MC	Pass
WBJV096D0	417.84	418.83	0.99	15	0.54	0.21	0.08	0.01	0.84	UG2MC	Pass
WBJV099D0	453.90	455.04	1.14	15	1.36	1.27	0.26	0.04	2.92	UG2MC	Pass
WBJV099D1	71.05	72.19	1.14	15	0.38	0.04	0.09	0.01	0.52	UG2MC	Pass
WBJV099D2	68.20	69.29	1.09	15	1.04	0.59	0.19	0.01	1.83	UG2MC	Pass
WBJV100D0	408.59	409.70	1.11	15	2.59	1.05	0.40	0.01	4.05	UG2MC	Pass
WBJV100D2	111.39	114.03	2.64	15	2.40	0.93	0.35	0.03	3.71	UG2MC	Pass
WBJV102D0	467.13	468.23	1.10	15	0.95	0.60	0.14	0.02	1.71	UG2MC	Pass
WBJV102D2	122.34	123.49	1.15	15	0.66	0.22	0.11	0.02	1.01	UG2MC	Pass
WBJV103D0	446.88	448.47	1.59	15	3.30	1.11	0.50	0.03	4.93	UG2MC	Pass
WBJV104D0	564.46	567.02	2.56	15	1.16	0.45	0.19	0.01	1.82	UG2MC	Pass
WBJV104D1	89.00	91.04	2.04	15	2.06	0.86	0.38	0.02	3.32	UG2MC	Pass
WBJV104D2	94.93	97.36	2.43	15	1.51	0.64	0.24	0.02	2.40	UG2MC	Pass
WBJV105D0	450.36	451.44	1.08	15	1.37	0.44	0.20	0.01	2.01	UG2MC	Pass
WBJV105D2	96.31	97.31	1.00	15	0.32	0.06	0.08	0.02	0.48	UG2MC	Pass
WBJV108D0	421.76	423.27	1.51	15	1.87	0.51	0.31	0.02	2.71	UG2MC	Pass
WBJV108D2	96.84	98.00	1.16	15	2.97	1.18	0.35	0.03	4.53	UG2MC	Pass
WBJV109D1	92.65	94.70	2.05	15	2.41	0.63	0.32	0.02	3.38	UG2MC	Pass
WBJV109D2	98.18	99.61	1.43	15	3.36	1.52	0.44	0.03	5.35	UG2MC	Pass
WBJV112D0	502.14	503.21	1.07	15	2.59	0.83	0.40	0.02	3.84	UG2MC	Pass

WBJV112D2	77.07	78.27	1.20	15	1.72	0.82	0.32	0.01	2.87	UG2MC	Pass
WBJV113D0	429.19	430.46	1.27	15	1.32	0.29	0.17	0.01	1.80	UG2MC	Pass
WBJV113D1	26.10	28.00	1.90	15	1.28	0.28	0.16	0.01	1.73	UG2MC	Pass
WBJV113D2	33.46	34.65	1.19	15	0.79	0.24	0.06	0.01	1.10	UG2MC	Pass
WBJV116D0	562.88	564.05	1.17	15	2.41	1.52	0.35	0.03	4.31	UG2MC	Pass
WBJV116D1	72.37	73.43	1.06	15	3.11	1.23	0.50	0.02	4.86	UG2MC	Pass
WBJV117D0	369.19	370.39	1.20	15	1.85	0.57	0.26	0.01	2.70	UG2MC	Pass
WBJV118D0	478.33	479.61	1.28	15	1.20	0.59	0.22	0.01	2.02	UG2MC	Pass
WBJV118D1	49.53	50.73	1.20	15	1.54	0.33	0.21	0.01	2.09	UG2MC	Pass

Footnote: Additional “Pass” intersections from the adjoining mine were used in the resource calculations as they have a sphere of influence relative to the property. Although deemed to be reliable, these are not publicly available.

Table 10(a): Detailed flow-of-ore methodology – stoping

STOPING

Reef Type		Merensky										
Data	Units	6	7	8	9	10	11	12	13	14	15	1
Face Advance.	m	4.7	5.2	5.1	5.1	5.1	5.1	5.1	5.2	5.1	5.0	
Square Metres	m²	56,853	335,043	366,754	366,717	361,886	359,154	366,700	357,113	231,589	78,727	1
Stope Width	cm	127	125	124	125	124	125	130	130	123	122	
Tons Broken	t	225,343	1,310,571	1,428,990	1,435,158	1,400,831	1,404,464	1,488,670	1,450,903	888,885	301,521	4
Content.	g	1,335,250	8,533,548	10,188,267	10,379,217	9,505,264	9,185,965	9,507,418	9,073,656	5,442,624	1,904,159	26
Face Value.	g/t	5.93	6.51	7.13	7.23	6.79	6.54	6.39	6.25	6.12	6.32	
Gully Sundries	t	11,297	67,551	74,260	73,899	73,789	72,436	70,895	68,977	47,645	16,233	
Tons To Mill	t	236,640	1,378,123	1,503,249	1,509,057	1,474,619	1,476,900	1,559,565	1,519,879	936,530	317,754	4

Reef Type		UG2										
Data	Units	6	7	8	9	10	11	12	13	14	15	1
Face Advance.	m								5.0	5.1	5.0	
Square Metres	m²								3,860	80,962	196,985	26
Stope Width	cm								201	174	152	
Tons Broken	t								27,920	507,380	1,076,369	1,43
Content.	g								125,978	2,259,289	4,402,416	5,83
Face Value.	g/t								4.51	4.45	4.09	
Gully Sundries	t								304	10,759	35,032	4
Tons To Mill	t								28,224	518,139	1,111,401	1,48

Reef Type		Total										
Data	Units	6	7	8	9	10	11	12	13	14	15	1
Face Advance.	m	4.7	5.2	5.1	5.1	5.1	5.1	5.1	5.2	5.1	5.0	
Square Metres	m²	56,853	335,043	366,754	366,717	361,886	359,154	366,700	360,973	312,551	275,712	27
Stope Width	cm	127	125	124	125	124	125	130	131	136	143	
Tons Broken	t	225,343	1,310,571	1,428,990	1,435,158	1,400,831	1,404,464	1,488,670	1,478,822	1,396,265	1,377,890	1,48
Content.	g	1,335,250	8,533,548	10,188,267	10,379,217	9,505,264	9,185,965	9,507,418	9,199,634	7,701,913	6,306,575	6,11
Face Value.	g/t	5.93	6.51	7.13	7.23	6.79	6.54	6.39	6.22	5.52	4.58	
Gully Sundries	t	11,297	67,551	74,260	73,899	73,789	72,436	70,895	69,281	58,405	51,266	4
Tons To Mill	t	236,640	1,378,123	1,503,249	1,509,057	1,474,619	1,476,900	1,559,565	1,548,103	1,454,670	1,429,155	1,53

Table 10(b): Detailed flow-of-ore methodology – reef development

REEF DEVELOPMENT

Reef Type		Merensky									
		Years									
Data	Units	5	6	7	8	9	10	11	12	13	
Reef Metres	m	631	7,220	7,414	4,501	2,877	4,475	4,412	3,715	2,384	
Square Metres	m²	1,136	12,996	13,346	8,103	5,178	8,055	7,942	6,687	4,292	
Tons Broken	t	8,889	101,695	104,429	63,403	40,516	63,032	62,150	52,323	33,564	
Content	g	26,965	331,108	375,742	210,351	128,671	206,957	210,977	165,452	100,395	
Face Value	g/t	3.03	3.26	3.60	3.32	3.18	3.28	3.39	3.16	2.98	
Tons To Mill	t	8,889	101,695	104,429	63,403	40,516	63,032	62,150	52,323	33,564	

Reef Type		UG2									
		Years									
Data	Units	5	6	7	8	9	10	11	12	13	
Reef Metres	m									1,159	
Square Metres	m²									2,086	
Tons Broken	t									18,770	
Content	g									53,851	1
Face Value	g/t									2.87	
Tons To Mill	t									18,770	

Reef Type		Total									
		Years									
Data	Units	5	6	7	8	9	10	11	12	13	
Reef Metres	m	631	7,220	7,414	4,501	2,877	4,475	4,412	3,715	3,543	
Square Metres	m²	1,136	12,996	13,346	8,103	5,178	8,055	7,942	6,687	6,377	
Tons Broken	t	8,889	101,695	104,429	63,403	40,516	63,032	62,150	52,323	52,354	
Content	g	26,965	331,108	375,742	210,351	128,671	206,957	210,977	165,452	153,946	2
Face Value	g/t	3.03	3.26	3.60	3.32	3.18	3.28	3.39	3.16	2.94	
Tons To Mill	t	8,889	101,695	104,429	63,403	40,516	63,032	62,150	52,323	52,354	

Table 10(c): Detailed flow-of-ore methodology – plant

PLANT

Reef Type									Merensky				
			Years										
Data	%	Units	5	6	7	8	9	10	11	12	13	14	15
Tons To Mill		t	8,889	338,335	1,482,552	1,566,652	1,548,573	1,537,651	1,539,050	1,611,888	1,553,464	945,534	217,0
Shortfall Tons (Dilution)	5%	t	468	17,807	78,029	82,455	81,556	80,929	81,003	84,836	81,761	49,765	16,7
Tons Milled.		t	9,357	356,142	1,560,581	1,649,108	1,631,130	1,618,580	1,620,053	1,696,724	1,635,225	995,298	334,6
Total Received		kg	27	1,666	8,909	10,399	10,508	9,712	9,397	9,673	9,174	5,470	1,9
MCF	97.5%	kg	26	1,625	8,687	10,139	10,245	9,469	9,162	9,431	8,944	5,334	1,8
MCF		g/t	2.81	4.56	5.57	6.15	6.28	5.85	5.66	5.56	5.47	5.36	5.
Concentrator	87.5%	kg	23	1,422	7,601	8,671	8,965	8,286	8,017	8,252	7,826	4,667	1,6
Concentrator		g/t	2.46	3.99	4.87	5.36	5.50	5.12	4.95	4.86	4.79	4.69	4.
Total Oz's.		oz	740	45,706	244,369	285,219	288,216	266,392	257,745	265,313	251,623	150,042	52,2

Reef Type									UG2				
			Years										
Data	%	Units	5	6	7	8	9	10	11	12	13	14	15
Tons To Mill		t	0	0	0	0	0	0	0	0	46,993	595,810	1,213,2
Shortfall Tons (Dilution)	5%	t									2,473	31,358	63,8
Tons Milled.		t									49,467	627,168	1,277,0
Total Received		kg									180	2,439	4,6
MCF	97.5%	kg									175	2,379	4,5
MCF		g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	3.54	3.79	3.
Concentrator	82.5%	kg									145	1,962	3,7
Concentrator		g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	2.92	3.13	2.
Total Oz's.		oz	0	0	0	0	0	0	0	0	4,651	63,088	120,3

Reef Type	Total	Total											
			Years										
Data	%	Units	5	6	7	8	9	10	11	12	13	14	15
Tons To Mill		t	8,889	338,335	1,482,552	1,566,652	1,548,573	1,537,651	1,539,050	1,611,888	1,600,457	1,541,343	1,531,0
Shortfall Tons (Dilution)	5%	t	468	17,807	78,029	82,455	81,556	80,929	81,003	84,836	84,235	81,123	80,5
Tons Milled.		t	9,357	356,142	1,560,581	1,649,108	1,631,130	1,618,580	1,620,053	1,696,724	1,684,692	1,622,466	1,611,6
Total Received		kg	27	1,666	8,909	10,399	10,508	9,712	9,397	9,673	9,354	7,910	6,5
MCF	97.5%	kg	26	1,625	8,687	10,139	10,245	9,469	9,162	9,431	9,120	7,712	6,3
MCF		g/t	2.81	4.56	5.57	6.15	6.28	5.85	5.66	5.56	5.41	4.75	3.
Concentrator	86.2%	kg	23	1,422	7,601	8,671	8,965	8,286	8,017	8,252	7,971	6,629	5,3
Concentrator		g/t	2.46	3.99	4.87	5.36	5.50	5.12	4.95	4.86	4.73	4.09	3.
Total Oz's.		oz	740	45,706	244,369	285,219	288,216	266,392	257,745	265,313	256,273	213,130	172,5

Table 11: Waste development.

Western Bushveld Joint Venture Project 1 - UG2 & Merensky Vertical Shaf
Waste Development Metres

Advance		Name							
Project	Year	Access Drives	Raw	Haulage	Xcut	Incline	Box Hole	Decline	Trav
Merensky	5	13,008	237	5,266	2,502		535		
	6		721	9,895	5,659		1,625		
	7		401	8,945	4,989		1,410		
	8		265	6,545	2,999	159	998		
	9		265	6,800	2,817	106	924		
	10		265	6,674	3,696		965		
	11		265	4,890	3,293		849		
	12		251	2,586	2,651		713		
	13			458	797		220		
	14			9	119		34		
Merensky Total		**13,008**	**2,670**	**52,067**	**29,520**	**265**	**8,273**		
UG2	11			124					
	12			1,093	293		34		
	13			5,988	1,866		413	144	
	14			7,869	4,613		1,215		
	15			3,812	3,349		947		
	16			1,711	1,276		341		
	17			44	399		137		
UG2 Total				**20,642**	**11,796**		**3,088**	**144**	
Grand Total		**13,008**	**2,670**	**72,709**	**41,316**	**265**	**11,361**	**144**	

Table 12: Reconciliation.

WBJV - PROJECT 1

SQUARE METRES RECONCILIATION

Merensky | **U**

Project Area | **Proje**

Source	Tonnage	Mine Width	SG	Square Metres
CR Facies Inferred	6,005,927	1.00	3.13	1,918,827
CR Facies Indicated	6,754,547	1.01	3.13	2,127,997
FPP Facies Inferred	2,925,351	1.24	3.13	755,429
FPP Facies Indicated	18,000,000	1.26	3.13	4,565,213
FPP Facies Measured	2,542,426	1.24	3.13	655,115
Total Resource	36,228,251	1.15	3.13	10,022,581
Irups				739,815
Project Area	**36,228,251**	**1.15**	**3.13**	**10,762,396**

Source	Tonnage	Mi
Inferred	13,400,000	
Indicated	28,600,000	
Measured	2,575,224	
Total Resource	44,575,224	
Irups		
Project Area	**44,575,224**	

Reconciliation

Source		Square Metres	Total %
Irups		739,815	6.8%
FPP Facies Inferred		755,429	7.0%
< 2g/t (CR Facies)		4,046,824	37.3%
Isolated blocks		109,844	1.0%
Faults		860,992	7.9%
Dykes		430,496	4.0%
Raises Mined		76,556	0.7%
Panels Mined		2,892,032	26.6%
Panels Removed		32,134	0.3%
Pillars		913,273	8.4%
Total Project		10,857,395	100.0%
Difference	**-0.9%**	**-94,999**	

Reconciliation

Source		Sq
Irups		
Inferred		
< 3.5 g/t		
Isolated blocks		
Faults		
Dykes		
Raises Mined		
Panels Mined		
Panels Removed		
Pillars		
Total Project		
Difference	**3.5%**	**310,**

Table 14: Financial model – vertical shaft.

Production Schedule of: WBJV PROJECT 1																	
Mining Royalty (Base Metal) 1.00% UNDERGROUND MINE with SHAFT - 140 000 Option 12 November 2006 Profile - MR & UG2 - (Run 13E)																	
Mining Royalty (PGM) 3.00%																	
		Totals	2008/09	2009/10	2010/11	2011/12	20012/13	20013/14	2014/15	2015/16	2016/17	2017/18	2018/19	2019/20	2020/21	2
			1	2	3	4	5	6	7	8	9	10	11	12	13	
PGM Basket Price	R/kg	-	-	-	-	-	185 800	186 800	185 800	185 800	186 800	185 800	185 800	186 800	186 475	
New Project Profile																
Tons Milled	tons	20 622 588	-	-	-	-	-	360 000	1 665 000	1 648 000	1 630 000	1 620 000	1 620 000	1 680 000	1 680 000	1
Concentrator Recovered PGM's	kg	87 834.8	-	-	-	-	-	1 423	7 617	8 865	8 958	8 294	8 017	8 171	7 947	
Payable Smelter PGM's	kg	75 823.8	-	-	-	-	-	1 224	6 551	7 624	7 704	7 133	6 894	7 027	6 834	
Overall Recovery Grade	g/t	3.867	-	-	-	-	-	3.40	4.19	4.63	4.73	4.40	4.28	4.38	4.07	
Operating Costs	R/toe milled	R 351.82	-	-	-	-	-	633.97	390.68	362.34	361.27	364.82	366.11	346.01	337.01	
	On mine R/kg In Concentrate	R 78 972	-	-	-	-	-	155 443	74 711	53 899	62 277	57 747	68 286	67 670	57 772	
	R/kg payable	R 95 942	-	-	-	-	-	186 511	90 921	78 325	78 437	82 809	83 441	82 723	82 848	
Project Capital Expenditure	R'000	2 457 107	235 002	341 747	525 357	520 340	313 011	156 823	433 858	115 350	43 891	(12 779)	8 740	34 879	52 894	
Financial Evaluation (Stand-Alone Basis)																
Revenue (Total)	R'000	18 219 360	-	-	-	-	-	264 261	1 385 154	1 802 817	1 617 091	1 505 376	1 458 994	1 489 184	1 462 885	1
Mining Royalty (PGM)	R'000	451 887	-	-	-	-	-	7 137	38 206	44 463	44 930	41 601	40 209	40 983	39 964	
Mining Royalty (Base Metal)	R'000	11 568	-	-	-	-	-	264	1 147	1 207	1 194	1 187	1 187	1 231	1 207	
Working Costs (Total)	R'000	7 255 518	5 000	5 000	5 000	5 000	62 482	228 230	595 603	597 129	589 870	590 683	575 256	581 297	568 170	1
Penalties	R'000	148 584	-	-	-	-	-	6 828	30	35	36	33	32	33	692	
Working Profit	R'000	9 361 807	(5 000)	(5 000)	(5 000)	(5 000)	(62 482)	21 803	753 170	959 983	982 061	871 872	842 281	866 621	844 952	
Total Capital Expenditure	R'000	2 457 107	235 002	341 747	525 357	520 340	313 011	156 823	433 858	115 350	43 891	(12 779)	8 740	34 879	52 894	
Pre-Tax Net Cash Flow	R'000	6 904 699	(240 002)	(346 747)	(530 357)	(525 340)	(385 493)	(135 020)	319 311	844 632	938 170	884 651	833 541	830 743	792 258	
Pre-Tax Cumulative Cash Flow	R'000		*(240 002)*	*(586 749)*	*(1 117 107)*	*(1 642 446)*	*(2 007 939)*	*(2 142 959)*	*(1 823 648)*	*(979 015)*	*(40 845)*	*843 806*	*1 677 347*	*2 508 090*	*3 300 348*	*3*
After Tax Net Cash Flow	R'000	3 448 399	(240 002)	(346 747)	(530 357)	(525 340)	(385 493)	(135 020)	319 311	844 632	938 170	534 471	497 622	486 985	463 471	
After Tax Cumulative Cash Flow	R'000	*R 0*	*(240 002)*	*(586 749)*	*(1 117 107)*	*(1 642 446)*	*(2 007 939)*	*(2 142 959)*	*(1 823 648)*	*(979 015)*	*(40 845)*	*493 626*	*991 248*	*1 467 232*	*1 930 704*	*2*
Tax Calculation:																
Unred.Cap.Bal.B/f	R'000		-	*240 002*	*586 749*	*1 117 107*	*1 642 446*	*2 007 939*	*2 142 959*	*1 823 648*	*979 015*	*40 845*	-	-	-	
Total Capital for Tax	R'000	*13 037 817*	*235 002*	*581 749*	*1 112 107*	*1 637 446*	*1 965 457*	*2 164 762*	*2 576 817*	*1 938 998*	*1 022 906*	*28 066*	*8 740*	*34 879*	*52 694*	
Redemption	R'000	*2 457 107*	*(5 000)*	*(5 000)*	*(5 000)*	*(5 000)*	*(62 482)*	*21 803*	*753 170*	*959 983*	*982 061*	*28 066*	*8 740*	*34 879*	*52 694*	
Unred.Cap.Bal c/f	R'000		*240 002*	*586 749*	*1 117 107*	*1 642 446*	*2 007 939*	*2 142 959*	*1 823 648*	*979 015*	*40 845*	-	-	-	-	
Taxable Income	R'000	*5 904 699*	-	-	-	-	-	-	-	-	-	*843 806*	*833 541*	*830 743*	*792 258*	
Tax Rate			*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	*29.0%*	
STC	%		*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	*12.5%*	
Tax Payable	R'000	*2 446 300*	-	-	-	-	-	-	-	-	-	*350 179*	*345 920*	*344 758*	*328 787*	

APPENDIX B

QA&QC 3SD Plotted Graphs

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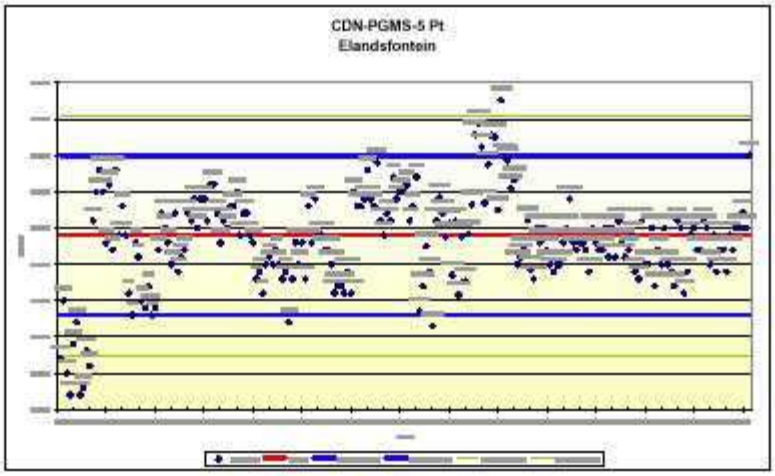

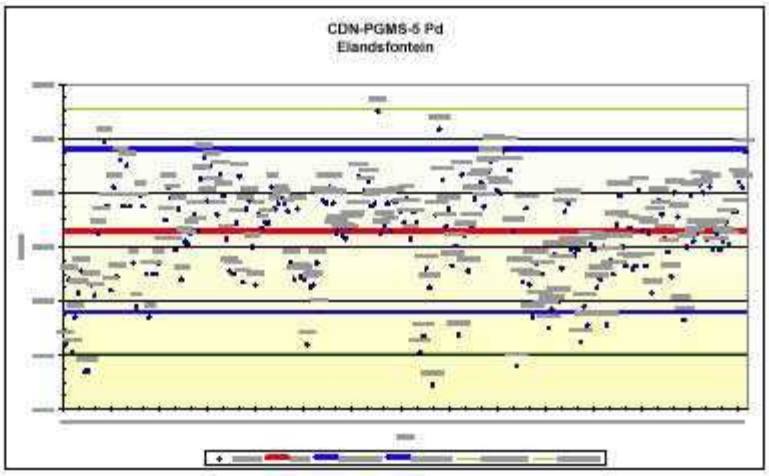

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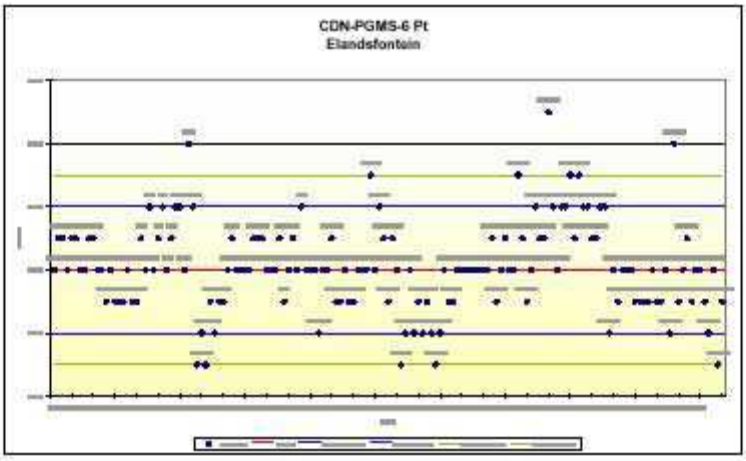
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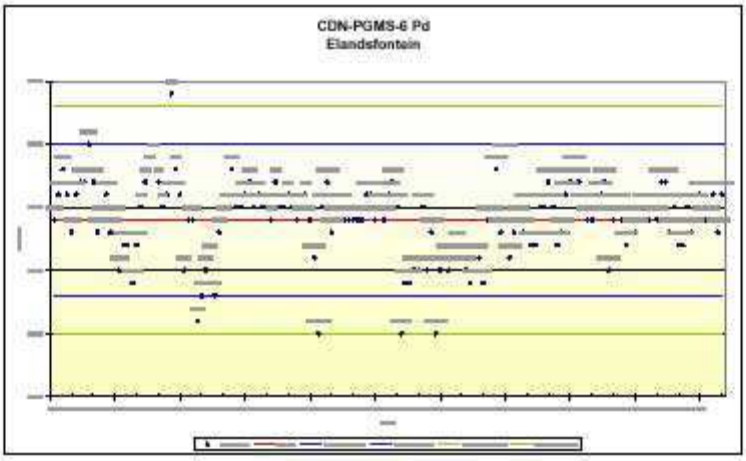
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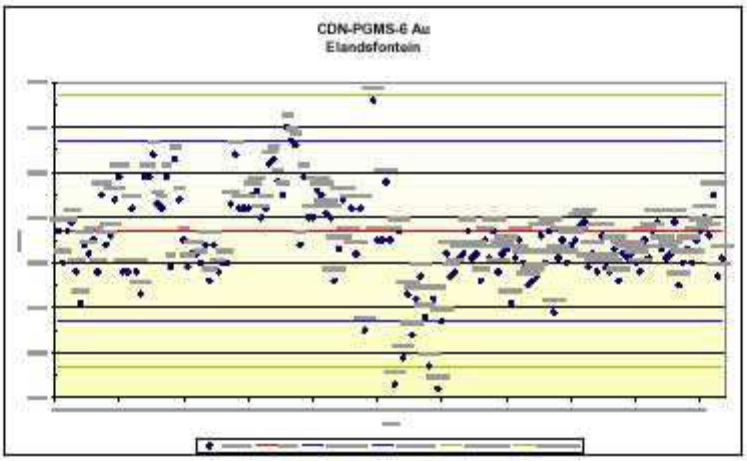
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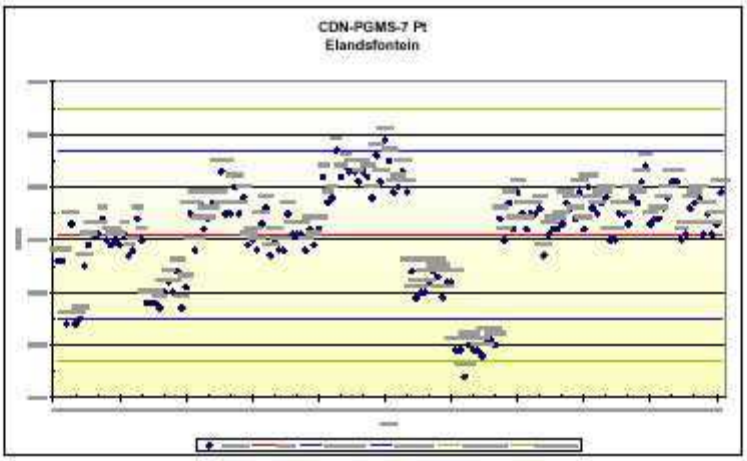
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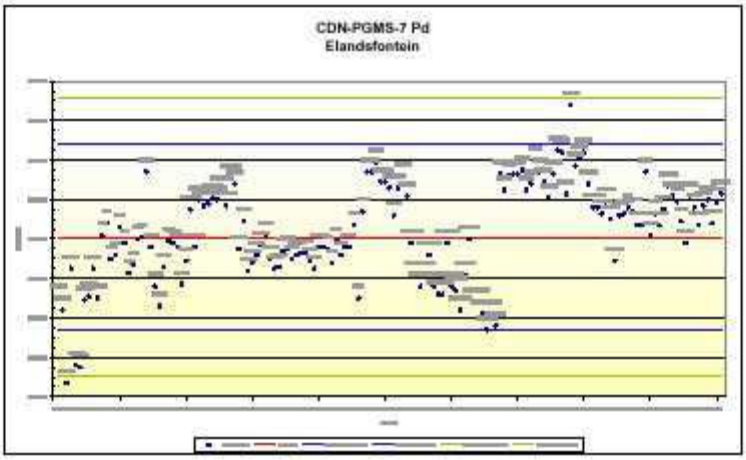
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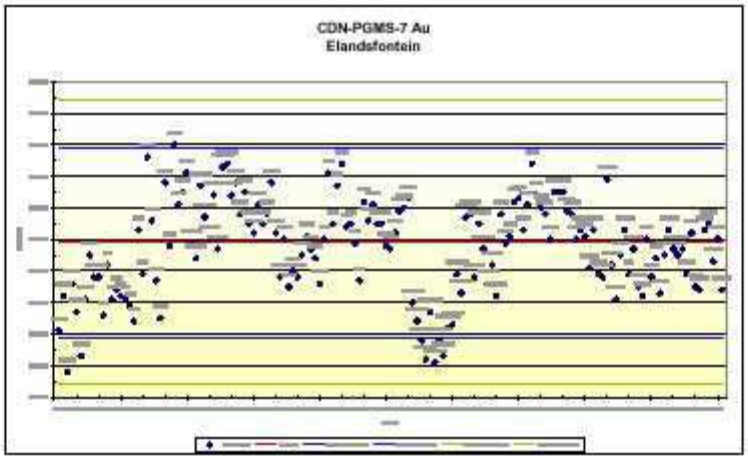
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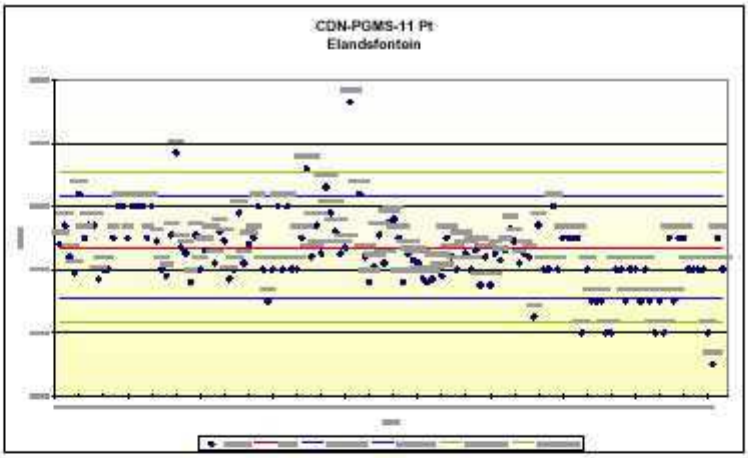
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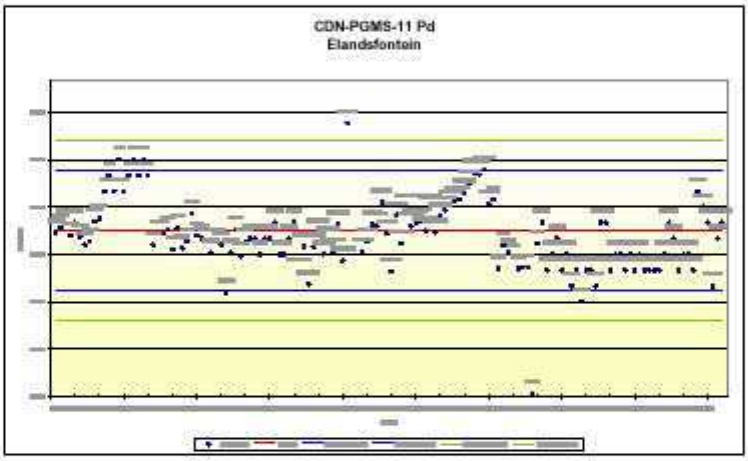
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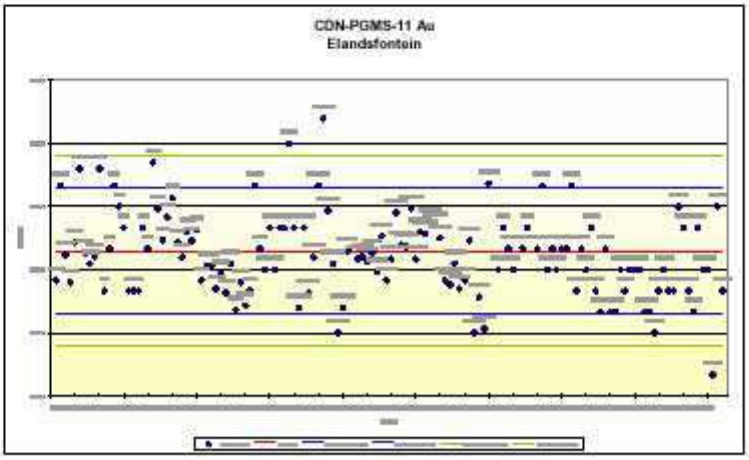
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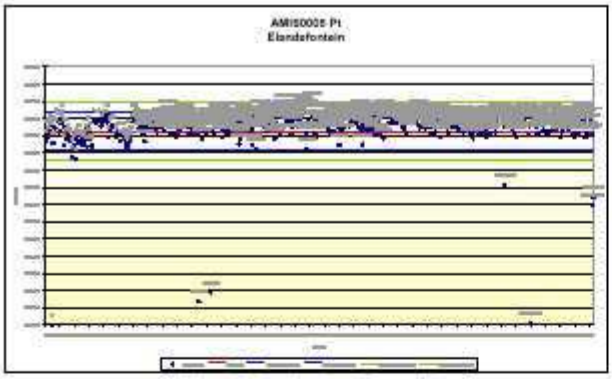
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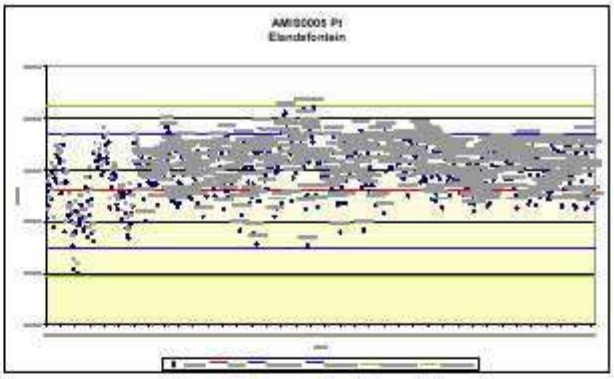
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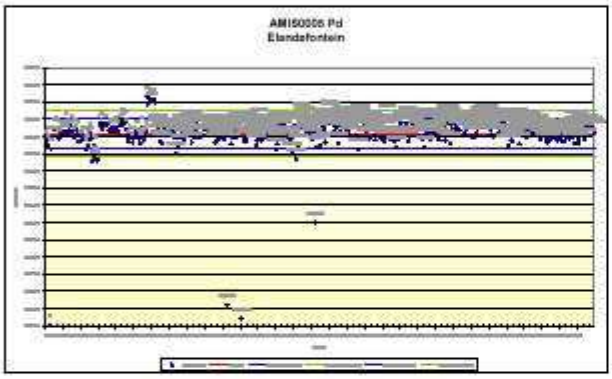
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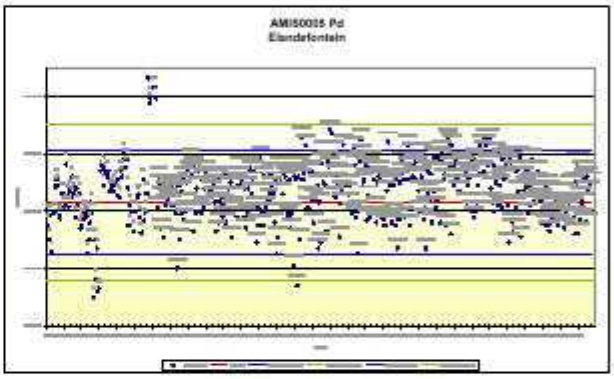
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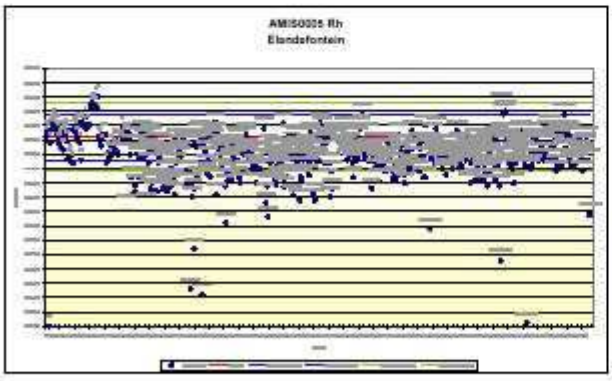
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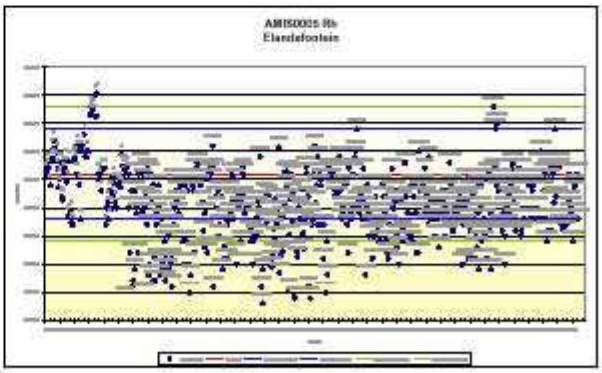
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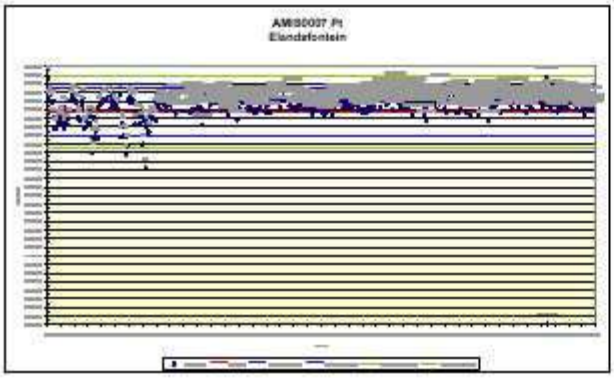
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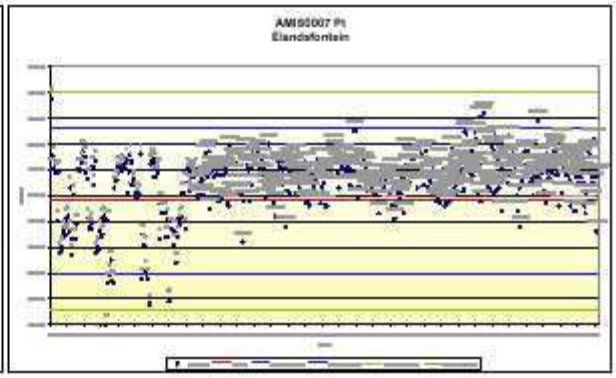
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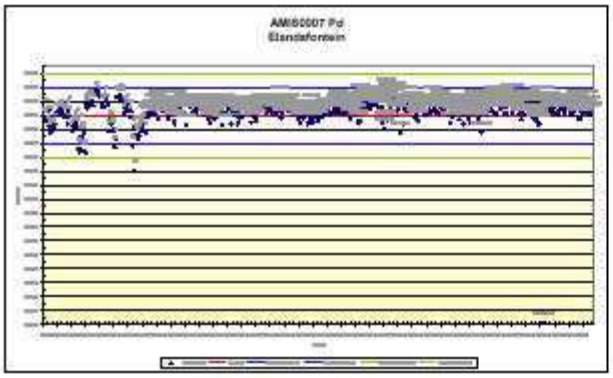
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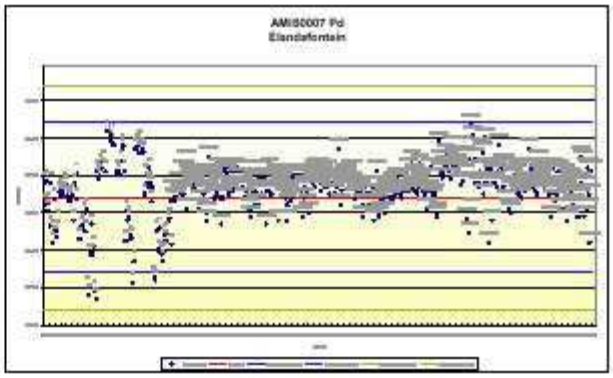
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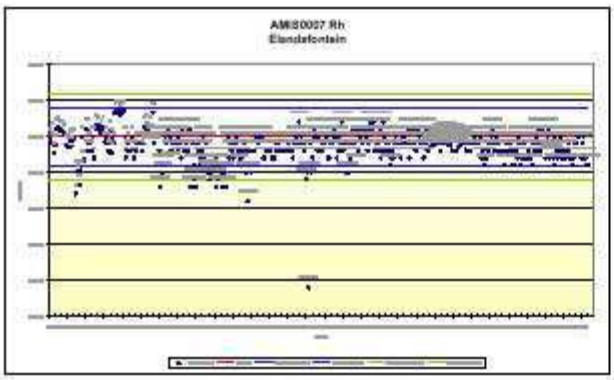
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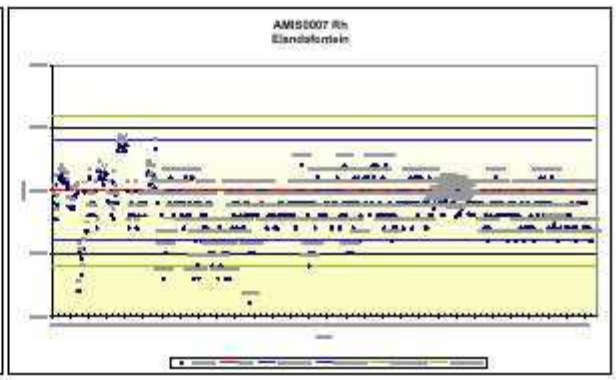
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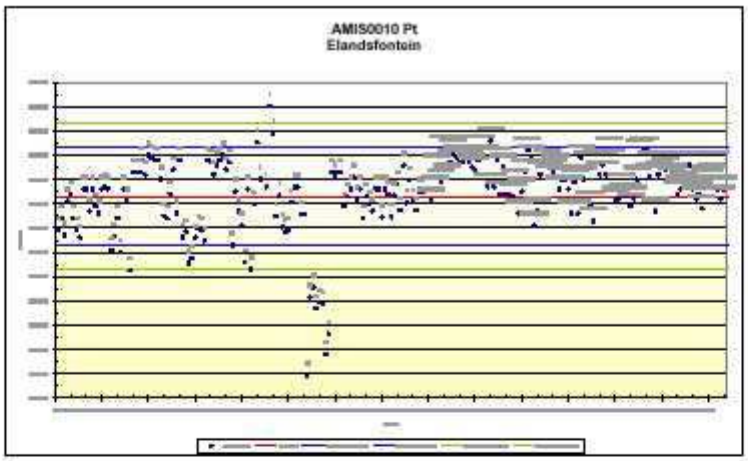
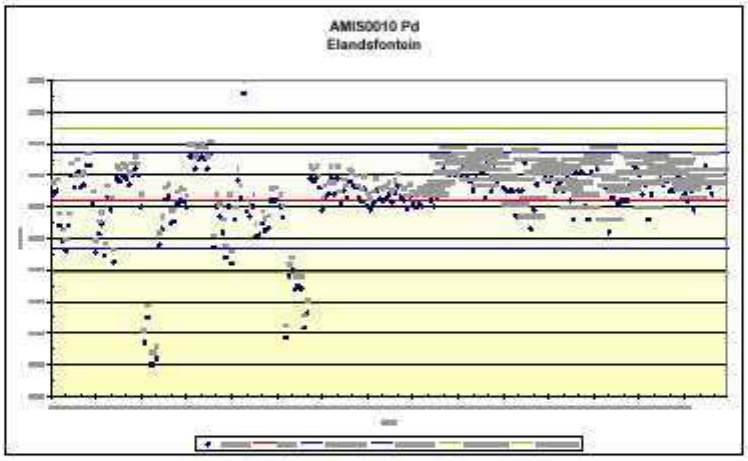
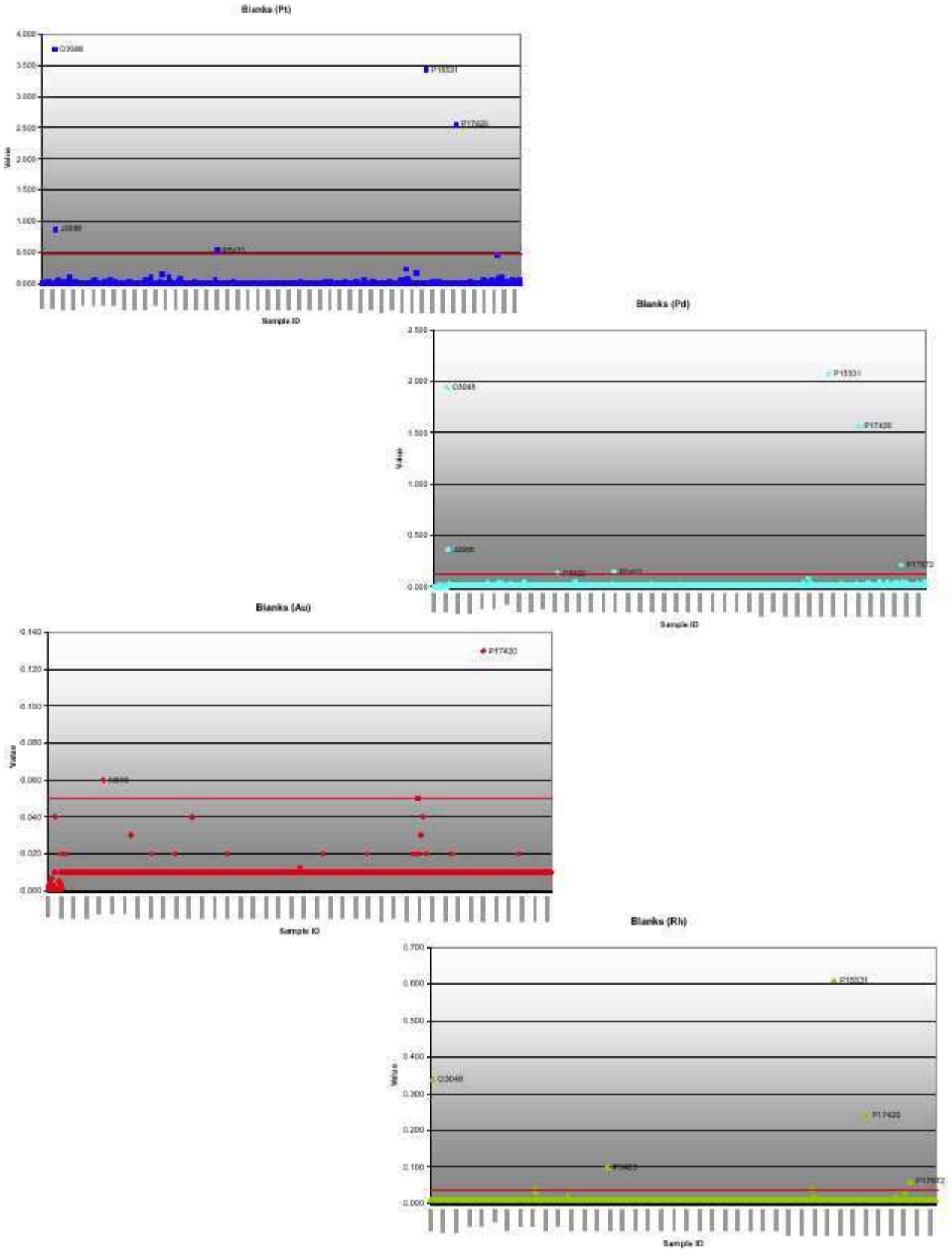
GRAPH 6





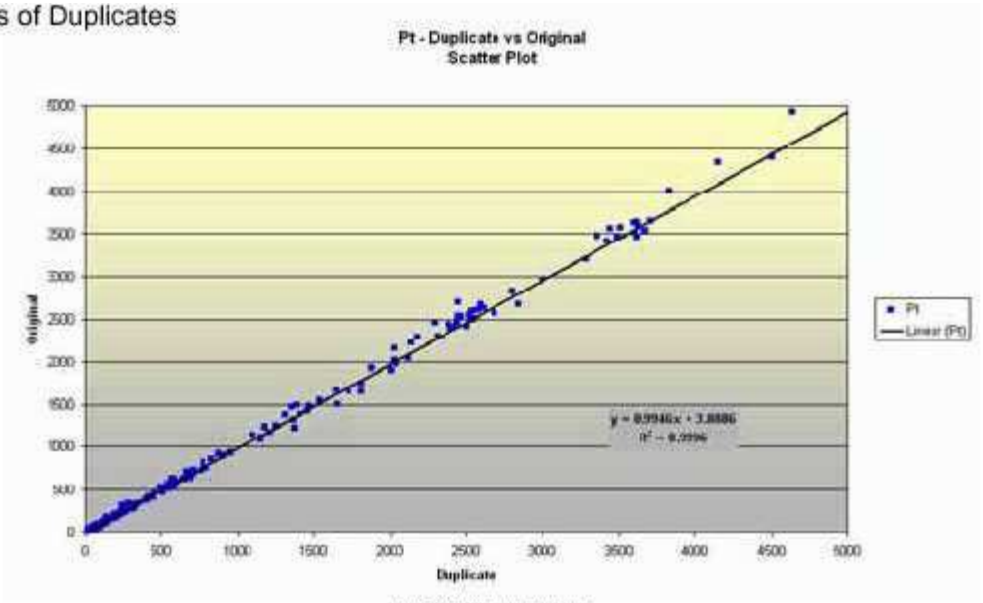
GRAPH 7

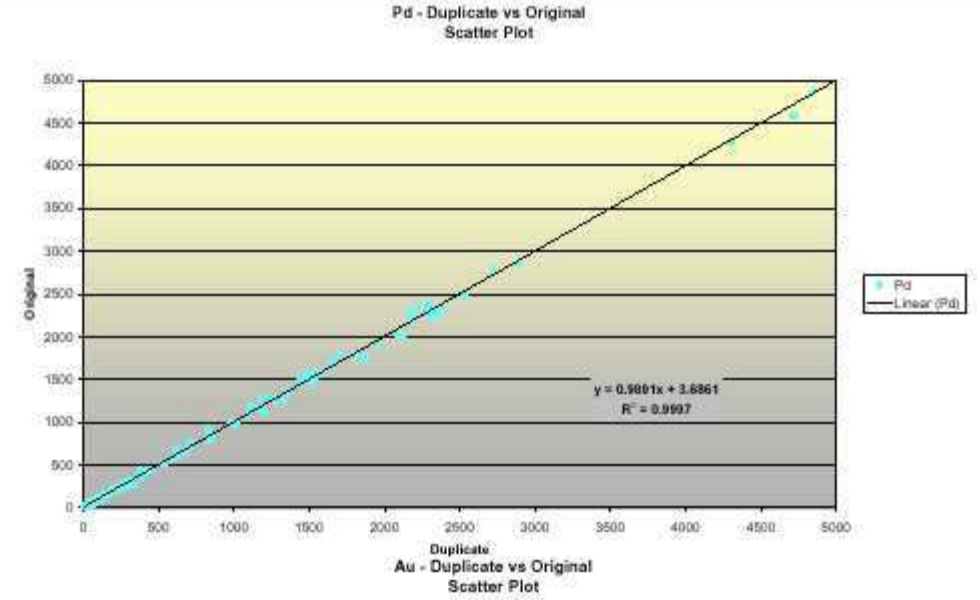
Plotted Graphs of Blanks

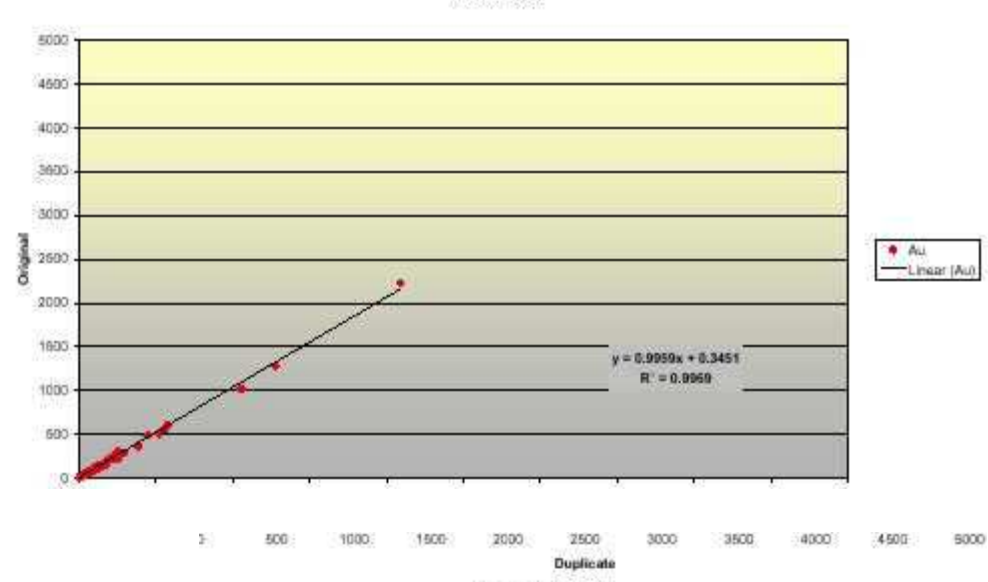


GRAPH 8

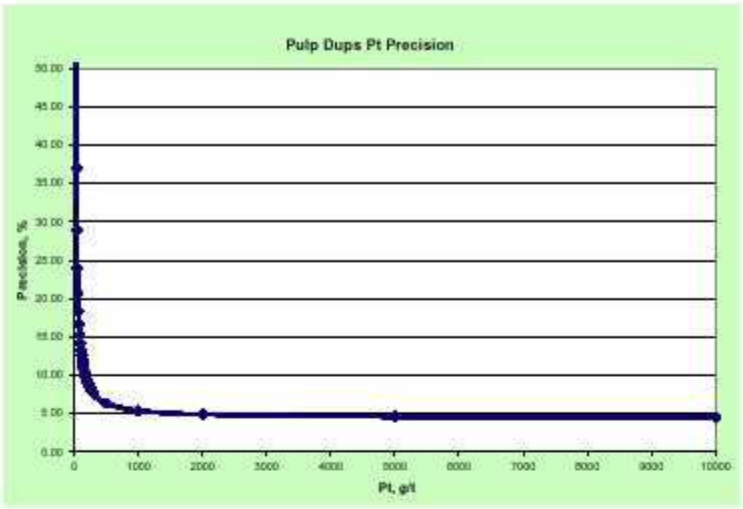
Plotted Graphs of Duplicates

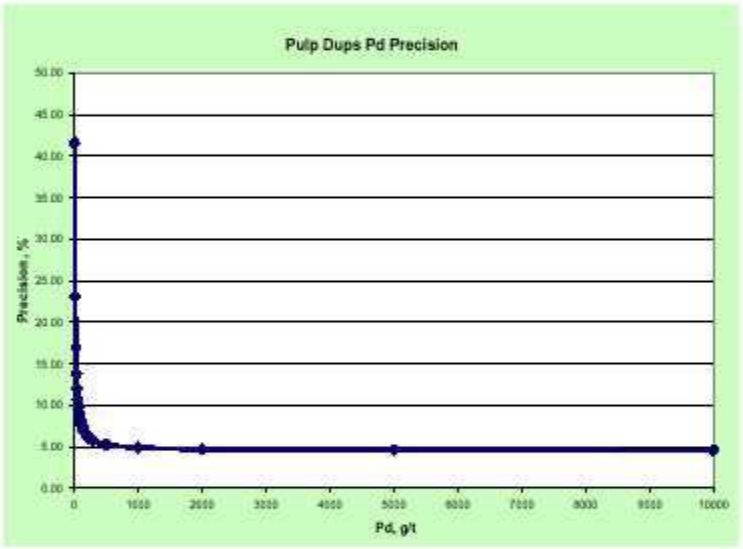






GRAPH 9


Pulp Dups Pt Precision
Precision, %
Pt, g/t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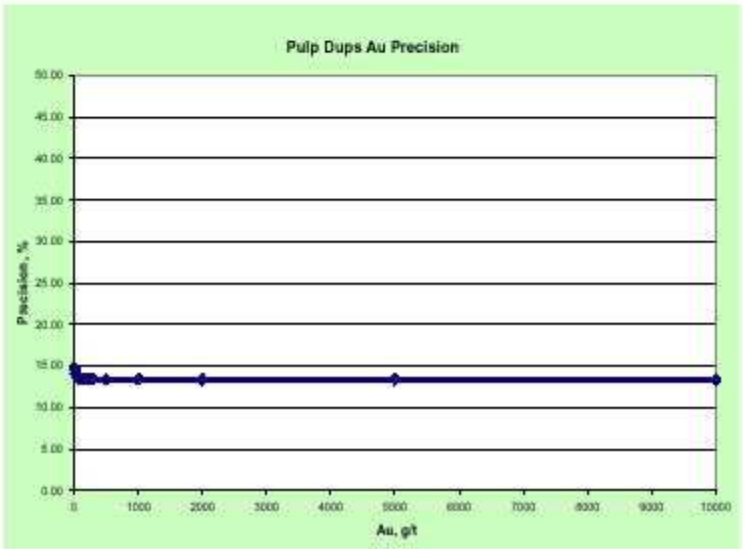
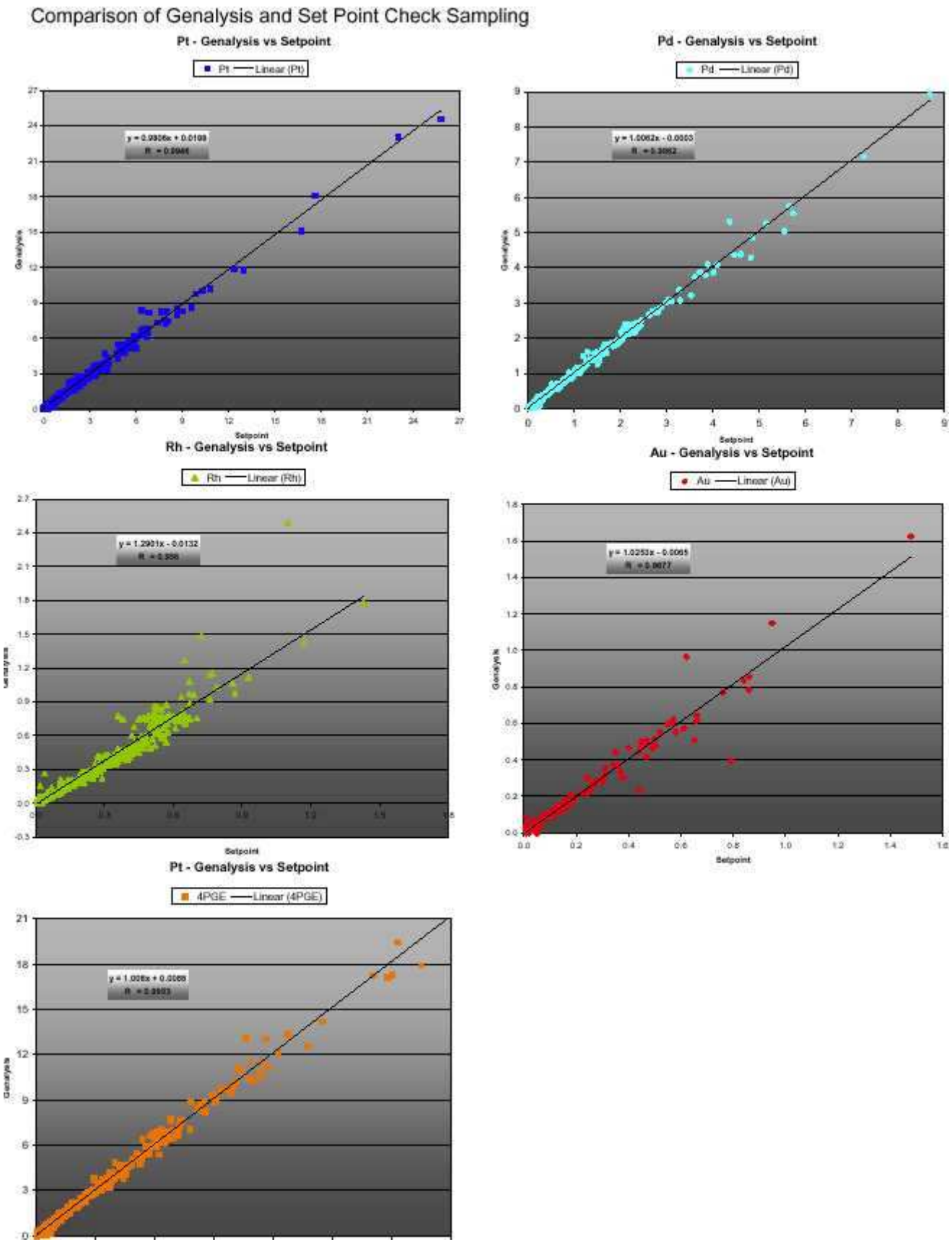
Precision, %
Pd, g/t


Pulp Dups Au Precision
Precision, %
Au, g/t

GRAPH 10



0 3 6 9 12 15 18 21

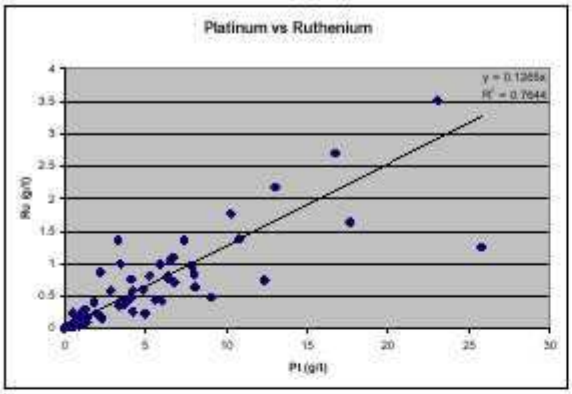
Setpoint

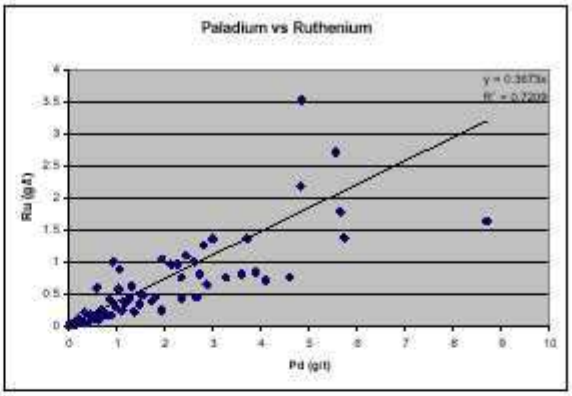
GRAPH 11

Plotted Graphs of Minor Elements (Ru, Ir, Os)

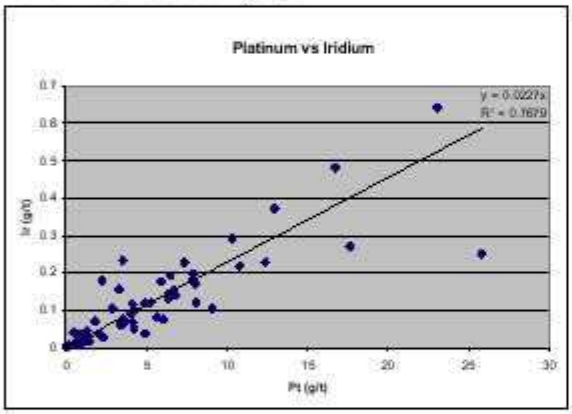
Merensky reef

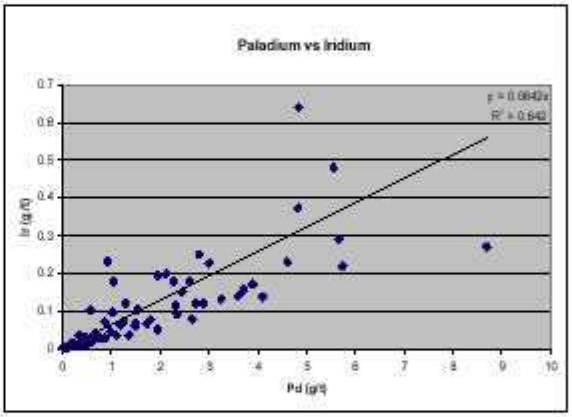
Ruthenium correlation graphs



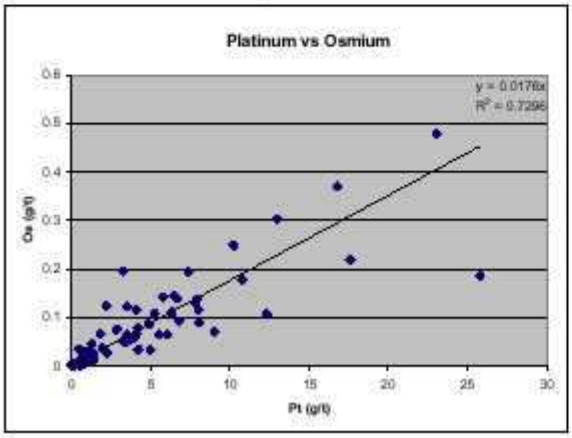


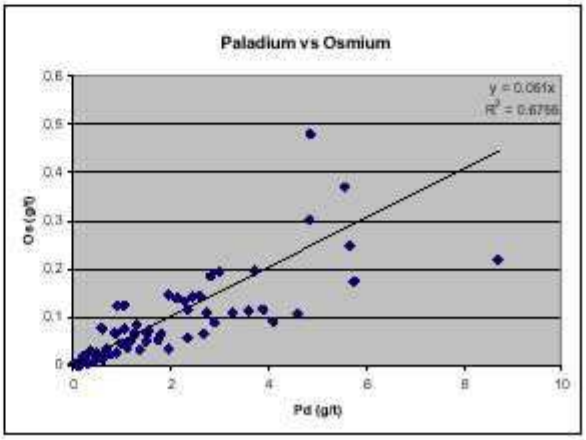
Iridium correlation graph





Osmium correlation graph

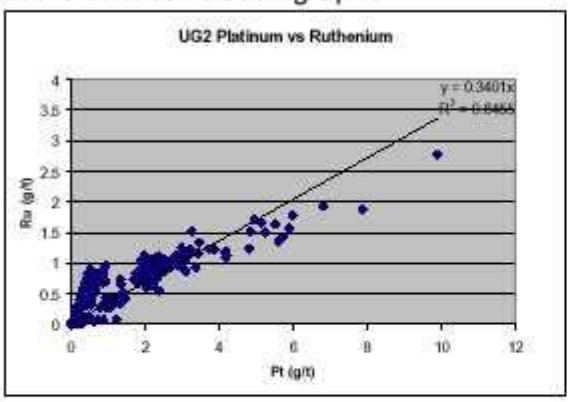


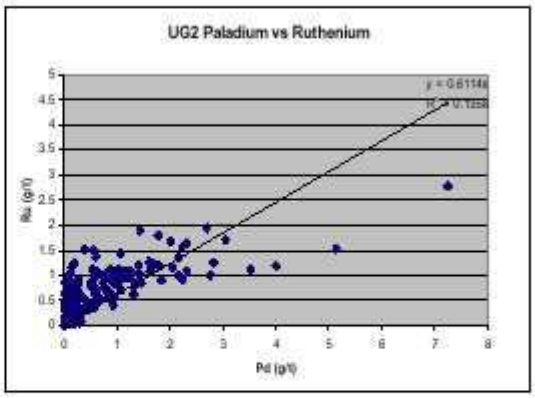


GRAPH 12

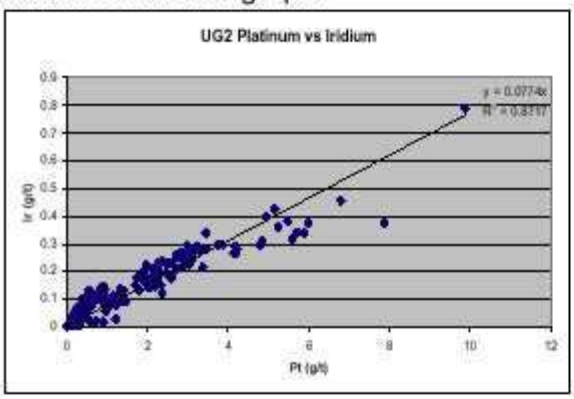
UG2 Reef

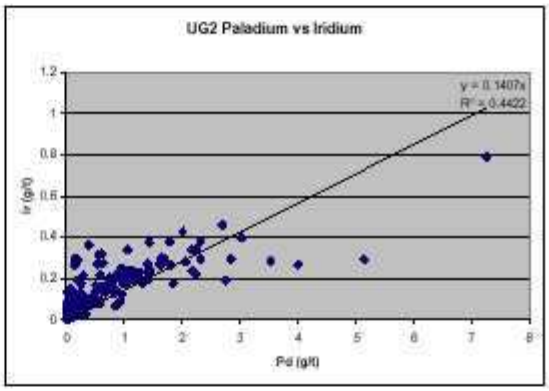
Ruthenium correlation graphs



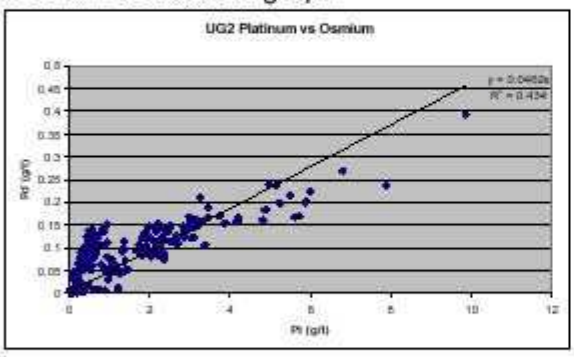


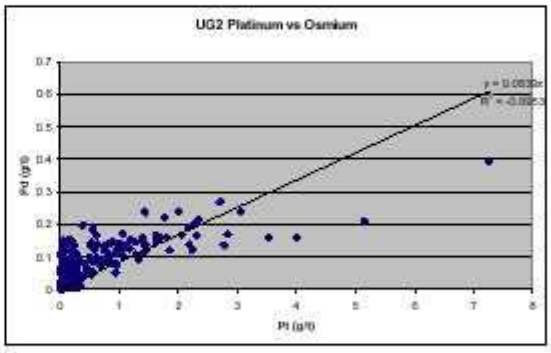
Iridium correlation graph





Osmium correlation graph





GRAPH 13

Turnberry Projects
(Pty) Ltd
Reg No. 1993/003160/07

CERTIFICATE OF AUTHOR

I, Gordon Ian CUNNINGHAM, B. Eng. (Chemical), Pr. Eng., do hereby certify that:

1. I am currently employed as a Director and Principal Engineer by:

   Turnberry Projects (Pty) Ltd.
   PO Box 2199
   Rivonia,
   Sandton 2128
   South Africa

2. I graduated from the University of Queensland (B. Eng. (Chemical) (1975)).

3. I am a member in good standing of the Engineering Council of South Africa and am registered as a Professional Engineer – Registration No. 920082. I am a member in good standing of the South African Institute of Mining and Metallurgy – Membership No. 19584.

4. I have worked as a Metallurgist in production for a total of 20 years since my graduation from university. I have worked as a Consulting Metallurgist for 5 years since graduation and I have been working for Turnberry Projects for 6 years as a Project and Principal Engineer and Director, primarily associated with mining and metallurgical projects.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with the professional associations (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I visited the property, viewed the core and discussed the technical issues and geology of the project with Willie Visser and John Gould of Platinum Group Metals RSA (Pty) Ltd. on numerous occasions over the period June 2005 to December 2006.

7. I am responsible for the preparation of sections of the report relating to the "Technical Report – Western Bushveld Joint Venture Project 1 – Elandsfontein and Frischgewaagd)", dated 15 January 2007. I have reviewed the entire Report and the work of other qualified persons who contributed to the Report. I, within reason and where appropriate, accept responsibility for the whole Report.

8. I have relied upon outside sources of information used in the completion of Items 18 and 25 of the Report. A dataset was compiled from technical data supplied by Anglo Platinum Limited as well as data collected during this study by Platinum Group Metals RSA (Pty) Ltd **.** Although the dataset is the responsibility of Platinum Group Metals RSA (Pty) Ltd., I have reviewed the dataset and have relied on the work of Qualified Person Charles Muller who has taken reasonable steps to provide comfort that the dataset is accurate and reliable. I am aware of no reason to believe the dataset is not accurate and reliable.

9. I am not aware of any material fact or material change with respect to the subject matter of the Pre- feasibility Study that is not reflected in the Pre-feasibility Study, the omission to disclose which makes the Review

misleading.

10. I am independent of the issuer, Platinum Group Metals (RSA) (Pty) Ltd. or any member of the Western Bushveld Joint Venture, applying all of the tests in Section 1.5 of NI 43-101.

11. I am familiar with the specific type of deposit found in the property area and its metallurgical aspects and have been involved in similar evaluations and technical compilations.

12. I have read National Instrument 43-101 and Form 43-101F1, and the Report has been prepared in compliance with that instrument and form.

"signed"

Gordon Ian Cunningham

B Eng. (Chemical), Pr. Eng.

Dated the 15 th day of January 2007

North Building, 272 Kent Avenue, Ferndale, Randburg, South Africa. Email: turnbery@iafrica.com
PO Box 2199, Rivonia, 2128, South Africa Tel: (011) 781 0116 Fax: (011) 781 0118 Cell: (083) 263 94

Director: G.I.Cunningham

Turnberry Projects
(Pty) Ltd
Reg No. 1993/003160/07

CERTIFICATE of AUTHOR

I, Timothy Vyvyan SPINDLER, B Sc. (Mining), Pr.Eng., do hereby certify that:

1. I am currently an Associate Principal Mining Engineer with:

Turnberry Projects (Pty) Ltd.
PO Box 2199
Rivonia,
Sandton 2128
South Africa

2. I graduated from the University of Witwatersrand (B Sc. (Mining) (1977)).

3. I am a member in good standing of the Engineering Council of South Africa and am registered as a Professional Engineer - Registration No. 880491. I am a member in good standing of the South African Institute of Mining and

Metallurgy – Membership No. 20021

4. I have worked as a Mining Engineer in production for a total of 16 years since my graduation from university. I have worked as a Consulting Mining Engineer for 12 years since graduation. I have been associated with Turnberry Projects for 5 years as a Principal Mining Engineer.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with professional associations (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of sections of the report relating to the "Technical Report – Western Bushveld Joint Venture Project 1 – Elandsfontein and Frischgewaagd)", dated 15 January 2007.

7. I have visited the property, viewed the core and discussed the technical issues and the geology of the project with Willie Visser and John Gould of Platinum Group Metals RSA

   (Pty) Ltd. on numerous occasions over the period June 2005 to November 2006. I am familiar with the specific type of deposit found in the area.

8. I am responsible for portions on Item 25 of this report. I have relied upon outside sources of information used in the completion of Item 25. A dataset was compiled from technical data supplied by Anglo Platinum Limited as well as data collected during this phase by Platinum Group Metals RSA (Pty) Ltd through drilling and modelling. Although the dataset is the responsibility of Platinum Group Metals RSA (Pty) Ltd., I have reviewed the dataset and have relied on the work of Qualified Person Charles Muller who has taken reasonable steps to provide comfort that the dataset is accurate and reliable. I am aware of no reason to believe the dataset is not accurate and reliable.

9. I am not aware of any material fact or material change with respect to the subject matter of the Pre- feasibility Study that is not reflected in the Pre-feasibility Study, the omission to disclose which makes the Review misleading.

10. I am independent of the issuer, Platinum Group Metals (RSA) (Pty) Ltd. or any member of the Western Bushveld Joint Venture, applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I am familiar with the specific type of deposit found in the property area and it's mining requirements and have been involved in similar evaluations and technical compilations.

12. I have read National Instrument 43-101 and Form 43-101F, and the report has been prepared in compliance with that instrument and form.

Dated this 15 th day of January 2007.

"signed"

_____

Timothy Vyvyan Spindler
B Sc. (Mining), Pr.Eng

North Building, 272 Kent Avenue, Ferndale, Randburg, South Africa. Email: turnbery@iafrica.com
PO Box 2199, Rivonia, 2128, South Africa Tel: (011) 781 0116 Fax: (011) 781 0118 Cell: (083) 263 94

Turnberry Projects
(Pty) Ltd
Reg No. 1993/003160/07

CONSENT OF QUALIFIED PERSON

Attention: **Alberta Securities Commission**
Autorité des marches financiers
British Columbia Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

(a) I, Gordon Ian Cunningham, B.Eng. (Chemical), Pr.Eng., a registered professional engineer with the Engineering Council of South Africa (Reg. No. 920082), am the co-author of the technical report entitled "Technical Report – Western Bushveld Joint Venture – Project 1 - Elandsfontein and Frischgewaagd", dated January 15 2007 (the "Report") and do hereby consent to the filing of the report with the regulatory authorities referred to above, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public. I further have consented to the company filing the report on SEDAR and consent to press releases made by the company with my prior approval. In particular I have read and approved the press release of Platinum Group Metals Ltd. dated January 10, 2007.

Dated this 15 th day of January 2007

"signed"

_____
Gordon Ian Cunningham B.Eng. (Chemical), Pr.Eng.

GLOBAL GEO SERVICES (PTY) LTD

Reg No2000/19232/07
Tel : 011 956 6264 P.O. Box 1574
Fax : 011 956 6264 Rant-enDal
global@ggs.co.za 1751

Professional Geoscience Services

CONSENT OF QUALIFIED PERSON

Attention:

Alberta Securities Commission
Autorité des marches financiers
British Columbia Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

(a) I, Charles Johannes Muller, BSc (Hons), Pr.Sc.Nat., a registered professional natural scientist with the South African Council for Natural Scientific Professionals (SACNASP) (Reg. No. 400201/04), am the co-author of the technical report entitled "Technical Report – Western Bushveld Joint Venture – Project 1 - Elandsfontein and Frischgewaagd", dated January 15 2007 (the "Report"), and do hereby consent to the filing of the report with the regulatory authorities referred to above, and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public. I further have consented to the company filing the report on SEDAR and consent to press releases made by the company with my prior approval. In particular I have read and approved the press release of Platinum Group Metals Limited dated 10 January, 2007 in which the findings of the Report are disclosed.

Dated this 15 th day of January 2007.

"signed"

_____
Charles Johannes Muller
BSc (Hons), Pr.Sc.Nat.

Turnberry Projects
(Pty) Ltd
Reg No. 1993/003160/07

CONSENT OF QUALIFIED PERSON

Attention:

Alberta Securities Commission
Autorité des marches financiers
British Columbia Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

(a) I, Timothy Vyvyan SPINDLER, BSc. (Mining), Pr.Eng., a registered professional engineer with the Engineering Council of South Africa (Reg. No. 880491), am the co-author of the technical report entitled "Technical Report – Western Bushveld Joint Venture – Project 1 - Elandsfontein and Frischgewaagd", dated January 15 2007 (the "Report") and do hereby consent to the filing of the report with the regulatory authorities referred to above, and any publication by them for regulatory purposes, including electronic publication in the public company files on

their websites accessible to the public. I further have consented to the company filing the report on SEDAR and consent to press releases made by the company with my prior approval. In particular I have read and approved the press release of Platinum Group Metals Ltd. dated January 10, 2007.

Dated this 15 th day of January 2007

"signed"

_____

Timothy Spindler
B Sc. (Mining), Pr. Eng.