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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 the period of: November, 2009

Platinum Group Metals Ltd.

(SEC File No. 0-30306)

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

Indicate by check mark whether the registrant files or will file annual reports under cover: Form 20-F [ ] Form 40-F [X]

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1): [ ]

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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.

Date: **December 16, 2009** *"R. Michael Jones"*

R. MICHAEL JONES
DIRECTOR & CEO

Exhibit List

Ex. 99.1 Technical Report
Ex. 99.2 Certificate of Author - Charles Johannes Muller
Ex. 99.3 Consent of Qualified Person - Charles Johannes Muller
Ex. 99.4 Certificate of Author - Gordon Ian Cunningham
Ex. 99.5 Consent of Qualified Person - Gordon Ian Cunningham
Ex. 99.6 Certificate of Author - Timothy Vyvyan Spindler
Ex. 99.7 Consent of Qualified Person - Timothy Vyvyan Spindler

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: NYSE AMEX - PLG

UPDATED TECHNICAL REPORT (UPDATED FEASIBILITY STUDY)

Western Bushveld Joint Venture

PROJECT 1

(ELANDSFONTEIN AND FRISCHGEWAAGD)

A REPORT ON THE UPDATED FEASIBILITY STUDY FOR A PORTION OF THE
WESTERN BUSHVELD JOINT VENTURE

TURNBERRY PROJECTS (PTY) LTD
MELVILLE, JOHANNESBURG,
REPUBLIC OF SOUTH AFRICA

MINXCON (PTY) LTD
SUITE 5, COLDSTREAM OFFICE PARK, LITTLE FALLS, ROODEPOORT
GAUTENG, REPUBLIC OF SOUTH AFRICA

Date of Report - 20 November 2009

Effective Date of Report - 08 October 2009

IMPORTANT NOTICE

This Technical Report on the Updated Feasibility Study includes only the results for Resources announced by Platinum Group Metals Ltd (“Platinum Group”) on 07 July 2008 (news release filed with SEDAR), but **excludes** the new Resource that was Press Released on 08 October 2009 and detailed in this Technical Report being issued on or about 20 November 2009. This report communicates Indicated and Measured Resources based on the Feasibility Study (filed 21 August 2008 on Sedar). 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 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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ANNEXURES

APPENDIX A
Mine Design and Production Scheduling

ITEM 3: SUMMARY

Since the release of the Feasibility Study in July 2008, changes have occurred to the anticipated capital and working costs, applied metal pricing and the improvement in grid power supply expectations. In addition, the method of access and off reef development has been reviewed. In the Feasibility Study, access development was to be via three decline systems with trackless development and ore transport, with raise development, ledging, equipping and stoping being conventional hand-held mining methodology and this approach has been maintained in the updated Feasibility Study.

Doubts over the power authorities ability to provide the necessary power have eased in South Africa, resulting in the reduced dependence upon self generation capacity of 100% of mine power requirements.

This three decline systems has been refined further and is the basis of this Updated Feasibility Study.

Reason for Updated Feasibility Study Report

The project team was requested by Platinum Group to update the Feasibility Study because of perceived changes in the international economic environment, revised mining methodology and the improvement in the perception of available grid power from ESKOM.

The property and terms of reference

The Western Bushveld Joint Venture ("WBJV") Project 1 is located in the heart of the Western Bushveld area of South Africa where 70% of the world's platinum is produced from the Merensky and UG2 Platinum; the same two horizons to be mined in the Updated Feasibility Study. The former active partners in the WBJV are Platinum Group 37% (operator), Anglo Platinum (AMS-JSE) 37% (the world's largest producer of platinum), and Wesizwe Platinum ("Wesizwe") (WEZ-JSE) 26%. Wesizwe is a company founded on Black Economic Empowerment principles as required under the Mineral and Petroleum Resources Development Act, 2002. The partners have agreed to restructure the WBJV. Under the revised structure that is in the process of government approval, Platinum Group will hold 74% and Wesizwe will hold 26%. The joint venture is a notorial contract and managed by a committee representing all partners. Platinum Group Metals RSA (Pty) Ltd ("PTM RSA") is the operator of the joint venture.

In a news release dated 2 September 2008, and subsequently updated on 9 December 2008, PTM announced that it has agreed to a term sheet with AP and Wesizwe to consolidate and rationalize the WBJV. Under the terms PTM will have the right to acquire effective ownership of 74% of WBJV Projects 1 and 3 and Wesizwe will acquire 100% of Project 2 and 26% of Projects 1 and 3. The transactions will become effective upon fulfilment of certain conditions precedent and regulatory approvals including the approval of the DME, Republic of South Africa, for transfer of Mineral Rights in accordance with Section 11 of the MPRDA .

This Updated Feasibility Study 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 (Minxcon), Mr TV Spindler (Turnberry Projects (Pty) Ltd) and Mr B Stewart (Wardrop Engineer).

The principle Qualified Person and other qualified experts have visited the WBJV Project 1 site and on several occasions throughout the period from 2005 to 2009 and detailed discussions were held with Platinum Group and PTM RSA technical personnel at the PTM (RSA) 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 ("MR") 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 conversions.

Geology

The WBJV property is 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 of 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 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 1.15m and the UG2 Reef diluted mining width is 1.53m. The grade content - centimetre gram per ton (cm.g/t) - was used as a Resource cut-off. Indicated Resources total 5.010 million ounces for 4E (platinum, palladium, rhodium and gold) for Project 1. In addition, the Resource calculation includes a Measured Resource of 2.224 million ounces 4E. This brings the total updated Indicated and Measured Resource base to an estimated 7.234 million ounces 4E. The Inferred Resource estimate is 1.256 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.

The historical Resource (feasibility study) estimate is shown in the following tables, as this Resource is the basis for the Updated Feasibility Study.

The updated resources completed at the same time as this Feasibility Study, are presented herein.

Mineral Resources

Table 1 - WBJV Project 1 - Mineral Resource Statement

Measured Mineral Resource (4E)	Cut-off 4E (cm.g/t)	Million Tonnes (Mt)	Grade 4E (g/t)	Resource Cut Width (m)	Tonnes PGE (4E) (t)	Metal Content (4E) (Moz)
Project 1 MR	300	5.491	7.94	1.08	43.599	1.402
Project 1 UG2	300	6.539	3.91	1.41	25.568	0.822
Total Measured	**300**	**12.030**	**5.75**	**1.26**	**69.173**	**2.224**

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 MR	64%	5.08	27%	2.14	4%	0.318	5%	0.398
Project 1 UG2	63%	2.46	26%	1.02	10%	0.39	1%	0.04

Indicated Mineral Resource (4E)	Cut-off 4E (cm.g/t)	Million Tonnes (Mt)	Grade 4E (g/t)	Resource Cut Width (m)	Tonnes PGE (4E) (t)	Metal Content (4E) (Moz)
Project 1 MR	300	10.814	7.75	1.09	83,809	2.695
Project 1 UG2	300	17.464	4.13	1.34	72,126	2.319
Total Indicated	**300**	**28.278**	**5.51**	**1.24**	**155,812**	**5.010**

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 MR	64%	4.96	27%	2.09	4%	0.31	5%	0.39
Project 1 UG2	63%	2.60	26%	1.08	10%	0.41	1%	0.04

Inferred Mineral Resource (4E)	Cut-off 4E (cm.g/t)	Million Tonnes (Mt)	Grade 4E (g/t)	Resource Cut Width (m)	Tonnes PGE (4E) (t)	Metal Content (4E) (Moz)
Project 1 MR	300	0.217	7.95	0.93	1,725	0.055
Project 1 UG2	300	2.311	4.47	1.34	10,330	0.332
Project 1A MR	300	1.871	6.48	1.15	12,124	0.390
Project 1A UG2	300	2.973	5.00	1.57	14,865	0.478
Total inferred	**300**	**7.372**	**5.30**	**1.37**	**39,072**	**1.256**

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 MR	64%	5.09	27%	2.15	4%	0.32	5%	0.40
Project 1 UG2	63%	2.82	26%	1.16	10%	0.45	1%	0.04
Project 1A MR	64%	4.15	27%	1.75	4%	0.26	5%	0.32
Project 1A UG2	63%	3.15	26%	1.30	10%	0.50	1%	0.05

MR- Merensky Reef; UG2- Upper Group 2 Reef

The mineral Resources reported as part of this Updated Feasibility Study are updated from previously reported Resources and filed in a Technical Report on SEDAR on October 30, 2007. The Resources were completed to conform to a minimum 80cm Resource cut which is in line with that used by Anglo Platinum. Sampling practice, bore hole data, other factors and quality control and assurance are as reported previously. The Resources are estimated by kriging of approximately 181 boreholes plus deflections and are reported under SAMREC. The categories are the same as CIM categories.

The Qualified Person for the Resources is Charles Muller of Minxcon.

A new Resource statement has been press released on 08 October 2009, but this new Resource has **NOT** been used in the Updated Feasibility Study and is thus not referred to in this Technical Report. A Separate N1 43-101 has been prepared on the new resources by Minxcon and will be filed simultaneously to this report

Mineral Reserves - derived from the Measured & Indicated Resources (not in addition to them)

A Probable Reserve is the economically mineable part of an Indicated, and in some circumstances a Measured Resource, demonstrated by at least a Pre-Feasibility Study including adequate information on mining, processing, metallurgical, economic and other factors that demonstrate, at the time of reporting, the economic extraction can be justified. A Proven Reserve is the economically mineable part of a Measured Resource demonstrated by the same level and factors as above. A Proven Mineral Reserve implies that there is a high degree of confidence. All approvals need not be in place for the declaration of Reserves.

The conversion to Mineral Reserves was undertaken at 3.5g/t stope cut-off grade from the historical Resource (and NOT the new Resource), each stope has been fully diluted, comprising of a planned dilution and additional dilution for all aspects of the mining process. The Inferred Resources are outside and in addition to the Reserves. The Qualified Person for the Statement of Reserves is Tim Spindler.

Mineral Reserves 4E

Merensky

Tonnage	4E	Content 4E	
t	g/t	Tonne	Moz
Merensky Proved			
6,677,304	5.61	37.478	1.205
Merensky Probable			
11,333,294	5.44	61.677	1.983
Total Merensky Mineral Reserves			
18,010,598	5.51	99.155	3.188

UG2

Tonnage	4E	Content 4E	
t	g/t	Tonne	Moz
UG2 Proved			
5,086,535	3.37	17.126	0.551
UG2 Probable			
8,448,801	3.41	28.831	0.927
Total UG2 Mineral Reserves			
13,535,336	3.40	45.957	1.478

Table 2 - Reserves - Converted from Measured and Indicated Resources

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 & 1A MR	64%	3.52	27%	1.49	4%	0.22	5%	0.28
Project 1 & 1A UG2	63%	2.15	26%	0.88	10%	0.34	1%	0.03

To convert the above Reserve Statement to individual elements as requested, the Resource Prill Splits have been applied to the 4E content to create the list of individual elements depicted in Table 3 . The individual element list as depicted in this table do not constitute a Reserve on an individual basis, only when referred to as 4E is it possible to report the Reserve.

Table 3 - Listing of Individual Elements from 4E Reserve Statement

Metal	Reef	Category	Prill Split	Tonnage Tonnes	g/t 4E	4E Content Tonne	4E Content Moz
Pt	Merensky	From Proved 4E Reserve	64%	6,677,304	3.59	23.99	0.771
		From Probable 4E Reserve	64%	11,333,294	3.48	39.47	1.269
		Total	64%	18,010,598	3.52	63.46	2.040
	UG2	From Proved 4E Reserve	63%	5,086,535	2.12	10.79	0.347
		From Probable 4E Reserve	63%	8,448,801	2.15	18.16	0.584
		Total	63%	13,535,336	2.14	28.95	0.931
Pd	Merensky	From Proved 4E Reserve	27%	6,677,304	1.52	10.12	0.325
		From Probable 4E Reserve	27%	11,333,294	1.47	16.65	0.535
		Total	27%	18,010,598	1.49	26.77	0.861
	UG2	From Proved 4E Reserve	26%	5,086,535	0.88	4.45	0.143
		From Probable 4E Reserve	26%	8,448,801	0.89	7.50	0.241
		Total	26%	13,535,336	0.88	11.95	0.384
Rh	Merensky	From Proved 4E Reserve	4%	6,677,304	0.22	1.50	0.048
		From Probable 4E Reserve	4%	11,333,294	0.22	2.47	0.079
		Total	4%	18,010,598	0.22	3.97	0.128
	UG2	From Proved 4E Reserve	10%	5,086,535	0.34	1.71	0.055
		From Probable 4E Reserve	10%	8,448,801	0.34	2.88	0.093
		Total	10%	13,535,336	0.34	4.60	0.148
Au	Merensky	From Proved 4E Reserve	5%	6,677,304	0.28	1.87	0.060
		From Probable 4E Reserve	5%	11,333,294	0.27	3.08	0.099
		Total	5%	18,010,598	0.28	4.96	0.159
	UG2	From Proved 4E Reserve	1%	5,086,535	0.03	0.17	0.006
		From Probable 4E Reserve	1%	8,448,801	0.03	0.29	0.009
		Total	1%	13,535,336	0.03	0.46	0.015

The prill splits are the same percentages as for the Measured and Indicated Resources. The effective date of the Reserve estimate is June 30, 2008. The Reserves are stated with certain risk factors including but not limited to mining projects risks as highlighted in the Risks and Opportunities.

Status of exploration

PTM RSA has completed approximately 122,361m of BQ-size core drilling (diameter 36.2mm) from borehole WBJV001 to WBJV181. All exploration has ceased on the site of Project 1. 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 14% and 23% for the UG2 Reef (this includes, minor faults, dykes and potholes) (major faults, dykes, weathered zone and iron replacement areas have already been excluded by delineating areas with no reef) 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.

Additional drilling has been completed and this has formed the basis of the new Resource, press released on 08 October 2009, but it must be noted that this new Resource has **NOT** been used in this Technical Report.

Potential development considerations

The results of this Updated Feasibility Study represent the culmination of 5 months of work by independent engineers from Turnberry Projects of South Africa, Platinum Group own engineers from South Africa and Canada, and a team of specialists from several South African firms.

Mine Plan Details

Improvements have been made to mine design and the projected maximum output has been increased to fully utilize the availability and flexibility of three separate decline systems.

The mining of UG2 material has increased due to increased panel length and better overall extraction ratio. The Updated Feasibility Study has removed the overall constraint of 48,000 tonnes per month per decline system. Basic production parameters have remained unchanged. This has resulted in a higher steady state production level of approximately 160,000 reef tonnes per month. This will require each individual decline system to produce approximately 57,000 tonnes per month. Although similar systems often produce between 80,000 and 100,000 reef tonnes per month, 57,000 reef tonnes per month is considered to be realistic given the strike width available from each decline system`, which provides 13 years of steady state tonnage production. First ore is reached by development 13 months from the commencement of underground work. First ore will be processed in the concentrator in first quarter 2013 subject to the installation of grid power from ESKOM. Ore will be stockpiled from mining until the concentrator is operational. Mining is only scheduled on the Reserves. The updated Resource which includes additional Indicated and Measured categories and the lower grade UG2 Resources also provide some future opportunities. The mining and development plan includes conventional hand held drilling utilizing electrical drills and scrapers as well as winch cleaning similar to the successful conventional mining operations at the adjacent mines. Declines and primary access to the deposit is designed for development with mechanized equipment. Ore is initially to be hauled out of the mine with mechanized equipment and assisted later by conveyor from year 4 of the planned mine life to end of mine life.

The Merensky Reef will be mined at widths between 93cm and 176cm and an average of 115cm. The UG2 Reef will be mined at widths between 105cm and 205cm and an average of 153cm.

At the recommended mining rate and modifying factors the mine plan generates approximately 234,000 to 300,000 4E ozs in concentrate per year. During 9 years of peak production approximately 275,000 4E ozs are produced from the Merensky Reef horizon. The mine has a 22 year planned mine life.

Infrastructure and Metal Recovery

The Updated Feasibility Study design for metallurgical extraction utilizes a standard plant design similar to other nearby plants in the Bushveld complex operating on the same reefs. The plant is designed with circuits that can process either Merensky Reef, UG2 Reef or a blended feed. The Merensky Reef is the target of initial mining because of its higher grade and low chrome content. The concentrator has been designed and re-costed, based on treating the optimal 140,000 tonnes per month. The revised mine plan has increased this treatment rate to 160,000 tonnes per month and for the concentrator to treat this increased quantity of reef, the recovery has been reduced with a discount of up to 2.5% for treatment in excess of nominal “name plate” capacity.

Metallurgical testing and the published experience of the adjacent operating mines support a “name plate” capacity plant recovery rate estimate of 87.5% of platinum, palladium, rhodium and gold on the Merensky Reef and 82.5% on the UG2 Reef. Recoveries of 45% for nickel and 70% for copper are also modeled for the Merensky Reef. Ruthenium and Iridium are also included as minor contributors. No additional metallurgical test work has been undertaken during the Updated Feasibility Study.

The mine infrastructure in the estimates includes the entire required surface infrastructure for a standalone mine including water, power, underground access and ventilation to establish full production.

Smelter Terms

The Updated Feasibility Study includes capital and operating estimates to produce concentrate but no capital is included for smelting or refining of this concentrate. The costs associated with smelting and refining of concentrate is modeled as a deduction from revenue arising from the sale of concentrate to others. While the terms of agreements governing the sale of such concentrates within the South African PGM industry are all confidential the Qualified Person believes deductions used in the Updated Feasibility Study financial model are indicative of deductions current in this industry. The party to whom concentrate will be sold and the terms of this potential sale are yet to be determined. Anglo Platinum has a 60 day right of first refusal to purchase all of the ore or concentrate produced by the WBJV on commercial terms. Estimated deductions in the Updated Feasibility Study include penalties and shipment charges and total approximately 15% from gross concentrate sales revenue. Should Anglo Platinum decide to purchase the concentrate produced by the Project 1 mine the structure of such purchase would be governed by the pro-forma off-take agreement included in the WBJV Agreement, however the commercial terms will be subject to negotiation. Approaches will now be made to Anglo Platinum and other parties in an attempt to secure an off-take agreement and the terms thereof based on the Updated Feasibility Study production profile.

Financial Details

The following tables set out the key details of the Updated Independent Feasibility Study

Table 4 - Metal Prices - 3 year Trailing and September 2009

METAL PRICES & EXCHANGE RATE		Case A 3 Year Trailing Prices	Case B Recent Prices
Platinum	(US$/oz)	1,343	1,295
Palladium	(US$/oz)	322	296
Rhodium	(US$/oz)	4,951	1,650
Gold	(US$/oz)	807	997
Copper	(US$/tonne)	6,428	6,250
Nickel	(US$/tonne)	25,529	18,000
Rand/US$		8.00	8.00

FINANCIAL PARAMETERS AND INDICATORS at R8.00				
Basket Prices 4E Pt, Pd, Rh, Au	R/kg Case A	US$/oz Case A	R/kg Case B	US$/oz Case B
Basket Price Merensky Reef	304,751	1,185	263,526	1,025
Basket Price UG2 Reef	368,571	1,433	274,639	1,068

Table 5 - Smelter and Refinery Discount

Smelter and Refinery Discount	Smelter Discount	US$/oz Case A	US$/oz Case B
Reduction in Basket Price MR	15.16%	$180	$155
Reduction in Basket Price UG2	15.16%	$217	$162
Received Basket Price for MR	15.16%	$1,005	$869
Received Basket Price for UG2	15.16%	$1,216	$906

Table 6 - South African taxes

TAXES		
Government Royalty Payment: PGM	(% of Revenue)	3%
Government Royalty Payment: Base Metals	(% of Revenue)	2%
Company Tax	(% of Profit)	28%
Secondary Tax on Companies (STC)	(% of profit)	10%

Table 7 - Summary capital costs

CAPITAL COST	Rand	US$
Peak funding Case A	3,545,237,000	443,125 000
Peak funding Case B	3,727,106,000	465,888,000
Total Life of Mine Capital Costs	4,760,280,000	595,035,000

Table 8 - Summary working costs

WORKING COSTS EXCLUDING SMELTER DISCOUNT	US$/4E oz	R / tonne milled
Cost during ramp up excluding any power generation	$543	R717
Cost at steady state grid power post 2013	$521	R491
Life of Mine Average	$526	R525

Table 9 - Economic Evaluation Summary

EVALUATION (NET PRESENT VALUE)		NPV 5% Discount	NPV 10% Discount
3 Year Trailing Price (Case A)	R (Million)	7,848	3,804
(Pre-tax)	US$ (Million)	981	475
	IRR		23.54%
3 Year Trailing Price (Case A)	R (Million)	4,660	2,031
(Post Tax)	US$ (Million)	582	254
	IRR		**18.94%**
Recent Prices (Case B)	R (Million)	3,671	1,330
(Pre-tax)	US$ (Million)	459	166
	IRR		**15.63%**
Recent Prices (Case B)	R (Million)	2,016	425
(Post Tax)	US$ (Million)	252	53
	IRR		**12.15%**

Table 10 - Sensitivity at 3-year Trailing Metal Prices

PRE-TAX

Parameter	Change in Parameter	Change in Parameter	Change in Parameter
PGM Basket Price	-20%	0%	20%
IRR (pre-tax)	14.5%	23.5%	31.2%
NPV (5% Discount) R(M)	3,494	7,848	12,202
US$ (M)	437	981	1,525
Opex	-20%	0%	20%
IRR (after tax)	27.5%	23.5%	19.4%
NPV (5% Discount) R (M)	9,835	7,848	5,861
US$ (M)	1,229	981	733
Capex	-20%	0%	20%
IRR (after tax)	28.4%	23.5%	19.9%
NPV (5% Discount) R (M)	8,635	7,848	7,061
US$ (M)	1,079	981	883

POST-TAX

Parameter	Change in Parameter	Change in Parameter	Change in Parameter
PGM Basket Price	-20%	0%	20%
IRR (post-tax)	11.3%	18.9%	25.3%
NPV (5% Discount) R (M)	1,892	4,660	7,393
US$ (M)	236	583	924
Opex	-20%	0%	20%
IRR (post-tax)	22.3%	18.9%	15.4%
NPV (5% Discount) R (M)	5,917	4,660	3,393
US$ (M)	740	583	424
Capex	-20%	0%	20%
IRR (post-tax)	23.0%	18.9%	15.9%
NPV (5% Discount) R (M)	5,202	4,660	4,111
US$ (M)	650	583	514

Social Development and Responsibilities

Feedback from the public consultation processes for the environmental assessment and social and labour plan development have been constructive and positive. This work has continued during 2008 and 2009 since the original July 2008 study. Consultation with the Department of Mineral Resources on the mine plan and social and labour plan has been positive. The mine capital development plan includes a significant investment in training through the life of mine, allocated to a social and labour plan to ensure maximum value from the project for all stakeholders including local residents. Based on interaction with the community, a skills and needs assessment, and our training plans, the project is planning for 2,700 jobs with a target of at least 30% from the local communities. The WBJV is committed to a strong community involvement in the project particularly as Wesizwe is a 26% partner in the project and their largest shareholder is one of the communities near the mine. The mine's financial estimates include an accumulated charge per tonne to create a fund for eventual closure of the mine projected in approximately 2031.

Risks and Opportunities

The project is subject to a number of risks including, but not limited to, the normal project risks associated with a mining project of this type, such as geology, grade, structure, mining plans, mining width, mining dilution, rates of extraction per reef type, water supply, power supply and labour shortages and estimation risks in the capital and operating costs and exchange rates and metal prices. The estimates in September 2009 money terms are ±10% on the capital and ±10% on the operating costs and ±15% on project timing. A contingency of 8.5% of the capital cost required to reach first production included in the financial model mitigates the risks of a capital over run. The estimated capital costs of Platinum Group's share of the project exceed its working capital and significant project funding is required. The transfer of the majority of the WBJV property has been approved by the Minister of the Department of Mineral Resources and final registration is in process. Final registration of title at the deeds office completes the 74% Platinum Group right and the agreements for finalization have been extended to February 2010 if required.

Not all of the surface rights over the proposed infrastructure design in the Updated Feasibility Study have been purchased. Negotiations are well advanced. An allowance for the cost of this has been made. The Mineral Petroleum Development Act provides firm assistance in ensuring that access to the minerals can be achieved; however if the process under the Act is required it may require additional time.

A significant portion of the defined Resources are not mined in the Updated Feasibility Study. Continued strong metal prices and implementation level engineering will provide an opportunity to re-evaluate these Resources. There are other platinum mining projects in the immediate area planned for development including Project 2 of the WBJV which is to be controlled 100% by Wesizwe in the restructured WBJV. Potential cost savings and reduced impacts by designing combined infrastructure are being investigated.

Qualified Person and Quality Assurance and Control

The Updated Feasibility Study has been completed by Gordon Cunningham (FSAIMM and ECSA) and Tim Spindler (FSAIMM and ECSA) as the projects leads of Turnberry Projects (Pty) Ltd. and Byron Stewart P.Eng of Wardrop Engineering Inc. as the lead for underground mechanized access and footwall mining. They have been supported by the following qualified specialists and firms:

Grinaker-LTA Mining	Construction & Mining Engineering
GRD Minproc	Process Engineering & Construction
Epoch Resources	Mine Residue & Environmental Engineering
Bluhm Burton Engineering Consulting	Mine Ventilation & Refrigeration Specialists
Dave Arnold & Associates	Rock Mechanics
Minxcon	Independent Consultants in Geology

As operator of the WBJV, Platinum Group Metals Ltd. is responsible for the management, exploration and engineering activities.

The Resources are estimated by Charles Muller of Minxcon (SACNASP) as independent consultant and he has completed the investigation and report in compliance with Canadian National Instrument 43-101 on this and the previous announced estimate. He provides the update given here on Resources.

Gordon Cunningham of Turnberry Projects (Pty) Ltd. (FSAIMM and ECSA) is the overall Qualified Person for NI 43-101 Technical Report which will support the statements made in the Updated Feasibility Study. He is a fellow with the South African Institute of Mining and Metallurgy (“SAIMM”), has had extensive experience in the assessment of mines including platinum mines and has been responsible for the mine designs and related investigations. He has over 30 years experience in project operation, project evaluation, design and feasibility work and is supported by a team of experienced engineers. Turnberry Projects has extensive experience with the design and evaluation of platinum projects and mines with past assignments including investigations ranging from metallurgical design to feasibility mine evaluation and project management.

Byron Stewart, P.Eng, of Wardrop Engineering Inc. is the Qualified Person for the NI 43-101 Technical Report which will support the statements made in the Feasibility Study relating to Mine Access Development and Footwall Development using mechanised mining methods. He has had extensive experience in consulting on mechanised mine development in various operations in three continents over 35 years.

Charles Muller, Gordon Cunningham, Tim Spindler and Byron Stewart have visited the property in 2008 and they have verified the data for their respective areas. Verification has occurred to a reasonable level in accordance with good engineering practice.

Conclusions - Qualified Person

The results of the Updated Feasibility Study are robust and the total project has a 24% IRR pre-tax with a Net Present Value at a 5% discount rate of US$981 million based on 3 year trailing metal prices. Using recent metal prices the project IRR is 16% pre-tax and its Net Present Value at a 5% discount rate is US$459 million.

The mining layout uses conventional proven approaches for mining and multiple accesses which will provide good flexibility and proven productivity at low risk. The metallurgical recovery and proposed concentrate should be attractive in the current tight market for platinum group metals. The project is highly competitive contrasted to many other projects and deeper platinum development alternatives.

Recommendations - Qualified Person

It is the recommendation of the Qualified Person and other qualified experts that the Updated Feasibility Study (RFS) be presented to the partners of the Joint Venture for consideration of the decision to mine, so that project implementation can commence.

Because of the release of the updated Mineral Resource statement, it is strongly recommended that the mine plan again be reviewed based on the changes to the mineable Resource with the updated model.

Table 11 - Salient Features for WBJV Project 1

SALIENT FEATURES - UPDATED FEASIBILITY STUDY REVIEW

RESOURCES

	Cut-off (cm g/t)	Mt	g/t	Average Mining Width (m)	Tons PGE (4E)	Moz
Measured	300	12.030	5.75	1.26	69.173	2.224
Indicated	300	28.278	5.51	1.24	155.812	5.010
Inferred	300	7.372	5.30	1.37	39.072	1.256

METAL PRICES & EXCHANGE RATE

		Case A 3-year Trailing Average	Case B Sept 2009	Case C High Road Scenario	Case D Low Road Scenario
Platinum	(US$/oz)	1,343	1,295	1,450	1,100
Paladium	(US$/oz)	322	296	600	400
Rhodium	(US$/oz)	4,951	1,650	2,500	2,000
Gold	(US$/oz)	807	997	843	500
Iridium	(US$/oz)	439	425	300	250
Ruthenium	(US$/oz)	349	90	300	250
Copper	(US$/lb)	2.92	2.84	2.27	2.04
Nickle	(US$/lb)	11.58	8.17	9.07	4.99
R/US$		8.00	8.00	8.00	8.00

FINANCIAL PARAMETERS AND INDICATORS

Basket Prices	ZAR/kg Case A	US$/oz Case A	ZAR/kg Case B	US$/oz Case B
Basket Price Merensky Reef	304,751	1,185	263,526	1,025
Basket Price UG2 Reef	368,571	1,433	274,639	1,068

Smelter and Refinery Payment Prices) (Case A	ZAR/kg	US$/oz
Reduction in Basket Price for MR	46,200	180
Reduction in Basket Price for UG2	55,875	217
Received Basket Price for MR	258,551	1,005
Received Basket Price for UG2	312,696	1,216

Smelter and Refinery Payment Prices) (Case B	ZAR/kg	US$/oz
Reduction in Basket Price for MR	39,951	155
Reduction in Basket Price for UG2	41,635	162
Received Basket Price for MR	223,575	869
Received Basket Price for UG2	233,004	906

Taxes		
Government Royalty Payment: PGM	(% of Rev.)	3.00%
Government Royalty Payment: Base Metals	(% of Rev.)	2.00%
Company Dividend Tax	(% of Pay.)	10.00%
Company Tax	(% of Profit)	28.00%

Capital	ZAR (Billion) million	US$ (Million) million
Capital: 3 Decline System (Total)	4760	595
Capital: 3 Decline System (Cost to First Production)	3431	429
Capital: 3 Decline System (Peak Funding)	3545	443

Working Costs (R/ton and US$/ton)	ZAR/tonne	US$/tonne
3 Decline System		
Total Working Cost for first 3 years	736.53	92.07
Total Working Cost for period after first 3 years	504.61	63.08
Total Working Cost for Life of Mine	539.59	67.45
On Mine Working Cost for Life of Mine	525.30	65.66

Working Costs (R/Kg and US$/oz)	ZAR/Kg	US$/oz
Total Cost to Metal Sold	161,648	628.48
Cost to Metal in Concentrate	135,335	526.17

EVALUATION (NET PRESENT VALUATION)

		5% DR	10% DR	12.5% DR
3 Decline System (Case A)	ZAR (Billion)	7.848	3.804	2.578
(Pre-tax)	US$ (Million)	981	475	322
	IRR	23.54%		
3 Decline System (Case B)	ZAR (Billion)	3.671	1.330	0.622
(Pre-tax)	US$ (Million)	459	166	78
	IRR	15.63%		
3 Decline System (Case C)	ZAR (Billion)	7.450	3.645	2.479
(Pre-tax)	US$ (Million)	931	456	310
	IRR	23.49%		
3 Decline System (Case D)	ZAR (Billion)	1.871	0.155	-0.344
(Pre-tax)	US$ (Million)	234	19	-43
	IRR	10.69%		

ITEM 4: INTRODUCTION

Item 4(a): Terms of reference

This report is compiled for Platinum Group 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. This report has been revised as per the request from regulators and refiled with Platinum Group and made available on SEDAR.

Item 4(b): Purpose of the Report

The intentions of the report are to

- inform investors and shareholders of the Updated Feasibility Study into the viability of the project
- make public, update and detail the mine designs for the project, based on the historical Mineral Resource

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 RSA. This data is derived from historical records for the area as well as information currently compiled by the operating company, which is PTM RSA. The PTM-generated information is under the control and care of Mr WJ Visser SACNSP 400279/04, who is an employee of PTM RSA and is not independent. The Anglo Platinum 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 (N1 43-101) was released on 30 October and 07 July 2009 by PTM RSA and has undertaken a due diligence with respect to the data.

Other information has been obtained from independent experts in the engineering field and where possible has been verified by the Qualified Person's for the project.

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

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

The Qualified Person's have visited the site on a number of occasions to view the drill core and the siting of the future mine.

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 RSA;
- geological and assay information supplied by PTM RSA and made available by AP;
- borehole analytical and survey data compiled by PTM RSA 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 the period 2005 to 2009 to review the core and the proposed mine site. The author's conclusion, based on diligence and investigation, is that the information is representative, accurate and forms a valid basis from which to complete the Updated Feasibility Study and make a recommendation to submit the same Updated Feasibility Study to the joint venture partners.

This report was prepared in the format of the Canadian National Instrument 43-101 Technical Report by the Qualified Person, Mr. GI Cunningham and others. The Qualified Person 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. In addition, the Qualified Person has also relied upon experts in the access development to the mine, the mining process, ventilation of the mine, rock engineering, metallurgical processing, all of whom have extensive and appropriate experience in evaluating the feasibility of mining projects. The Qualified Person has reported and made conclusions within this report with the sole purpose of providing information for PTM RSA's use subject to the terms and conditions of the contract between the Qualified Person and PTM RSA. The contract permits PTM RSA 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 Qualified Person. 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 Qualified Person bears no responsibility.

Specific areas of responsibility

The Qualified Person accepts overall responsibility for the entire report. The Qualified Person 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 RSA geological personnel in compiling this filing. In addition, the Qualified Person has relied upon information from other qualified persons, namely Tim Spindler and Byron Stewart for sections of the overall Updated Feasibility Study as per the appropriate certificates.

They have been supported by the following qualified specialists and firms:

Grinaker-LTA Mining	Construction & Mining Engineering
GRD Minproc	Process Engineering & Construction
Epoch Resources	Mine Residue & Environmental Engineering
Bluhm Burton Engineering Consulting	Mine Ventilation & Refrigeration Specialists
Dave Arnold & Associates	Rock Mechanics
Minxcon	Independent Consultants in Geology

ITEM 6: PROPERTY DESCRIPTION AND LOCATION

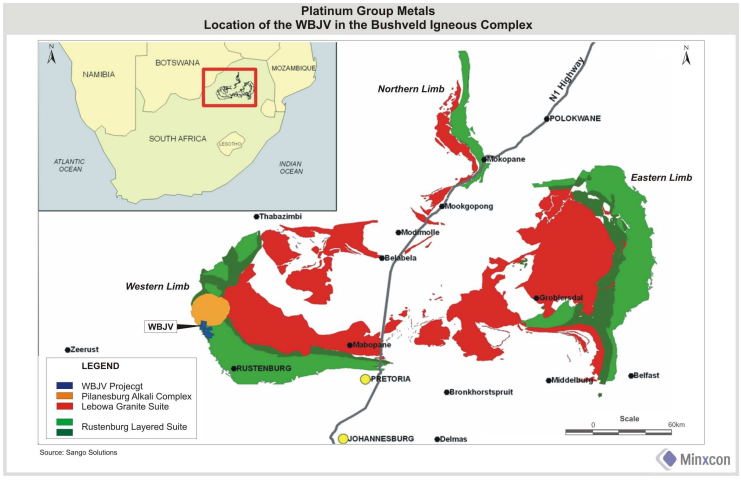
Item 6(a) and Item 6(b): Area and Location of project

The (“WBJV”) is owned 37% by Platinum Group Metals RSA (Pty) Ltd. - a wholly-owned subsidiary of Platinum Group Metals Ltd (Canada) - 37% by Rustenburg Platinum Mines Ltd, (“RPM”) - a subsidiary of Anglo Platinum - and 26% by Wesizwe. The joint venture is a notarial contract and managed by a committee representing all partners. The WBJV is divided into three distinct project areas, namely Projects 1, 2 and 3. PTM RSA is the operator of the joint venture. There is an agreement whereby this ownership will change to 74% PTM RSA and 26% Wesizwe and the transfer and registrations to make this agreement effective are in process.

This report focuses on Project 1 and 1A (collectively known as the Project Area herein), which are located on the southern extent of the WBJV, which is underlain by the Merensky Reef and UG2 Chromitite seam. The Merensky Reef subcrop forms the western boundary of the Project Area.

The WBJV property is located on the south-western limb of the BIC 110km west-northwest of Pretoria and 120km from Johannesburg (Figure 1).

Figure 1 - Location of the WBJV in the BIC

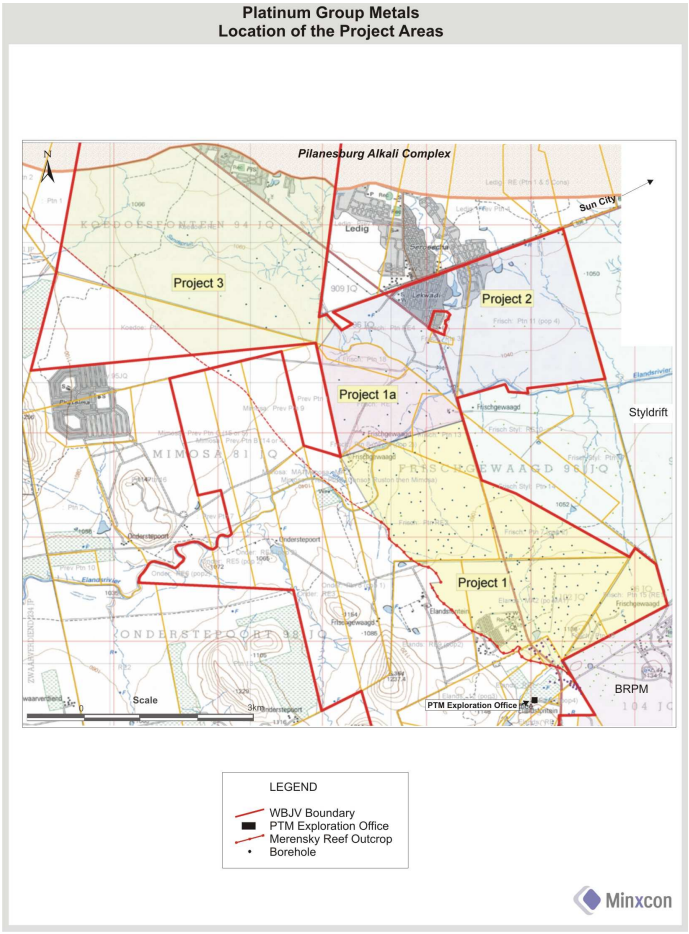


The Resources of the WBJV Project Area 1 and 1a are located approximately 11km along strike from the active Merensky Reef mining face at the operating BRPM which completed opencast mining on the UG2 Reef within 100m of the WBJV property boundary.

Project Area 1 and 1a covers an area of 10.87km ^{2} or 1,087ha in extent.

Specifically, the Project Area consist of a section of Portion (Ptn) 18, the Remaining Extent (Re), Ptn 13, Ptn 8, Re of Ptn 2, Ptn 7, Ptn 15 and Ptn 16 of the farm Frischgewaagd 96JQ, sections of Ptn 2, Ptn 9 and Ptn 12 of the farm Elandsfontein 102JQ and a small section of the Re of the farm Mimosa 81JQ (Figure 2).

Figure 2 - Locality Plan of the Project Area in the WBJV



The WBJV property is partly situated in a layered igneous complex known as the 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 (PGE's), chrome and vanadium.

The potential economic horizons in the WBJV Project Area 1 and 1A are the Merensky Reef and UG2 Chromitite seam situated in the Critical Zone ("CZ") of the Rustenburg Layered Suite ("RLS") of the BIC; these horizons are known for their continuity. The Merensky Reef and UG2 Chromitite seam are mined at the BRPM adjoining the WBJV property as well as on other contiguous platinum-mine properties. In general, the layered package dips at less than 20 degrees and local variations in the reef attitude have been modelled.

The Merensky Reef and UG2 Chromitite seam, in the Project Area, dip between 4 and 42 degrees, with an average dip of 14 degrees.

Drilling, in the form of diamond drilling, has been carried out over the Project Area and to-date 181 boreholes have been drilled for the purposes of understanding the geology, structure and metallurgy of the orebody in the Project Area. PTM RSA have established a site office to the south of the Project Area, and all core is stored in the core yard on site. All logging and sampling of the core is undertaken at the site office core yard and the samples have been sent to Genanalysis (Perth), ALS Chemex (South Africa) and currently samples are sent to Set Point Laboratories (South Africa). A total of 29,303 samples have been assayed and utilised in the estimation of the Mineral Resources over the Project Area.

For the purpose of the mine plan presented herein. Measured Mineral Resource total 2.224 million ounces (Moz), Indicated Mineral Resources total 5.010Moz and Inferred Mineral Resources total 1.256 Moz of 4E (platinum, palladium, rhodium and gold) for Project Area 1 and 1a. 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 WBJV has 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 below for cross-referencing to the licence specifications. The licences over the WBJV area are as follows:-

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

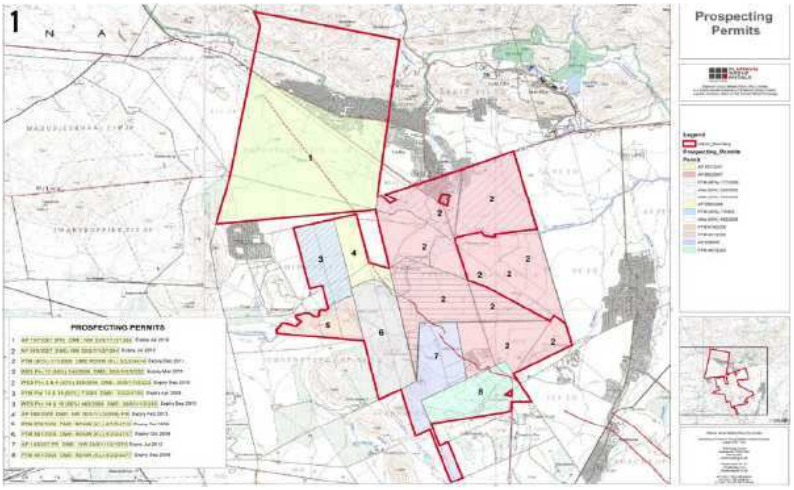
Applications have been made in a timely fashion for conversion to the new Mineral and Petroleum Resources Development Act, 2002 (“MPRDA”). The Prospecting Rights (“PR”) are all held in the North West Province Region of the DME and are held for PGM’s Nickel, Chrome and Gold. The following table details the aspects of the PR’s:-

Table 12 : Legal Aspects and Tenure of the WBJV Area

Area	Farm Name	Ptn No.	Area (ha)	Old Order PR	New Order PR	Expiry Date
Elandsfontein (PTM RSA)	Elandsfontein 102JQ	12 (a Ptn of Ptn 3)	213.4714	PP269/2002 reference RDNW (KL) 5/2/2/4477 **(Expired)**	Notarially executed under protocol no. 467/2005.	14 June 2012
		14	83.4968			
		RE of Ptn 1	67.6675			
Elandsfontein (RPM)	Elandsfontein 102JQ	8 (a Ptn of Ptn 1)	35.3705	PP50/1996 reference RDNW (KL) 5/2/2/2305) & PP73/2002 reference RDNW (KL) 5/2/2/4361. **(Expired)**	A conversion to a new-order prospecting right was approved.	3 July 2012
		RE9	403.9876			
		Mineral Area 2	-			
Onderstepoort (PTM RSA) Portions 4, 5 and 6	Onderstepoort 98JQ	4 (a Ptn of Ptn 2)	79.8273	PP48/2004 (reference no. RDNW (KL) 5/2/24716) **(Expired)**	Notarially executed under protocol no. 879/2006.	22 July 2011
		5 (a Ptn of Ptn 2)	51.7124			
		6 (a Ptn of Ptn 2)	63.6567			
Onderstepoort (PTM RSA) 3 and 8	Onderstepoort 98JQ	Re of Ptn 3	274.3291	PP26/2004 reference RDNW (KL) 5/2/2/4717) **(Expired)**	Notarially executed under protocol no. 881/2006.	22 July 2011
		8 (a Ptn of Ptn 1)	177.8467			
Onderstepoort (PTM RSA) 14 and 15	Mimosa 81JQ	A Ptn of Re	245.2880	K46/1971 RM **(Expired)**	Notarially executed under protocol no. 7.	15 December 2011
	Mimosa 81JQ	A Ptn of Re	183.6175			
Onderstepoort (RPM)	Mimosa 81JQ	9 (a Ptn of Ptn 3	127.2794	Unknown	Notarially Executed under Protocol No.79/2008	14 February 2013
		Mineral Area 1 of Ruston 97JQ	29.0101			
		Mineral Area 2 of Ruston 97JQ	38.6147			
Frischgewaagd (PTM RSA)	Frischgewaagd 96JQ	RE 2	640.7	Unknown	Covering 23/24 th share of the undivided Mineral Rights. Notarially executed under protocol no.117.	14 December 2011
		7 (a Ptn of Ptn 6)				
		8 (a Ptn of Ptn6)				
Frischgewaagd (RPM)	Frischgewaagd 96JQ	Re	1622,5627	PP294/2002 reference RDNW (KL) 5/2/2/4414	Covering the remaining undivided Mineral Rights.	3 July 2012
		2				
		3				
		4				
		7				
		8				
		11				
		13				
		15				
		16				
		17				
		18				
Koedoesfontein (RPM)	Koedoesfontein 94JQ PGE's, Gold, Silver, Copper and Nickel	All	1702.8204	PP70/2002 (reference 5/2/2/4311)	Notarially Executed under Protocol 14/2007	3 July 2012
Koedoesfontein (RPM)	Koedoesfontein 94JQ Chrome and Cobalt	All	1702.8204	PP/2007/04/25/005 NW 30/5/1/1/2/1682 PR	Notarially Executed Under Protocol 590/2008	14 February 2013

The locations of the PRs are illustrated graphically in Figure 3.

Figure 3 : WBJV Prospecting Right Holders



In a news release dated 2 September 2008, and subsequently updated on 9 December 2008, PTM RSA announced that it has agreed to a term sheet with Anglo Platinum and Wesizwe to consolidate and rationalize the WBJV. Under the terms PTM RSA will have the right to acquire effective ownership of 74% of WBJV Projects 1 and 3 and Wesizwe will acquire 100% of Project 2 and 26% of Projects 1 and 3. The transactions will become effective upon fulfilment of certain conditions precedent and regulatory approvals including the approval of the DME, Republic of South Africa, for transfer of Mineral Rights in accordance with Section 11 of the MPRDA.

The following points summarise the transaction:-

1. The various joint venture partners who participated in the WBJV each contributed certain PRs to the WBJV, which PRs were registered in their individual names;
2. In the process of unbundling the WBJV, the various PR's were allocated to either of Projects 1, 2 or 3. The PRs for Projects 1 and 3 must be transferred from the name of the current holders into the name of Maseve Investments 11 (Proprietary) Limited (Maseve), at this juncture, a wholly owned subsidiary of PTM RSA;

3. In terms of Section 11 of the MPRDA, the consent of the Minister of Mineral Resources (South Africa) is required before PRs can be transferred from one holder to another. This consent ensures that the transferee is financially and technically capable of executing the prospecting work programme and rehabilitation obligations, and also ensures that the transferee has the requisite level of black economic empowerment; and
4. To obtain Section 11 consent, the transferor and transferee must apply in writing to the Minister, giving details of the transferee's financial and technical competence, together with details of its black economic empowerment credentials. If the transferee meets these requirements, the Minister must issue consent. The matter is not discretionary.

The Section 11 application for transfer of Mineral Rights is currently being processed by the DME, South Africa.

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

Regarding Elandsfontein (PTM RSA), 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 RSA to the rights to the minerals as well as the freehold. PTM RSA has purchased the surface rights to the property. The surface rights to Portions Re 1, 12 and Re 14 measure 364.6357 Ha. Option agreements in respect of Onderstepoort (PTM RSA) 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 was bought out by way of a settlement agreement and a new order prospecting right covers this area. The remainder of the WBJV property is covered by Anglo Platinum prospecting rights contributed to the Joint Venture.

WBJV Terms

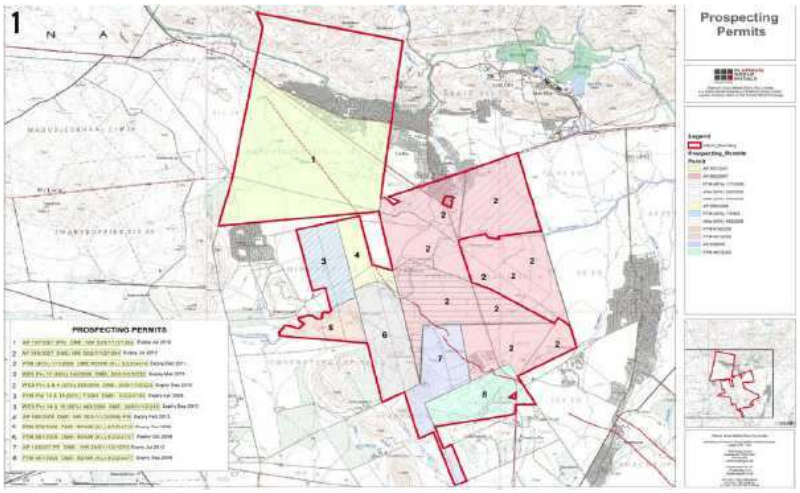
Original Joint Venture Terms:

The detailed terms of the WBJV - relating to Elandsfontein (PTM RSA), Elandsfontein (RPM), Onderstepoort (PTM RSA), Onderstepoort (RPM), Frischgewaagd (PTM RSA), 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 MPRDA. PTM RSA and RPM will each own an initial 37% working interest in the farms and mineral rights contributed to the joint venture, while Wesizwe will own an initial 26% working interest. Wesizwe 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).

Restructured JV Agreement

In a news release dated 2 September 2008, and subsequently updated on 9 December 2008, PTM RSA announced that it had signed an agreement with AP and Wesizwe to consolidate and rationalize the WBJV. Under the terms PTM will have the right to acquire effective ownership of 74% of WBJV Projects 1 and 3 and Wesizwe will acquire 100% of Project 2 and 26% of Projects 1 and 3. The transactions will become effective upon fulfilment of certain conditions precedent and regulatory approvals including the approval of the DME, Republic of South Africa, for transfer of Mineral Rights in accordance with Section 11 of the MPRDA. These transfers are in process.

Figure 4 - Prospecting License Area



Ore and Concentrate Treatment Agreements

There are draft pro-forma ore and concentrate treatment agreements in place, which form part of the WBJV documentation. The Updated Feasibility Study project team have assumed that certain terms and conditions will be negotiated between the WBJV project operators and the Anglo Platinum smelter operator and have maintained the terms and conditions detailed in the Feasibility Study.

Environmental Management Programme

The Environmental Impact Assessment (EIA) and the Environmental Management Programme Report (EMPR) have been completed to a Scoping level and further work is being completed for submission with the Mining Rights Application (MRA), which is awaiting the formation of 'Newco'.

Social and Labour Plan

The draft Social and Labour Plan (SLP), which is to be submitted to the Department of Mineral and Energy (DME), has been developed and will be submitted with the MRA.

Item 6(e): Survey

Elandsfontein (PTM RSA) 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 RSA) 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 RSA), 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 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 worlds 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 RSA 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 RSA)* 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 RSA 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 RSA)* area that was contributed to the WBJV by PTM RSA, 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 RSA 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 total 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 north-easterly 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

Oryx Environmental Consulting have undertaken a groundwater investigation as part of the formal EIA process. 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-HC0 $_3$. The groundwater chemical data indicates that magnesium, calcium and 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 project 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 are mainly agriculture and grazing. The main impact on the air quality is vehicle emissions. Concerning the regional air quality, it is heavily impacted by S04 emissions from smelter operations in the area. In addition, the potential release of fine dust from tailings dam and roads is a concern for the air quality on a cumulative basis.

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 North 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:

Table 13 - Bird Species (Red Data) that may occur on project area

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.

Table 14 - Snakes (Red Data) that potentially occur in project area

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.

Table 15 - Mammals that potentially occur in project area

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 sandier 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).

Table 16 - Vegetitation species evident in project area

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): Environmental Process

Public consultation

A comprehensive public consultation programme was carried out from January to April 2008 during the scoping phase, including meetings with:

- Affected landowners within the study area.
- Landowners adjacent to the study area.
- Bakubung Tribal Authority.
- Bafokeng Tribal Authority.
- Representatives of the mining, tourism and conservation industries.
- Labour unions.
- Environmental organisations, community-based organisations and NGOs.
- Robega Village Council.
- Sun City Hotels and Resort.
- Community forum representatives from Rasimone, Robega, Chaneng and Mafenya.
- The general public.

From this process, a comprehensive database of interested and affected parties has been developed.

Summary of issues raised by interested and affected parties

EIA process

- The cumulative impacts of the mine within the context of the new mining activities in the region should be assessed.

Soils, land capability and land use

- Consideration of the use of the land after mining.
- Consideration of what appropriate rehabilitation will be undertaken and clear assignment of the responsibilities for it.
- Consideration of backfilling of mine tailings into the mine.
- Consideration of the use of mine waste rock to clad the tailings dam in an effort to reduce its visual impact on the landscape.

Fauna, flora and ecosystems

- Impact on habitat and populations of Red Data species.
- Impact on wetlands and riparian zones.

Surface and groundwater

- Source of water supply for the mine.
- Impact of mine water pollution on the environment.
- Pollution of groundwater, particularly associated with the tailings dam.
- Cumulative impact of mining on water quality.
- An indication of amount of water used per ton of ore mined and gram of platinum produced.
- Options for water saving, particularly in the disposal of tailings, through the use of technologies such as paste.
- Options for improved water efficiency through the re-use of water or use of alternative water sources.

Air quality

- The impact of dust on the health and quality of life of adjacent communities.
- Cumulative impact of mining on air quality in the region
- Options for alternative disposal of tailings in an effort to reduce dust impacts, including the use of paste technology.
- Options to mitigate the impacts of dust in the disposal of tailings, including the irrigation of tailings when wind speeds exceed defined limits.

Blasting and vibration

- Vibration impacts resulting in cracking and damage to houses and consequent devaluation.

Noise

- The potential for mining noise to disrupt adjacent communities.

Heritage

- Impacts on archaeological sites and artefacts.
- Impacts on graves.

Visual

- Impact of the mining operation on tourist traffic on the R565 and R556 roads.
- Impact of the mining operation on surrounding tourism operations at the Sundown Ranch, the Pilanesberg National Park and Sun City.

Roads, transport and infrastructure

- Impacts on the R565, which is likely to be the primary access road to the mine.
- Requirements for the upgrading of the R565 in conjunction with the operations and programmes of mines in the region, in particular BRPM.
- Cumulative traffic impacts of the mine’s operation on the road network.

Electricity

- Options for the supply of electricity, given the problems of supply from ESKOM.
- Impacts of the use of alternatives such as diesel and logistical consequences of such options.
- An indication of amount of power used per ton of ore mined and gram of platinum produced.
- Options to reduce power demand and consideration of renewable energy sources.

Socio-economic

- How the local communities adjacent to the proposed mine will benefit from it.
- How the project will invest in the local communities adjacent to the proposed mine.
- What training and other opportunities the mine will provide to the local communities so that they may benefit from it.
- How the WBJV will ensure that local small businesses are able to provide services to the proposed mine.
- In the past similar projects have made promises of jobs and benefits to the community and these have not been fulfilled.
- It is very difficult for local communities to communicate and interact with the mines in the region.
- Where the mine's workers will be housed and how the WBJV will ensure that the mine does not result in the creation of informal settlements.
- The influx of people from elsewhere looking for work puts pressure on infrastructure and social services.
- The influx of people as a result of the mine will be a cause of crime and a risk to the security of adjacent landowners and communities.

Plan of study for EIA and EMP

The plan of study for the EIA and EMP sets out the rationale for the different levels of study for the various environmental components, based on the issues raised by interested and affected parties, the expected severity of impacts and the level of confidence required in their prediction. The level of information required to develop adequate, practical management and mitigation measures was also a consideration in determining the terms of reference of studies.

Scope of studies

Information on the following environmental aspects will be obtained from existing information sources combined with site visits to confirm findings:

- Climate
- Topography
- Land use
- Wetlands

The following aspects will require collation of existing information, field surveys, sampling and mapping with impact assessment scenario's being undertaken using geographic information systems and qualitative forms of impact analysis:

- Soils
- Land capability
- Fauna
- Vegetation
- Noise and vibration
- Archaeology and heritage
- Socio-economic

The following specialist studies will require detailed study with quantitative modelling of various scenarios to predict impacts for planning and assessment purposes:

- Surface water
- Groundwater
- Air quality
- Traffic
- Visual aspects

Both the MPRDA and NEMA require an assessment of cumulative impacts. The following cumulative impacts for the mine will be specifically addressed in the EIA:

- Air quality.
- Visual
- Socio-economic
- Traffic

Environmental Conclusion

The draft Scoping Report sets out the proposed scope of the EIA and EMP that will be prepared for the proposed WBJV Project 1 Platinum Mine. This includes the range of alternatives that will be evaluated for various aspects of the project, the key environmental impacts and issues that need to be addressed, the studies that will be undertaken, the terms of reference of the specialist studies and the qualifications and experience of the study team.

The updated Scoping Report will be available for public review approximately March 2010 at various public venues and on the internet. Registered interested and affected parties will be notified of the availability of the updated Scoping Report.

Following the public review period, the draft updated Scoping Report will be corrected on the basis of comments received, finalised and submitted to the relevant authorities for their comment.

Item 7(c): 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 south-western 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. 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(d): 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 area, ambient or background noise levels are rather low.

Item 7(e): 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 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 temperatures 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.

Table 17 - Average Rainfall in Project Area

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.

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:

Table 18 - Average rate of Evaporation in Project Area

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 typically occurs in the form of showers and thunderstorms with which strong gusty winds are associated.

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

Item 7(f): 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 (400kV) 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 Anglo Platinum 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. In addition, some 575ha, comprising Van Vuuren (Re: 10 Frischgewaagd - 222ha) and J C Grobbler (Re: 2 (Portion 19) Frischgewaagd - 353ha) has also been purchased and further surface rights will be required and purchased

ITEM 8: HISTORY

Item 8(a): Prior ownership

Elandsfontein (PTM RSA) , *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 Anglo Platinum 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. Anglo Platinum 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 RSA 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 south-eastern to north-western 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

The following table summarises the historically estimated Mineral Resources on Project Area 1:-

Table 19 - Summary of Historical Mineral Resources (Project 1)

Effective Date	Date Sedar Filed	Measured Resources	Indicated Resources	Inferred Resources
12 December 2005	13 January 2006		6.92Mt grading 5.89g/t (1.31 Moz)	20.28Mt grading 5.98g/t (3.90 Moz)
2 March 2006	13 April 2006		20.45Mt grading 3.91g/t (2.57 Moz)	30.99Mt grading 5.16g/t (5.14 Moz)
21 September 2006	6 November 2006	4.453Mt grading 5.20g/t (0.744 Moz)	40.284Mt grading 4.28g/t (5.546 Moz)	15.051Mt grading 4.15g/t (2.006 Moz)
10 January 2007	30 January 2007	4.453Mt grading 5.20g/t (0.744 Moz)	40.926Mt grading 4.31g/t (5.676 Moz)	14.363Mt grading 4.03g/t (1.863 Moz)
07 September 2007	30 January 2007	13.47Mt grading 5.29g/t (2.289 Moz)	30.76Mt grading 5.077g/t (5.02 Moz)	7.520Mt grading 5.23g/t (1.266 Moz)
07 July 2008	21 August 2008	12.03 Mt grading 5.07g/t (2.24 Moz)	28.28 Mt grading 5.51 g/t (5.01 Moz)	7.37 Mt grading 5.3 g/t (1.2.56 Moz)

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): Technical Reports issued for Development of the Project

An Independent Preliminary Assessment was released in August 2005 recommending that a Pre-Feasibility Study be conducted to evaluate the economic potential of the WBJV Project 1 area.

In November 2006, a Technical Report detailing the Pre-Feasibility Study was released recommending that a full Feasibility Study be commissioned to evaluate the further economic potential of the WBJV Project 1 area.

In July 2008, a Technical Report detailing the full Feasibility Study was completed and released recommending that the project develop to implementation, subject to project financing and the approval of the Mining Right Application and other elated legal issues.

In June 2009, a Technical Report detailing the Revised Feasibility Study was completed but with only minor changes from the full Feasibility Study dealing with SEC issues.

Item 8(e): Production from the property

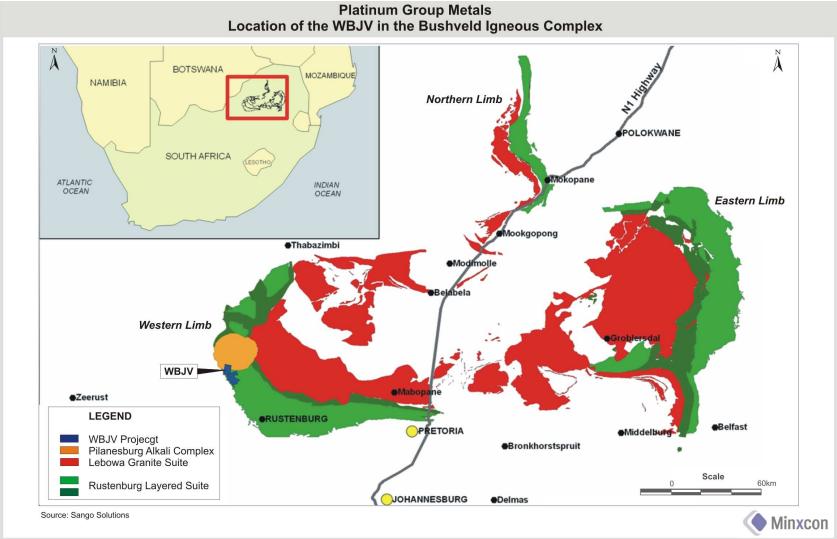
There has been no previous production from any of the WBJV properties.

ITEM 9: GEOLOGICAL SETTING

Item 9(a): 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 PGE's, 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 south-eastern/Bethal limbs.

Figure 5 - Location of the WBJV in relation to the Bushveld Igneous Complex



Item 9(b): Local geology

The WBJV is underlain by the lower portion of the RLS, the Critical Zone and the lower portion of the Main Zone. The ultramafic Lower Critical Zone and the Mafic Upper Critical Zone and the Main Zone weather to dark, black clays with very little topography. The underlying Transvaal Supergroup comprises shale and quartzite of the Magaliesberg Formation, which creates a more undulating topography. Gravity, magnetic, LANDSAT, aerial photography and geochemistry have been used to map out lithological units.

The MR outcrops, 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 between 4° and 42° with an average of 14° in the Project 1 and 1A areas. The top 32m of rock formation below the soil column is characterized by a highly weathered rock profile (regolith) consisting mostly of gabbro within the Main Zone. Thicknesses of this profile increase near intrusive dykes traversing the area.

Item 9(c): Stratigraphy

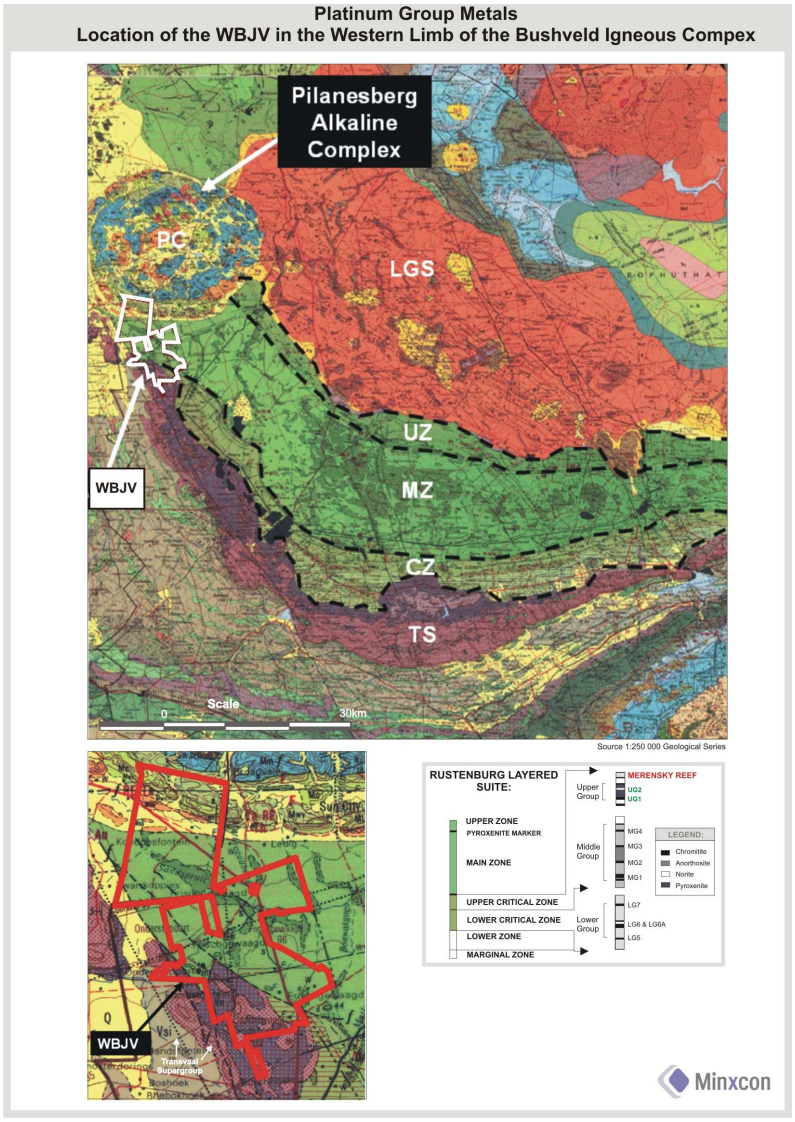
The RLS intruded into the rocks of the Transvaal Supergroup ("TS"), largely along an unconformity between the Magaliesberg quartzite of the Pretoria Group and the overlying Rooiberg felsites, which is a dominantly felsic volcanic formation. The mafic rocks of the RLS are subdivided into the following 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 (“CZ”) - 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 PGE’s. The two uppermost cycles of the Critical Zone include the Merensky and Bastard cycles. The Merensky Reef 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 (“MZ”)** - 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 (“UZ”)** - 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.

The location of Project Area 1 and 1A on the BIC, as well as a summary of the stratigraphic column, is illustrated below:-

Figure 6 - Location of the WBJV in the Western Limb of the BIC



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 (Schürmann).

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.

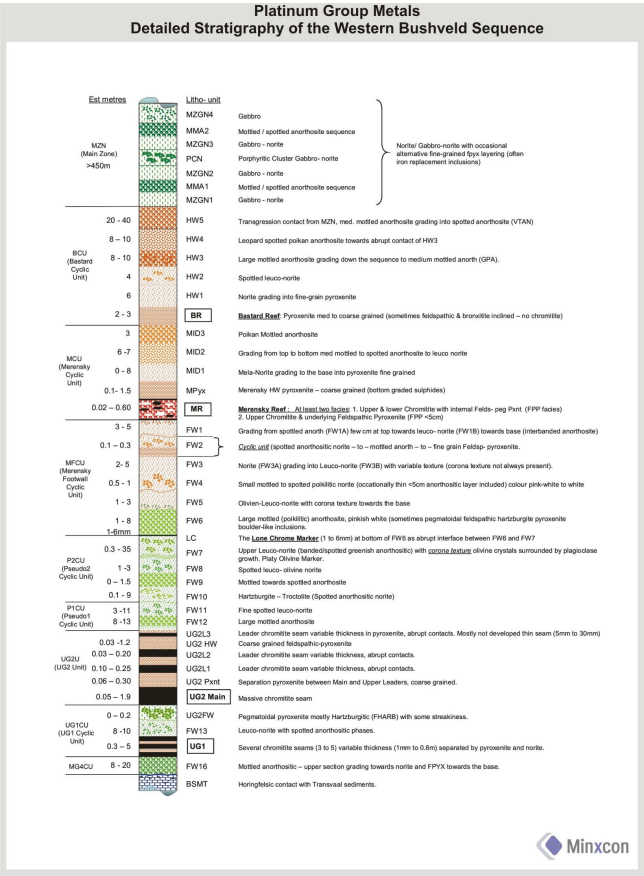
Merensky Footwall Unit

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 pegmatite 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.

Figure 7 - Detailed Stratigraphy of the Western Bushveld Sequence



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 1m 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

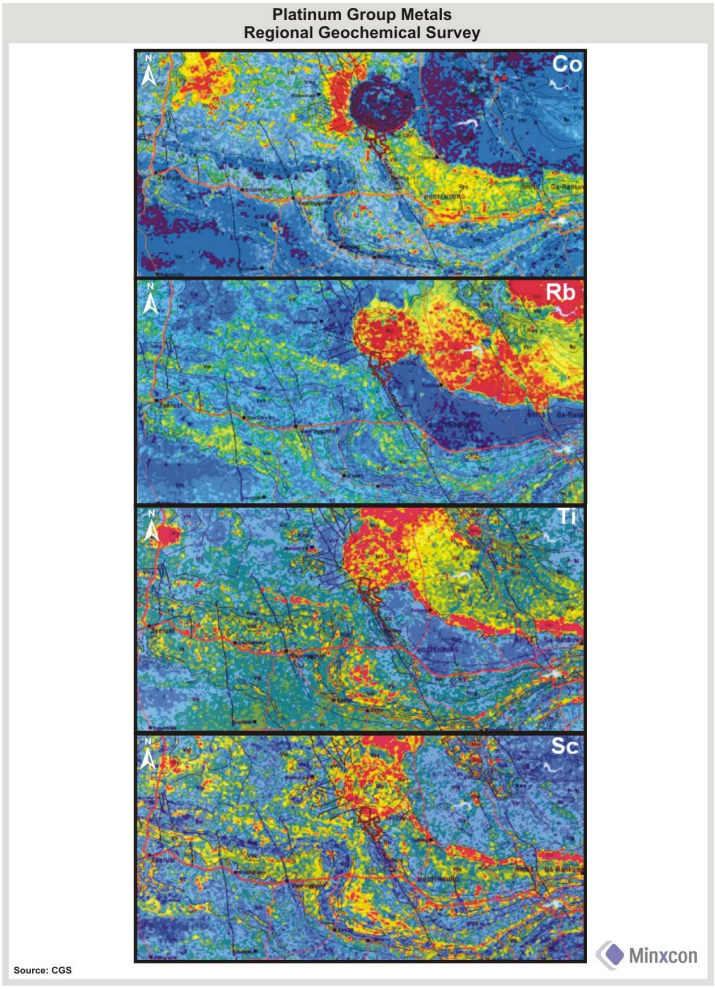
The UG1 chromitite layer is approximately 1m thick and forms the base of this unit. It is underlain by the 10m-thick FW14 mottled anorthosite. The UG1 chromitite layer bifurcates and forms two or more layers within the footwall mottled anorthosite, while lenses of anorthosite also occur within the chromitite 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 9(d): Regional Geochemistry Survey

The Council for Geoscience ("CGS") undertook the task of producing regional geochemical maps of South Africa to complement the existing geological information. The aim of the survey was to create a geochemical database and to determine the baselines for different lithological units. Sampling on Sheet 2526 Rustenburg was carried out in the winter months of 1991 and 1992. The sampling density was 1 sample per 1km ^{2}. Samples were analysed for 23 elements on a simultaneous X-Ray Fluorescence Spectrometer. Twenty three geochemical element-distribution contour maps were compiled (Wilhelm and Van Rooyen, 2001). The contour maps for the following elements: Sc, TiO_{2}, V, Cr, MnO, FeO_{3}, Co, Ni, Cu, Zn, As, Rb, Sr, Y, Zr, Nb, Sn, Sb, Bu, W, Pb, Th and U were examined. The BIC is readily subdivided into the mafic / ultramafic RLS and the felsic Lebowa Granite Suite ("LGS"). The Pilanesberg Complex north of the Project Area can also be clearly delineated. The RLS has elevated Co concentrations, particularly in the Lower Zone and Lower Critical Zone. The Upper Zone is clearly defined by the high concentrations of TiO_{2}. The RLS is depleted in Rb which the LGS is elevated in this element.

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.

Figure 8 - Regional Geochemical Survey



Item 9(e): Regional Geophysical Surveys

The regional gravity and magnetic maps of the BIC highlight the broad structure of the intrusion. The gravity anomalies delineating the BIC peak at approximately the top of the RLS, at the contact between the Upper Zone and the LGS. This defines the area where the mafic rocks are at their thickest and uncovered. The magnetics can be used to delineate the iron rich (magnetic layers) in the Upper Zone.

Item 9(f): Regional Structure

Floor rocks in the south-western 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° and dipping very steeply south-southeast or north-northwest, are related to the Rustenburg fault system. These structures were reactivated during the intrusion of the Pilanesberg Alkaline Complex. Dykes associated with this Complex intruded along these faults and joints.

Major structures, which occur within the WBJV area, include the Caldera and Elands faults and Chaneng Dyke and a major north-south trending feature, which can be observed across the entire Pilanesberg Complex.

These east-west trending structures dip steeply (between 80° and 90°). The magnetics indicate that the Chaneng Dyke dips steeply to the north. This is consistent with similar structures intersected underground on the neighbouring Bafokeng Rasimone Platinum Mine, which all dip steeply northward.

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 outcrop position.
There is also a subcrop of the Critical Zone against the main zone rocks.

Figure 9 - Gravity and Total Field Magnetics across the BIC and Surrounds

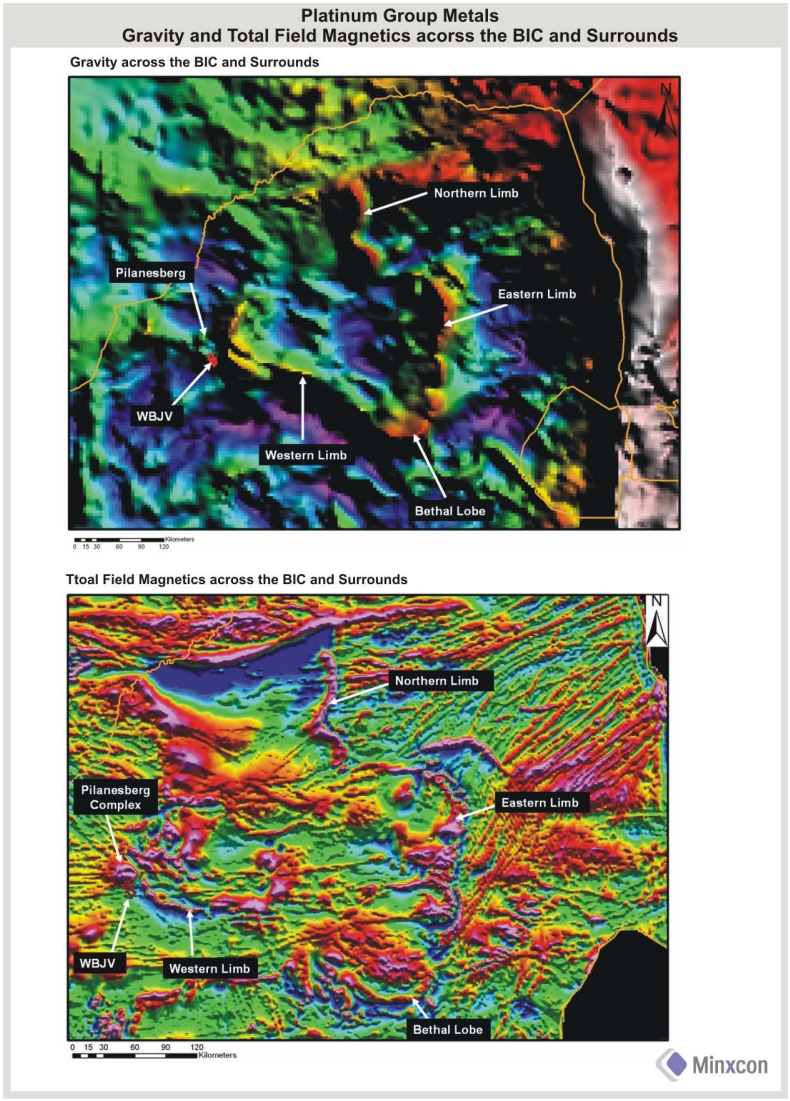
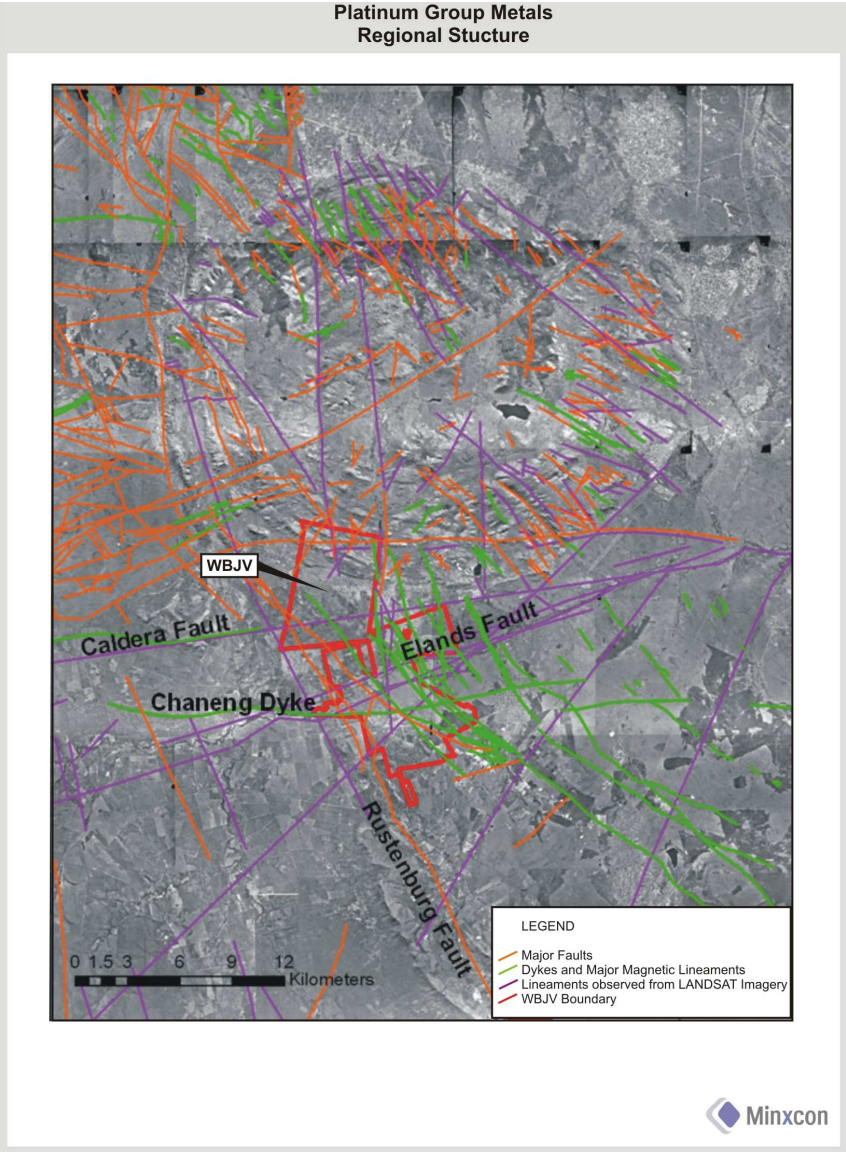


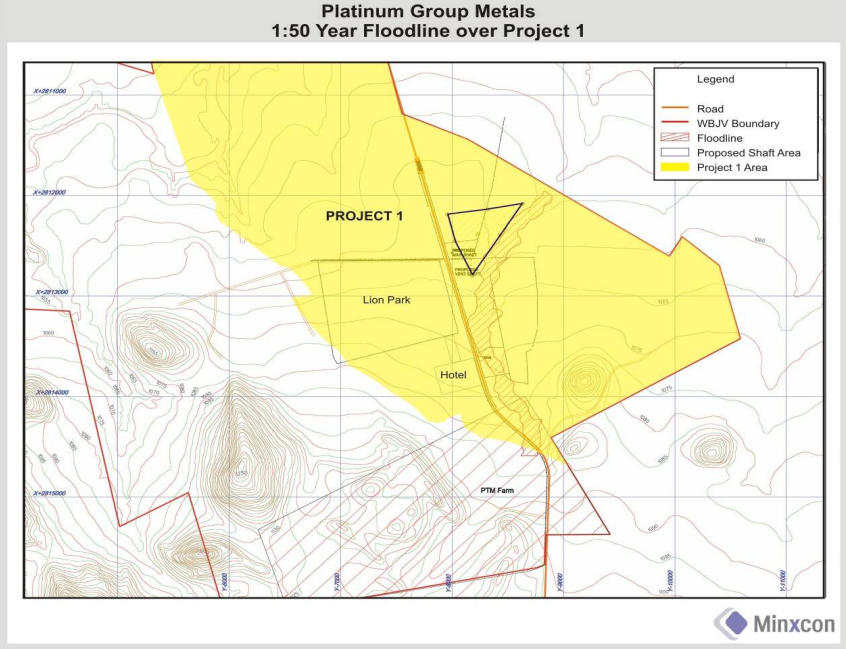
Figure 10 - Regional Structural Data



Item 9(g): Hydrogeology

The area is considered to be semi arid, with an annual rainfall of 520mm, which is below the average rainfall for South Africa. The rainy season falls between the summer months of October through to April and highest rainfall occurring during December and January. The highest rainfall ever recorded in the area was 65mm over a 24- hour period. Analysis of the meteorological data showed that the prevailing winds are from the east and south-east and wind speed average 2.5m/s. GCS (Pty) Ltd conducted a floodline determination for the proposed Elandsfontein Mining Development. Floodline calculations were conducted on all major surface water Resources that fall within the proposed development area. Floodlines for the 1:50 and 1:100 year floods were determined. The 100-meter lines from the centre of a watercourse have also been indicated on maps. No site visit was conducted and all calculations were based on desktop analyses.

Figure 11 - 1:50 Year Floodline over Project 1



The groundwater level elevation has a good correlation with surface elevation. The general groundwater flow direction is toward the north and northeast in the direction of the surface runoff channels. Groundwater within the study area generally occurs in secondary aquifers created by weathered and fractured geological processes. The aquifer is unconfined/semi-confined. The aquifer beneath the site would classify as a minor aquifer. The average depth of the groundwater table in the study area is 26m below surface. The depth of the water level varies between 3m to >60m and the associated yields are depicted in Figure 13 - Water Depth and Borehole Yield.

The deep water level is mainly attributed to the large-scale abstraction for irrigation. Most boreholes in the study area have a depth of 40-80m. The available borehole information from previous investigations in the region shows that adjacent to the study area water strikes occur at 23m depth.

Figure 12 - Correlation between Groundwater Level and Surface Elevation

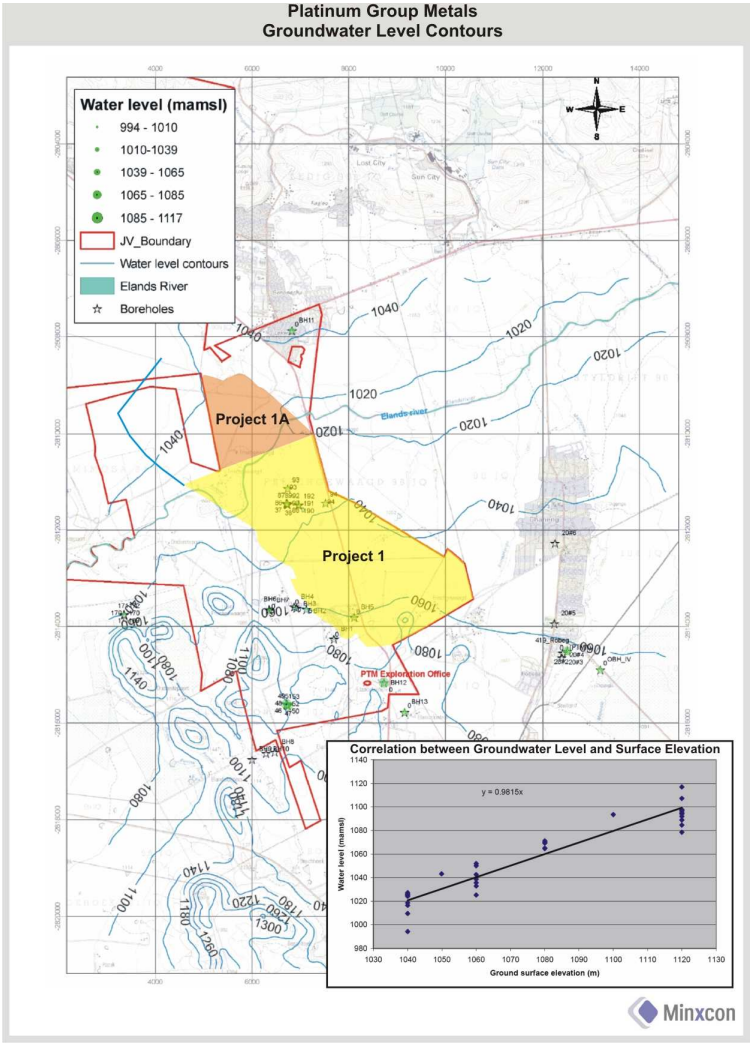
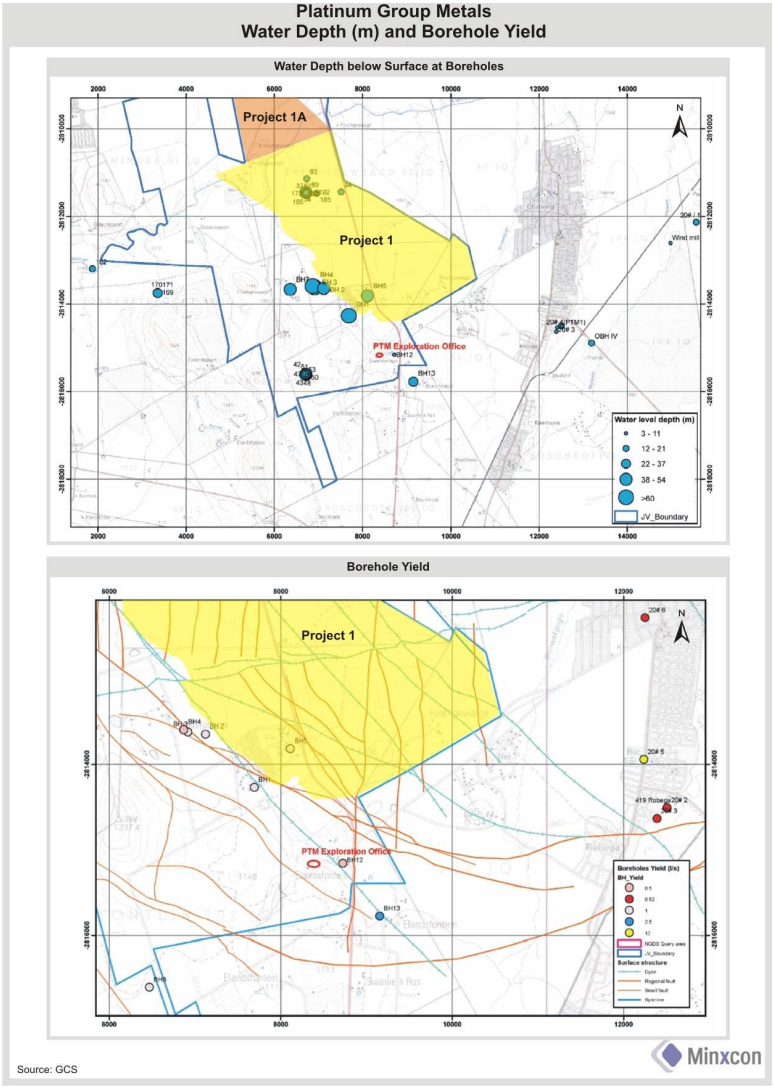


Figure 13 - Water Depth and Borehole Yield



The chemical analyses of water samples taken during the pre-Feasibility Study show that the dominant water type is Mg-Ca-HCO_3. The groundwater chemical data indicates that generally magnesium, calcium and TDS are present in concentrations that exceed the DWAF Domestic Water Quality Guidelines ("SAWQ"), as illustrated in the table below. This is attributed to the geology of the study area. Two boreholes located within villages show elevated nitrate concentrations, which are attributed to human activities.

Table 20 - Groundwater Chemical Results and Comparison with SAWQ Guidelines

Parameter (mg/l)	SAWQ Target Values	BH2	BH 8	BH 11	BH13	PTM1
Total Dissolved Solids	0 - 450	582	254	528	438	1032
Nitrate (NO3)	0 - 6	35	0.9	10.3	18.2	105
Chlorides (Cl)	0 - 100	52	6.4	103	49	121
Total Alkalinity (CaCO3)	NS	412	232	328	296	564
Fluoride (F)	0 - 1	<0.1	<0.1	1.3	<0.1	<0.1
Sulphate (SO4)	0 - 200	19.7	<0.1	21	11.2	74
Total Hardness (CaCO3)	0 - 100	499	223	349	338	791
Calcium (Ca)	0 - 32	78	40	64	48	101
Magnesium (Mg)	0 - 30	74	30	46	53	131
Sodium(Na)	0 - 100	10.2	8.5	61	20	48
Iron (Fe)	0 - 0.1	0.06	0.06	7.1	0.06	0.05
Manganese (Mn)	0 - 0.05	0.003	0.006	0.62	0.001	0.001
Conductivity (25°C in ms/m)	0 - 70	89.6	41.3	87	66.2	145
pH Value at 25°C	6 - 9	7.4	7.8	7.6	7.6	7.6
Zinc (Zn)	0 - 3	0.003	0.21	11.5	0.05	0.02
Lead (Pb)	0 - 0.01	<0.01	<0.01	0.06	<0.01	<0.01
Aluminium (Al)	0 - 0.15	0.04	0.05	0.09	0.06	0.04

Notes: All parameters are in mg/t, unless otherwise stated
NS = Not Specified
Red = Exceeds SAWQ Guideline
Source: GCS

Groundwater usage in the study area is mainly for domestic purposes and livestock watering as well as irrigation. The total groundwater abstraction from the study area is 30,000l/day. It is expected that dewatering of the aquifers surrounding the proposed mining area will occur due to dewatering of the mine workings.

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 TSF and the waste rock dump could occur.

Surrounding farmers and residents largely make use of groundwater. The potential for diminishing groundwater Resources could impact on the activities of other users.

ITEM 10: DEPOSIT TYPE

Item 10(a): Local Geological Setting

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. 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 vibrosis seismic data for the south-western 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 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.

The MR is a well developed seam along the central part and towards the north eastern boundary of the Project Area. Islands of thin reef 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 deeper portions.

The UG2 reef is well developed towards the northeast of the 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 ~2m.

The thickness of the sequences between the UG2 and MR in the Project 1 and 1A areas increases from ~10m to 80m in a southwest-northeast direction. A similar situation exists in the north of the Project Area but with the thickness between the reefs ranging from 6m to 25m at depths of 200m below surface. In general, the thickness between the reefs appears to increase in a north-easterly direction, sub-parallel to the strike of the BIC layered lithologies.

Item 10(b): Geophysical Surveys

Geophysical information obtained from Anglo Platinum 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

Mr 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.

Mr WJ Visser (PTM) and Mr BW Green were responsible for the interpretation and modelling of the information, with the assistance of Anglo Platinum. 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.

Details of 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:-

- 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.
- Position - Garmin GPS 12, Garmin GPS 72 and Magellan eXplorist 300 - These are 12- (Garmins) and 14-channel (Magellan) hand-held navigation GPS's; 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 GPS's 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.
- 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.

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 GPS's 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° true north. The GPS's 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 65km.

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 Bouguer 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 were 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.

Details of Ground 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 carried out after the gravity survey.

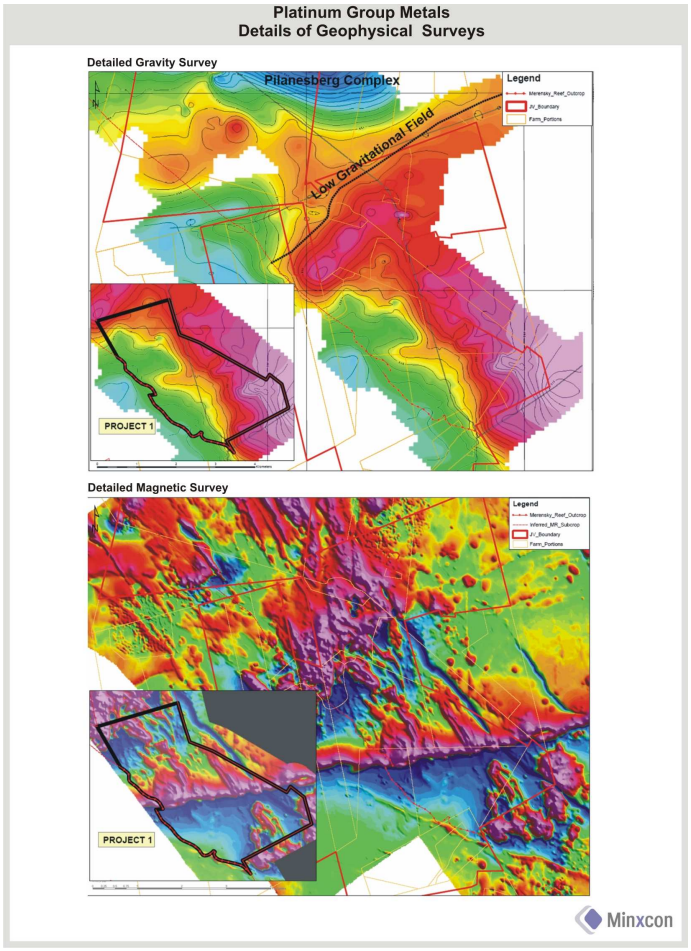
The instruments used for this survey included:

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

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.

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.

Figure 14 - Detailed Gravity and Magnetic Surveys



Details of Aeromagnetic Survey

The survey was flown by Fugro Airborne using a Midas Heli-borne magnetic gradiometer system. A total of 25,324 line kilometres were flown on lines with a direction of 55° (true north) and with a sensor at a nominal elevation of 20m. The area covered by the survey was some four times larger than the WBJV area, which was situated in the north western quadrant of the surveyed area. The high resolution survey data was of a very high quality. To assist with the interpretation, various wavelength filters were applied, the gradients were derived in the three primary directions, the Euler grid path process was run on the dataset and some of the major discrete magnetic features modeled.

The structural features identified from the aeromagnetic data were interpreted in terms of a regional structural model. 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 Project Area 1. A second dyke occurs along the north-eastern 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.

The interpretation of the regional data showed:

- the Main Zone lithologies clearly lapping onto the Transvaal floor rocks to the west of the JV area and over the stratigraphically lower RLS units, particularly north of the Elands river,
- that the presence of the IRUPs could be determined on regional data but as composite zones and not as individual bodies.

The high resolution data allowed:

- the mapping of individual IRUP bodies,
- some ability to determine the relative depth of individual IRUP bodies,
- the fault bound nature of the composite IRUP zones to be made clear,
- the determination of the position and relative displacement of the Transvaal quartzites in the west of the area,
- the more precise positioning of the major faults , fault zones and dykes and their attitude and their incorporation in the determination of mining blocks and
- the determination of the presence and effect of numerous smaller structural elements.

Unfortunately the relative vertical movement of the various major RLS mafic units was not easily determined by the magnetometry and the magnetics was much less successful than the gravity survey in giving information on the sub-RLS structure, the variation in thickness of the mafic pile and the effect of the latter on the distribution and tenor of economic mineralisation.

Details of the Downhole Geophysics

The WBJV geological structural model was reconstructed throughout the pre-Feasibility Study and progressively upgraded as the borehole population increased. The base criteria and approach towards structural interpretation has resulted in a model that displayed fault orientations of northwest - southeast and dipping in a western direction. This was regarded as rather unique situation in the BIC. It became evident from plotting results on numerous cross-sections drawn through the Project Area that faults tend to increase in dip from west to east away from the subcrop environment.

This interpretation/extrapolation was positively supported by an initial internal process of core orientation through gonio-scope and stereo-net plotting of a small population of holes across the area. In an effort to confirm the above structural orientation the company committed to a down-hole geophysics project on 31 boreholes carefully selected across the prospect area

Figure 15 - Details of Downhole Geophysically Surveyd Boreholes

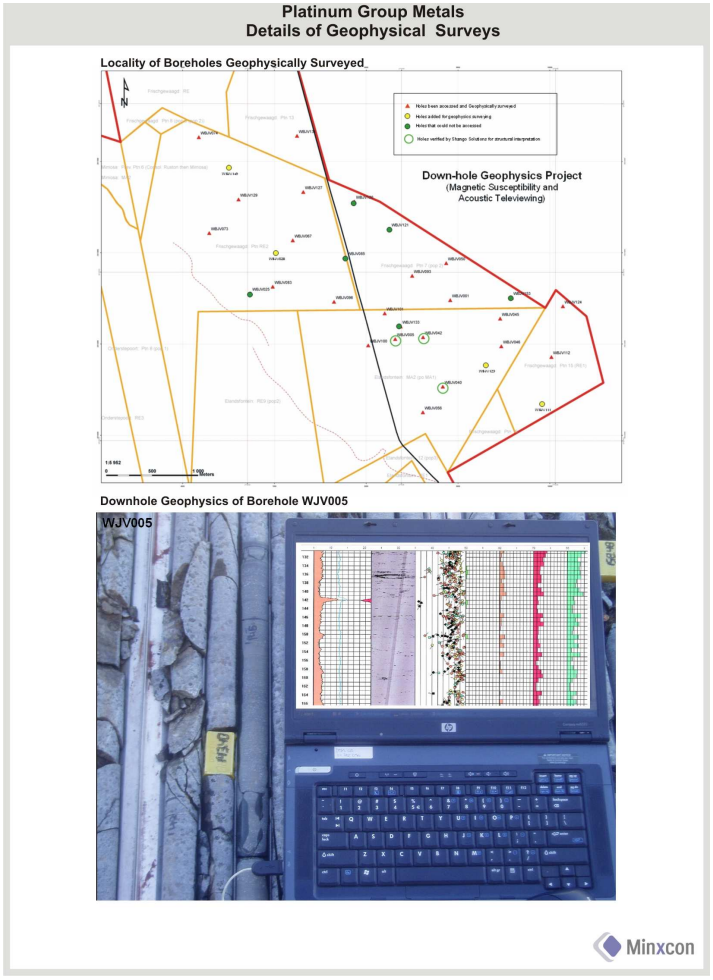


Table 21 - Details of Downhole Geophysics

Hole ID	EOH metres	Depth surveyed acoustic televiewing	MR depth	UG2 depth	Choice
WBJV001	476	469	448.65	472.80	Y
WBJV005	518	509	476.00	486.00	Y
WBJV025	137	caved	114.66	123.12	Y
WBJV040	388	387	385.84	434.14	Y
WBJV042	507	505	503.93	525.50	Y
WBJV045	587	507	422.00	575.00	Y
WBJV046	527	509	503.06	544.80	Y
WBJV050	603	596	531.09	592.42	Y
WBJV056	320	302	250.00	287.86	Y
WBJV067	400	391	329.14	377.57	Y
WBJV073	195	507	146.56	160.14	Y
WBJV074	552	544	522.20	531.49	Y
WBJV083	186	183	144.13	182.37	Y
WBJV085	516	caved	468.16	509.16	Y
WBJV093	474	432	401.17	440.43	Y
WBJV096	479	432	338.82	418.78	Y
WBJV100	444	433	326.70	409.65	Y
WBJV101	553	507	499.00	550.00	Y
WBJV112	551	501	453.35	503.20	Y
WBJV120	446	caved	331.30	331.06	Y
WBJV121	427	caved	342.12	388.67	Y
WBJV124	588	577	490.21	541.71	Y
WBJV127	527	486	447.00	417.35	Y
WBJV129	376	363	315.70	348.65	Y
WBJV131	590	586	549.56	-	Y
WBJV133	574	calliper stuck	509.24	528.40	Y
WBJV153	585	caved	513.83	551.25	Y
WBJV028	231	232	171.27	224.42	Y
WBJV158	0	0	756.27	767.00	n
WBJV170	452	298	259.10	268.00	Y
WBJV179	433	359	334.94	351.24	Y
WBJV162	0	0	586.53	600.19	n
WBJV149	444	439	418.67	427.55	Y

Holes completed successfully

Holes not able to access - obstructions

Holes surveyed during March/April 2007

Quik Log Geophysics (Pty) Ltd was contracted by PTM RSA to commence with field activities in mid December 2006. These 31 holes were selected based on:

- representative structural domains;
- where possible broad facies domains reconstructed at the time;
- holes which provided sound mother-holes with respect to minimal adverse ground conditions.

Some holes that intersected prominent structural features bounding regional structural blocks were intentionally selected.

Survey Equipment - Acoustic Televiewer Probe (ATV)

The Quik-Log acoustic tool is manufactured by ALT of Luxembourg and is a highly competitive instrument for down-hole survey activities. The ABI40 model been used at this operation is the new generation of this type of tool and supersedes the previous FAC40 model. The principal purpose of the Acoustic Televiewer ultrasonic borehole imaging tool is to provide detailed, oriented, structural information from a borehole side-wall. The Televiewer tool generates an image of the borehole side-wall by transmitting ultrasound pulses from a rotating sensor and recording the amplitude and travel time of the signals reflected at the interface between borehole fluid and the borehole wall. The amplitude of these reflections is representative for the properties of the rock surrounding the borehole.

Additionally, the travel time represents the borehole shape and diameter and is used to provide exceptionally accurate borehole diameter measurements which make the tool ideal for casing inspection; for the detection of very small diameter variation due to differential stress or for uses where a very precise borehole volume is required.

A borehole does not need to be clean for this tool to function but should be water-filled for optimal data collection efficiency during the wire line down-the-hole operation.

Results and interpretation

The first three holes' information that Quik Log produced; WBJV05, 40 and 42 were initially verified by PTM RSA and compared with core observations and depth positions of structures and found to be sound and correct.

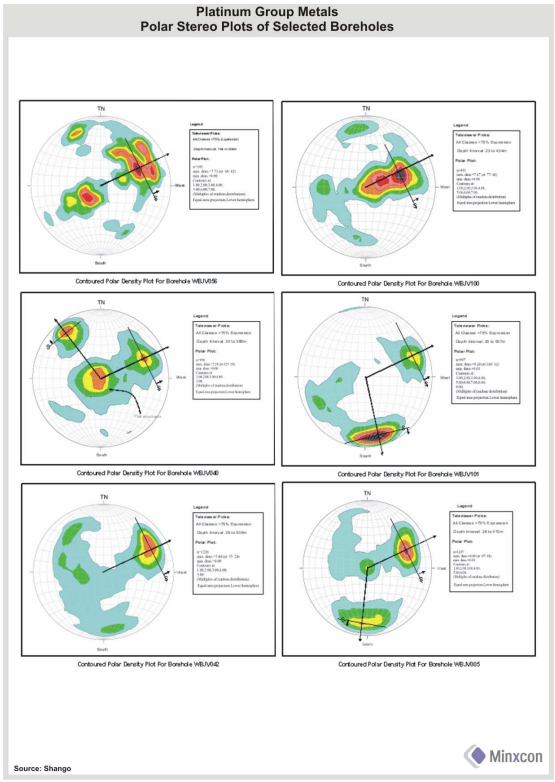
Shango Solutions was then contracted to verify and correlate the geophysics dataset with core and logging after they had discussed the data and approach to interpretation at the Quik Log office. The team conducted a workshop on the above mentioned holes. The conclusion of the team was that the comparison between the dataset and core information correlated positively. The dip angle; fault structures and pseudo-layering of the lithology; apparent dip direction of structures and lithology could all be positively correlated. This was supportive to the structural model being developed.

The results indicate that for the majority of the holes, a strong western dipping population of faults exists on the western and central section of the Project Area. Faults tend to increase in dip from west to east and seem to swing through the vertical to eastern steep dipping faults on the eastern boundary of the WBJV prospect. The impact this had on the existing structural model was that the dimension of fault losses on the eastern section of the prospect is smaller than previously interpreted.

Where sills, dykes and iron pegmatite's occur, magnetic susceptibility and gamma anomalies were generally detected. An in situ density profile at ~15cm vertical resolution along a borehole was measured and mapped. Structures intersecting a borehole were measured as anomalies on the borehole sidewall by high resolution orientated acoustic scans of the sidewall. The quality of the dataset was scrutinized by Anglo American Corporation and in general the capturing process and data quality was found to be at an acceptable standard. The data falls within the range of values expected in the BIC and stratigraphic features are well represented in the dataset as it would be expected (pyroxenite and chromitite units were recognized). It was confirmed that the ATV images are of good quality. The data was orientated to magnetic north which was subsequently rectified by Quik Log and referenced to true north. The structural information "picked" from the ATV display resulted into polar stereo plots and rose-plots as final products and that the majority of holes intersected strong western dipping fault populations and that the initial approach by PTM introduced to the structural model appears to be correct.

A process was conducted to verify these polar plots with features seen in core. The position and orientation of these structures was plotted on the geological cross-sections. This process was completed by verifying feature orientation with the plan view of structural maps to confirm the orientation of the dyke and fault losses as interpreted in the structural model. The polar plots show other orientations such as northeast-southwest for borehole WBJV040. Some easterly dipping features can also be seen in the plots. The high frequency of faulting (noisiness in geophysics "picks") in this part of the BIC, will affect mining activities. A high frequency of fault losses is interpreted since fault features were observed in the core and confirmed by geophysics results.

Figure 16 - Polar Stereo Plots of Selected Boreholes



Item 10(c): Stratigraphy

The detailed stratigraphy of the western BIC is depicted in Figure 7. The identifiable units within the WBJV area are, from top to bottom:

- the anorthositic hanging wall sequence (HW5-1)
- the Bastard Reef pyroxenite
- the Mid3-1 units
- the Merensky Reef pyroxenite
- FW1-5
- the anorthositic footwall (FW6-12)
- the UG2 unit
- the underlying medium-grained norite (FW13)
- the multiple UG1 chromitite seams
- the underlying medium-grained mottled anorthositic FW16
- 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 and 1a area. The Main and Critical Zone sequences of the BIC, as seen in the WBJV boreholes, 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 mm's 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 BIC 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 due to palaeo-high environments within the footwall below the BIC. The general stratigraphy of the upper Critical Zone proximal to the primary economic reefs - Merensky and UG2 - is outlined as follows:

- 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 25m.

- 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. The BR is currently not perceived as 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 <1m 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 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 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 are not present in the Project Area and have 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 (~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 2m in the Abutment region to 30m 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 11: MINERALISATION

Item 11(a): Mineralisation Styles and Distribution

Bulk modal analyses were estimated based on the results from XRD analysis (RIR method) and optical microscopic examination. The results were as follows:-

Alteration - Silicates showed low to moderate alteration, mainly associated with fractured zones. The degree of alteration is not expected to hinder flotation results, but should be monitored.

Sulphide Assemblages - Sulphide composition of the samples was variable. The results of the estimated sulphide composition of the composite sample were as follows:-

- Chalcopyrite ($CuFeS_2$): 20%
- Pyrite (FeS_2): 2%
- Pyrrhotite (Fe_7S_8): 35%
- Pentlandite ((Fe, Ni)S): 43%

Examination of the polished thin-sections showed that the sulphides occurs as sporadically distributed, fine grained clusters associated with interstitial silicates or as isolated, coarse composite particles and blebs. The liberation characteristics of the sulphides are expected to be relatively good apart from the fine disaggregated disseminated chalcopyrite.

Item 11(b): PGE and Gold Deportment

PGE searches (including gold bearing phases) were conducted by manually scanning a selection of the polished sections utilizing a scanning electron microscope to obtain a statistical particle count. Approximately 237 particles were located in 8 thin sections and image analysis software was employed to measure the size of each particle.

Speciation

Taken as a whole, the proportions of the various PGE (+Au) species are depicted in Table 22 - The major PGE phase encountered was cooperite (PtS) which comprise 63% of the observed particles. Moncheite ($PtTe_2$) was the next most common PGE encountered (11%).

The major gold-bearing phase, electrum (AuAg), was found to comprise 6% of the observed particles. Braggite (PtPd)S is also fairly common and comprises 5% of the observed particles. Sperrylite ($PtAs_2$) is less common, comprising about 4% of the observed PGE's. A PGE phase composed of Pd, Pt, As and some Te was found to be present in 2.4% of the observed particles. Hollingworthite (RhAsS), isoferroplatinum (Pt_3Fe) and laurite (RuS_2) are less common PGE's, each comprising about 1.5% of the total observed particles. Froodite ($PdBi_2$) comprise about 1% of the observed particles and was found only in one thin-section (41/D4/B). The remaining 2.8% of the observed particles are composed of 9 other PGE and gold species.

In order to reach a better understanding of the PGE speciation, they were classified into five groups: a) sulphides, b) arsenides, c) Te-, Sb- and Bi-bearing, d) Au-bearing phases and e) Fe bearing PGE's. The sulphides comprise about 71% of PGE's observed (of which cooperate comprise about 90%).

Table 22 - PGE + Au speciation and proportional occurrence based on area (um2)

	No of Particles	Area (um^2)	% of Total Area	% in Group	Group Area	Group as % of Total
Sulphides						
Cooperite	69	12719.0	63.2	89.29		
Braggite	2	1069.1	5.3	7.51		
Laurite	10	319.5	1.6	2.24		
Platarsite	1	136.9	0.7	0.96	14241.6	70.8
Arsenides						
Sperrylite	8	754.3	3.8	46.56		
Palladoarsenide	1	87.0	0.4	5.37		
PdPt(Te)As	3	476.4	2.4	29.41		
Hollingworthite	2	302.3	1.5	18.66	1620.0	8.1
Te-Be- and Bi-Bearing						
Moncheite	37	2128.8	10.6	81.24		
Michenerite	5	119.1	0.6	4.55		
Stbiopalladinite	6	99.9	0.5	3.81		
Stumpflite	1	7.0	0.0	0.27		
PdSbBi	9	14.3	0.1	0.93		
Froodite	22	241.3	1.2	9.21	2620.4	13.0
Au-Bearing						
Electrum	52	1228.4	6.1	6.11		
Aurostibite	1	37.0	0.2	0.18		
Gold	1	72.5	0.4	0.36		
AuPdTe	2	4.9	0.0	0.02	1342.8	6.7
Fe-Bearing PGE's						
Isoferroplatinum	5	281.4	1.4	100	281.4	1.4
TOTAL	237	20106.2	100		20106.2	100

Mineral Association

With regard to the mineral associations, 77% of the total PGE's (+Au-phases) observed are associated with sulphides (mainly occluded or attached to chalcopyrite or pentlandite), 21% is occluded in silicates (usually in close proximity to sulphides), and only 2% occur on the boundary between silicate minerals and chromite. Microscopic observation indicates that PGE's (+Au-phases) concluded in silicate minerals occur mainly in the alteration silicates and in interstitial silicate phases i.e. talc, chlorite, quartz, amphibole and phlogopite.

Grain-Size Distribution

With regard to the grain size distributions, nearly 40% of the total PGE's are sulphides that are larger than $1000\mu m^2$ in size. Approximately 75% of the observed PGE's are larger than $100\mu m^2$ in size. It was also noted that the Te-, Sb- and Bi-bearing PGE's are generally smaller than the sulphides.

The largest PGE particle observed was measured at ~$5000\mu m^2$. Only 2 particles were measured at >$1000\mu m^2$, but this accounted for nearly 37% of total PGE's observed. The sulphide and PGE composition of the composite sample is normal for the Merensky Reef. The most significant observations resulting from these processes are:

- the formation of deleterious alteration products such as talc and chlorite which will tend to dilute grades of flotation concentrates, and affect the milling and filtration characteristics of the ore;
- alteration tends to disaggregate primary sulphides (and PGE's) in situ, to form very fine disseminated clusters within alteration silicates, which will require finer grinding to achieve effective liberation.

Item 11(c): 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.

Item 11(d): Geological Controls on Rock-Mass Behaviour

The following note, compiled by Dave Arnold of Turnberry Projects, summarises the geological controls that have influenced and governed geotechnical aspects of the design of underground mining operations at the WBJV project, from the point of view of both regional and local stability.

The geological environment in which the mine is situated plays an overriding role in determining and dictating the geotechnical mine design strategies that have been adopted for the optimum exploitation of the Resource. These strategies relate to fundamental design aspects such as regional mine support, internal stope support, stoping sequences, access tunnel layouts, tunnel support, and the protection of underground and surface access and engineering infrastructure.

The mine will primarily exploit Merensky reef, together with concurrent exploitation of selected areas of UG2 reef. The width and dip of these narrow, tabular reef horizons, their separation distance from one another, and the geological and geotechnical nature, strength and structure associated with them and the surrounding hangingwall and footwall country rock strata are vital elements in the design of the mine. The information

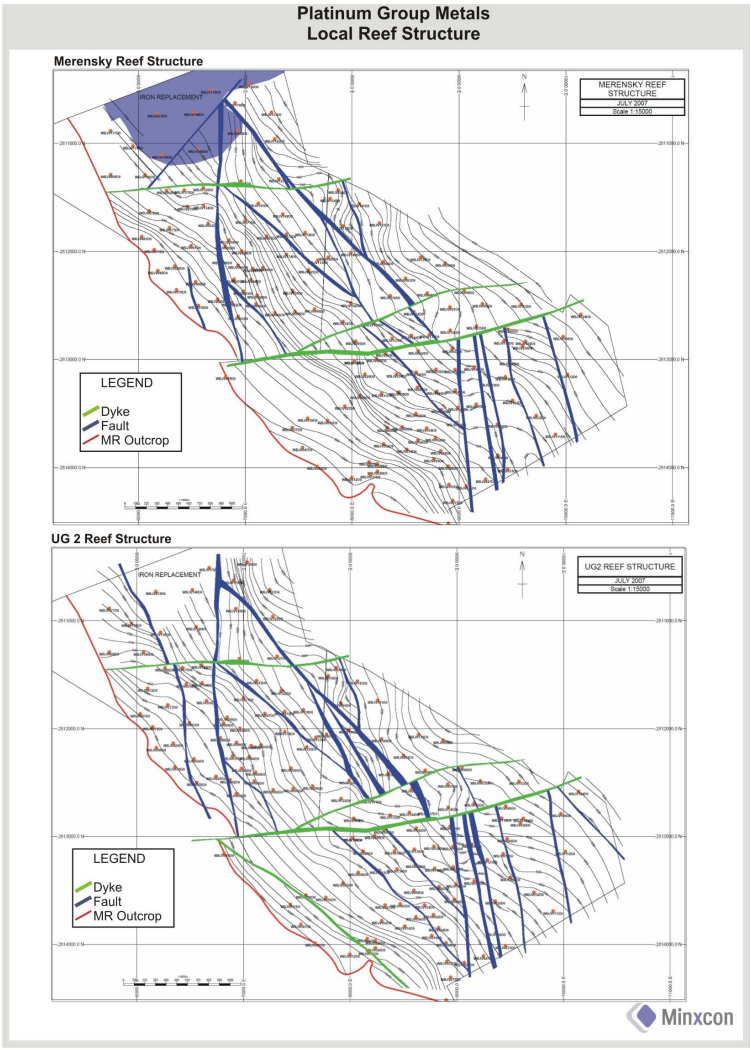
required to characterise the rockmass and execute the mine design has been largely gathered from an evaluation of borehole intersections, and this work is described in the following section of this note.

The property straddles the outcrop of the south-western limb of the Bushveld, only some four to eight kilometres south of the Pilanesberg intrusive alkali ring complex. The proximity of this extinct volcano has contributed to the generally fractured and disturbed nature of the orebody and its surrounding host rock formations.

Apart from the important issue of intact rock strength, a geotechnical assessment of the host rockmass, and the design of mining operations and mining infrastructure therein, is dominated by consideration of its structural characteristics.

The rockmass is dislocated and broken up by numerous **faults and dykes** . Some faults have large displacements of many tens of metres, but most have throws of only a few metres. The trend of the majority of faulting is northwest-southeast (i.e. orientated roughly parallel or sub-parallel to the general strike of the orebody), and many appear to comprise low angle shear planes. A number of the faults are combined with dyke intrusions. The spacing of the larger faults is generally less than a hundred metres. The major dykes, on the other hand, are orientated roughly due west-east, occur far less frequently, and are often associated with strike-slip displacements. The widths of the larger dykes vary up to a maximum of a few tens of metres. Other smaller dykes and sills are interposed with the major northwest-southeast orientated fault swarm. Two main dyke and sill types are encountered.
The majority are hard and competent dolerites that will pose little or no threat to mining operations, while the lamprophyres are invariably extremely weathered on exposure, sometimes to the extent of being completely decomposed, even at depth, and will present significant stability challenges in shafts, winzes, tunnels and stopes. The probable presence of water on these features cannot be excluded. Some of the faults and dykes are associated with large displacements that have the effect of dividing the orebody into a number of separate blocks, and that has also led to the creation of multiple reef intersections on several of the planned levels. The geotechnical ramifications of this are that, firstly, regional support in the mine will be somewhat enhanced by the relatively limited stoping spans that will be generated, and, secondly, main crosscut tunnels accessing these various faulted blocks are likely to be sited in a variety of differing host rock horizons, rendering prediction of likely rockmass conditions and support requirements problematic to some extent. The structure in the Project Area is illustrated in figure below:-

Figure 17 - Merensky and UG2 Reef Structure



Joint sets parallel the predominant fault and dyke trends. Their frequency, orientations and nature indicate that, of all the structural features, jointing is the most pervasive and is likely to play the most prominent role in dictating the in-situ strength, stability and competence of support pillars, stope hangingwall, and tunnel rockwalls, particularly in situations where these joints have little or no cohesion for whatever reason. Low angle thrust (ramp fault) or 'dome' jointing is also observed and is likely to adversely affect hangingwall stability in stoping operations on both of the reef horizons.

The immediate hangingwall and footwall **strata** associated with the Merensky and UG2 reefs is not significantly different from that of surrounding operations south of the Pilanesberg Complex. The immediate hangingwall of the Merensky reef is pyroxenite, grading upwards to norites and anorthosites. The footwall comprises norite and anorthosite. Although layered, there are few reef-parallel partings in either the hangingwall or footwall of the Merensky Reef. The immediate hangingwall of the UG2 Reef on the other hand sometimes comprises intercalated layers of pyroxenite and chromitite of various thicknesses. These all represent potential partings and planes of weakness, which are likely to emphasize the need for adequate pre-stressed stope support measures on this horizon. The footwall of the UG2 is usually a pegmatoidal felspathic pyroxenite of variable thickness, as is discussed a little later below.

The Bushveld Igneous Complex has been intruded into sedimentary rocks of the **Transvaal system** , and is thus underlain by quartzites and shales of this formation. The property sits astride the abutment zone where the footwall rock strata of the Bushveld Complex below the UG2 reef horizon are almost entirely truncated in the shallow regions close to the surface outcrop. This becomes less marked further down-dip, and the presence and thickness of these Bushveld footwall strata increases with increasing depth. The nature of the Transvaal palaeo-footwall surface also has some influence on the structure and character of the Bushveld reef zones.

The depth of **surface soil** and decomposed overburden is generally in the order of 4m or 5m, while the zone of weathered and unconsolidated rock (regolith) can extend to as much as 20m or more in places, but this is also very varied over the property. This naturally has implications for the collaring of shafts and declines, and for the foundations of headgears, hoists, plant and similar structures, depending upon their location.

Weathering associated with percolating groundwater, mainly along structural features, is in places persistent to some considerable depth. Some Bushveld rock types appear to be somewhat more prone to such deterioration than others, as is particularly the case with lamprophyre dyke and sill intrusions, as was noted above.

The **geotechnical investigation** of the orebodies and the immediate surrounding rockmass included conducting:-

- intact rock strength tests,
- drill core fracture logging and
- rockmass quality rating determinations.

Two deflections through target lithologies, intersected in each of ten vertical boreholes, representative of the whole mining area, were selected for this work. In addition, the entire length of boreholes drilled down the barrels of three proposed vertical shaft positions, situated close to the centre of gravity of the orebodies, were similarly evaluated.

The following tables summarise the geotechnical testwork and evaluation procedures that were undertaken to characterise the rockmass . They also provide a synopsis of results that emanated this work. In excess of one thousand **strength** and modulus tests were conducted on representative specimens selected from core recovered from the deflections and the shaft holes. The distribution of test samples selected from the deflection holes is shown in the table below:-

Table 23 - Intact Rock Strength Tests

Borehole Number	**General Location**	**Number of UCS Tests**	**Number of Shear Tests**
WBJV 005 D1, D2	Centre	48	1
WBJV 040 D5, D6	South-West	24	0
WBJV 042 D5, D6	Centre-South	115	0
WBJV 046 D5, D6	South-East	175	0
WBJV 050 D2, D3	Centre-East	136	0
WBJV 067 D2, D3	North-West	105	0
WBJV 093 D2, D3	Centre-East	103	0
WBJV 100 D4, D5	Centre-West	87	1
WBJV 101 D3, D4/2	Centre	31	1
WBJV 127 D5, D6	North-East	117	1
WBJV 213 D0 Shaft	Centre	241	0
WBJV 200 D0 Dyke	Centre	45	0
Total Tests		**1227**	**4**

UCS = Uniaxial compressive strength

The rock types making up each lithological unit encountered in the representative borehole deflections were sometimes found to vary, and, in some of the holes, whole lithological units were absent. This was to be expected given that the rockmass comprising the property is structurally complex. Nevertheless, an inventory of the lithological units and the rock types that are generally associated with each, and from which representative samples were selected, is tabulated below.

Table 24 - Lithological Units and Rock Types

Lithological Unit	Rock Types
Main Zone	Norite or Mottled Anorthosite
Hangingwall 5	Poikilitic Anorthosite
Hangingwall 4	Anorthosite (variously textured)
Hangingwall 3	Poikilitic Anorthosite
Hangingwall 2	Leuconorite
Hangingwall 1	Norite
Bastard Reef	Feldspathic Pyroxenite
Mid 3	Poikilitic Anorthosite or Leuconorite
Mid 2	Norite or Leuconorite
Mid 1	Norite
Merensky Reef Hangingwall	Feldspathic Pyroxenite
Merensky Reef	Chromitite / Feldspathic Pyroxenite
Merensky Reef Footwall	Feldspathic Pyroxenite or Fel-Pyrox / Hartzbergite (Tarentaal)
Footwall 1	Norite
Footwall 3	Norite or Leuconorite
Footwall 5	Leuconorite or Norite
Footwall 6	Poikilitic Anorthosite
Footwall 7	Norite or Hartzburgite or Norite-Leuconorite
Footwall 8	Leuconorite
Footwall 9	Poikilitic Anorthosite
Footwall 10	Hartzbergite
Footwall 11	Norite / Leuconorite
Footwall 12	Poikilitic Anorthosite
UG-2 Hangingwall	Feldspathic Pyroxenite
UG-2 Reef	Chromitite
UG-2 Footwall	Feldspathic Pyroxenite
Footwall 13	Norite or Anorthosite or Leuconorite
UG-1 Hangingwall	Norite or Feldspathic Pyroxenite
UG-1 Reef	Chromitite or Hartzbergite or Feldspathic Pyroxenite / Chromitite
UG-1 Parting	Feldspathic Pyroxenite or Norite or Leuconorite or Anorthosite
UG-1 Footwall	Feldspathic Pyroxenite or Leuconorite
Footwall 16	Feldspathic Pyroxenite or Norite or Leuconorite to Anorthosite
Basement	Basement Granulites and Sedimentary Rocks
IRUP	Iron Replacement Pegmatoid
Lamprophyric Dyke and Sills	Lamprophyre
Dolerite Dykes and Sills	Dolerite

Samples were tested for Density, Uniaxial Compressive Strength, Elastic Modulus and Poisson's Ratio. In most cases, at least three individual specimens were cut and tested from each sample, yielding average results for the sample. These results, combined with the rock mass ratings discussed later, were used to calculate the rock mass strength, which in turn allowed for the design of support pillars and stope spans for stable mining operations. The table below shows the average results that were obtained from all tests that were conducted on each of the lithological horizons that were intersected in the deflection boreholes.

Table 25 - Average Test Results for each Lithological Unit

Lithology & Rock Description	Density (g/cm^3)	UCS Strength (MPa)	Secant Modulus (GPa)	Poisson's Ratio
Hangingwall Zone				
Main Zone	2.93	177.74	91.91	0.26
Hangingwall 5	2.93	188.31	89.64	0.26
Hangingwall 4	2.81	229.05	84.16	0.26
Hangingwall 3	2.79	209.17	79.55	0.26
Hangingwall 2	2.80	216.42	73.65	0.27
Hangingwall 1	2.85	204.14	83.78	0.26
Bastard Reef	3.20	165.89	103.45	0.22
Mid 3	2.83	202.10	81.49	0.26
Mid 2	2.83	200.55	77.52	0.25
Merensky H/Wall				
Mid 1	2.97	179.85	81.98	0.22
MReef Hangingwall	3.10	166.31	107.12	0.24
Merensky Reef Zone				
Merensky Reef	3.18	152.10	106.25	0.24
Merensky Footwall				
MReef Footwall	3.05	163.32	94.47	0.24
Footwall 1	3.13	179.21	89.74	0.27
Footwall 2				
Footwall 3	2.95	169.54	82.64	0.23
Footwall 4	2.84	191.84	85.38	0.26
Middling Zone				
Footwall 5	2.84	170.97	85.66	0.26
Footwall 6	2.77	239.91	81.88	0.27
Footwall 7	2.92	164.65	88.60	0.25
Footwall 8	2.82	177.53	80.95	0.27
Footwall 9	2.72	187.65	82.48	0.27
Footwall 10	2.84	188.27	87.83	0.29

Footwall 11	2.86	185.26	84.78	0.25
Footwall 12	2.77	209.45	74.49	0.26
UG-2 Hangingwall				
UG-2 Hangingwall	3.18	156.53	107.05	0.21
UG-2 Leader 2				
UG-2 Leader 1				
UG-2 Pyroxenite				
UG-2 Reef Zone				
UG-2	4.01	85.08	107.56	0.28
UG-2 Footwall				
UG-2 Footwall	3.18	126.41	88.19	0.22
Footwall 13	2.91	181.71	90.34	0.25
Footwall Zone				
UG-1 Hangingwall	3.09	180.97	110.38	0.23
UG-1	3.93	90.81	94.13	0.32
UG-1 Parting	3.24	176.49	97.05	0.24
UG-1 Footwall	3.08	181.69	101.34	0.20
Footwall 16	3.04	177.74	107.73	0.25
Basement	3.03	261.74	90.31	0.23
Intrusives				
Irup	3.25	188.08	108.22	0.23
Lamprophyre	2.67	47.34	9.80	0.17
Dolerite	3.16	182.00	86.17	0.27

Core from each deflection was **geotechnically logged** from above or as close as possible to the Bastard reef to about 20m below the UG1 horizon. In a number of instances some of the horizons, including on occasion the Merensky or UG2 reefs, were either not developed or were faulted out so that the entire lithological sequence was not always present. Fracture logging described the positions of each natural fracture and its angle, planarity, roughness and filling. Standard rock quality designation ("RQD") values were also recorded for each core run and, where possible, for each lithology, calculated as follows:-

$$\text{RQD\%} = \frac{\text{Total Length of Core} \geq 100\text{mm} \times 100}{\text{Length of core run}}$$

All of the other parameters that are required for the calculation of rock mass ratings were also recorded for each lithology.

Using all of the information gathered above, a number of industry-accepted rating systems were employed to **classify the rockmass** associated with each of the various lithologies for the purpose of providing fundamental input into the geotechnical models, and for facilitating the design of the geotechnical aspects associated with mining operations and excavation support requirements.

The **Q-System classification** is based on three aspects:

- rock block size (RQD/Jn)
- joint shear strength (Jr /Ja)
- confining stress (Jw/SRF)

Where: RQD is the rock quality designation

Jn	is the joint set number
Jr	is the joint roughness number
Ja	is the joint alteration number
Jw	is the joint water reduction factor
SRF	is the stress reduction factor

Q is calculated as:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{J_a} \times \frac{J_w}{SRF}$$

The modified stability number ‘N’ is derived from the Q-System. The parameter SRF (the stress factor) is not used, and three factors are applied to take account of the rock strength to stress effect, joint orientation, and gravity. In this situation ‘Q’ and ‘N’ have similar values, so they have been combined in the table below.

Table 26 - Combined ‘Q’ and ‘N’ Values for each Lithological Unit

Lithology & Rock Description		**Average ‘Q’&‘N’ Value**	**Maximum ‘Q’&‘N’ Value**	**Minimum ‘Q’&‘N’ Value**	**Standard Deviation**
Hangingwall Zone					
	MZn	2.5	5.3	0.1	2.1
	HW-5	40.2	138.7	0.4	42.0
	HW-4	36.7	139.3	0.9	36.4
	HW-3	52.2	152.8	3.6	50.4
	HW-2	51.4	149.9	2.6	53.3

	HW-1	50.0	142.6	9.0	40.9
	BR	37.8	112.6	8.4	31.2
	MID-3	37.0	118.6	0.4	35.0
	MID-2	32.4	142.7	0.4	35.7
	MID-1	37.1	142.7	0.4	43.3
	MR-HW	38.9	142.7	0.4	45.3
Merensky Reef Zone					
	MR	42.3	142.7	0.4	46.5
Merensky Footwall					
	MR-FW	36.8	142.7	2.4	45.0
	FW-1	66.0	129.9	31.0	45.3
	FW-2	28.7	129.9	0.4	43.1
	FW-3	22.9	76.3	0.3	25.6
Middling Zone					
	FW-4	11.3	31.0	0.2	11.0
	FW-5	27.0	104.4	0.2	38.5
	FW-6	25.8	96.0	0.3	25.7
	FW-7	24.0	151.3	0.1	38.0
	FW-8	28.1	93.3	0.2	28.8
	FW-9	21.2	58.5	0.2	20.4
	FW-10	25.3	58.5	4.7	19.5
	FW-11	12.3	52.0	0.0	13.4
	FW-12	14.7	51.8	0.0	15.8
UG-2 Hangingwall					
	UG2-L3				
	UG2-H/W	15.9	77.3	0.6	19.2
	UG2-L2	14.6	28.0	0.6	11.2
	UG2-L1	14.6	28.0	0.6	11.2
	UG2-Pyx	19.7	77.3	0.6	25.1
UG-2 Reef Zone					
	UG-2	13.9	77.3	0.6	18.5

UG-2 Footwall					
	UG2-FW	13.8	57.2	0.6	16.4
	FW-13	16.9	153.9	0.1	37.1
Footwall Zone					
	UG1-HW	10.4	35.6	1.1	8.7
	UG-1	22.4	128.5	0.4	33.8
	UG1-P	20.1	95.4	0.4	25.3
	UG1 FW	123.4	745.3	0.4	238.0
	FW-16	20.2	55.2	1.8	18.0
Basement					
	Granulites	31.3	121.7	0.2	42.1

The geomechanical **Rock Mass Rating ("RMR")** is obtained by summing five parameter values and then adjusting this total by taking into account the effects of joint orientations. The parameters included in the system are as follows:

- rock material strength (UCS)
- RQD
- joint spacing
- joint roughness and separation
- groundwater

The average RMR values determined for the various lithologies are presented in the table below:-

Table 27 - RMR Values for each Lithological Unit

Lithology & Rock Description		**Average RMR Value**	**Maximum RMR Value**	**Minimum RMR Value**	**Standard Deviation**
Hangingwall Zone					
	MZn	55.8	72.3	41.8	12.6
	HW-5	65.0	82.8	52.5	8.3
	HW-4	63.8	79.0	54.7	7.8
	HW-3	65.8	75.0	55.2	7.1
	HW-2	69.7	87.0	55.3	10.6
	HW-1	67.4	87.0	55.7	7.9
	BR	65.8	81.0	55.4	6.5

	MID-3	63.5	75.0	47.6	8.1
	MID-2	64.6	75.0	52.6	7.3
	MID-1	63.5	75.0	42.8	8.4
	MR-HW	62.7	75.0	42.8	8.8
Merensky Reef Zone					
	MR	62.8	75.0	42.8	9.1
Merensky Footwall			0.0	0.0	
	MR-FW	61.7	72.9	55.6	6.8
	FW-1	59.2	67.7	42.8	11.6
	FW-2	61.0	75.0	42.8	10.4
	FW-3	66.0	75.0	55.6	6.5
Middling Zone					
	FW-4	64.1	75.0	55.6	6.1
	FW-5	63.5	75.0	55.7	6.5
	FW-6	62.9	75.0	52.8	6.1
	FW-7	62.2	73.0	42.5	8.1
	FW-8	63.4	82.0	52.4	7.7
	FW-9	63.7	82.0	55.5	8.0
	FW-10	62.8	67.9	55.7	5.8
	FW-11	62.3	82.0	49.7	8.7
	FW-12	60.5	87.0	35.3	10.6
UG-2 Hangingwall					
	UG2-L3				
	UG2-H/W	61.5	87.0	35.3	11.2
	UG2-L2	72.3	87.0	62.8	10.6
	UG2-L1	72.3	87.0	62.8	10.6
	UG2-Pyx	65.8	87.0	47.3	11.7
UG-2 Reef Zone			0.0	0.0	
	UG-2	60.8	87.0	35.9	11.3
UG-2 Footwall					
	UG2-FW	65.9	87.0	47.3	10.1
	FW-13	63.5	81.7	47.3	9.0

Footwall Zone					
	UG1-HW	59.8	72.9	42.8	8.3
	UG-1	61.0	82.4	42.4	9.1
	UG1-P	59.9	83.0	42.4	9.9
	UG1 FW	66.5	80.0	57.9	8.0
	FW-16	60.7	75.4	45.5	8.1
Basement					
	Granulites	61.7	67.1	54.8	4.5

The **Mining Rock Mass Rating ("MRMR")** system takes into account the same parameters as the geomechanics system, but combines the groundwater and joint condition, resulting in consideration of the following four parameters:

- rock material strength (UCS)
- RQD
- joint spacing
- joint condition and ground water.

Further adjustments are applied to the MRMR value to take account of weathering of the rock mass, joint orientation relative to mining excavations, mining-induced stresses and blasting effects. In the context of this project, MRMR values have been used to calculate rock mass strengths for the panel pillar design.

The average Mining Rock Mass Rating values determined for the various lithologies are presented in the table below:-

Table 28 - MRMR Values for each Lithological Unit

Lithology & Rock Description		**Average MRMR Value**	**Maximum MRMR Value**	**Minimum MRMR Value**	**Standard Deviation**
Hangingwall Zone					
	MZn	44.6	58	33	10.1
	HW-5	51.9	66	42	6.7
	HW-4	50.9	63	44	6.0
	HW-3	52.7	60	44	5.5
	HW-2	55.4	70	44	7.8
	HW-1	53.9	70	45	6.1

	BR	52.4	65	44	5.0
	MID-3	50.3	60	38	6.5
	MID-2	51.5	60	42	5.7
	MID-1	50.9	60	34	6.5
	MR-HW	50.3	60	34	6.7
Merensky Reef Zone					
	MR	50.3	60	34	7.0
Merensky Footwall					
	MR-FW	49.3	58	44	5.4
	FW-1	47.3	54	34	9.2
	FW-2	49.2	60	34	7.8
	FW-3	52.6	60	44	5.0
Middling Zone					
	FW-4	51.1	60	44	4.6
	FW-5	50.7	60	45	5.0
	FW-6	50.2	60	42	4.8
	FW-7	49.7	58	34	6.3
	FW-8	50.6	66	42	5.9
	FW-9	50.9	66	44	6.4
	FW-10	50.2	54	45	4.6
	FW-11	49.8	66	40	7.0
	FW-12	48.5	70	28	8.2
UG-2 Hangingwall					
	UG2-L3				
	UG2-H/W	49.2	70	28	8.7
	UG2-L2	57.7	70	50	8.4
	UG2-L1	57.7	70	50	8.4
	UG2-Pyx	52.6	70	38	9.4
UG-2 Reef Zone					
	UG-2	48.7	70	29	8.7
UG-2 Footwall					

	UG2-FW	52.4	70	38	7.7
	FW-13	50.7	65	38	7.0
Footwall Zone					
	UG1-HW	47.9	58	34	6.4
	UG-1	48.7	66	34	7.0
	UG1-P	47.8	66	34	7.9
	UG1 FW	53.1	64	46	6.4
	FW-16	48.4	60	36	6.2
Basement					
	Granulites	49.3	54	44	3.6

The geotechnical information presented in all of the above tables has been condensed from a large amount of detailed geotechnical data that has been gathered during the course of the investigation. The raw data comprises individual fracture logs of each hole, photographs of each box of core, and rock mass ratings and strength tests conducted on each lithological unit intersected in every borehole. These summaries do, however, serve to clarify and simplify the huge mass of collected data, and help establishment the necessary input data required for generalised mine design.

The geological and geotechnical information that was gathered during the course of the study, and which has been summarized and outlined in the preceding paragraphs of this note, was incorporated into the detailed planning and design of mine. Fundamental design aspects include those relating to regional stability, stope support, access layouts, tunnel support, and the layout and support of shafts and associated infrastructure. These aspects are elaborated upon in the mining section of the Updated Feasibility Study. However, for completeness, the geotechnical risks relating to the geology of the project, have been summarised below:-

- The orebody is **structurally complex** , particularly in terms of the frequency and nature of faulting and jointing. Structure that is even worse than predicted is likely to impact negatively on production profiles that do not take cognizance of, and make due allowance for, this important detail.
- **Rockmass conditions** that are significantly weaker than those that have been determined by geotechnical evaluation and have been anticipated in the modelling could negatively impact upon a number of critical design features such as pillar sizes, panel spans, extraction ratios and expected stope and tunnel support requirements.

- The precise geotechnical characteristics of the deep footwall succession remain relatively unknown cross the majority of the property as a result of the termination of most exploration boreholes immediately following the intersection of the reef zones of financial interest. It is however unlikely that any excavations will be sited in these horizons, other than the deeper reaches of the vertical shaft where a number of boreholes have, in fact, penetrated well into the basement rocks.
- Adverse **stope hangingwall rockmass conditions** that are weaker than those that have been determined by geotechnical evaluation could possibly lead to a significant reduction in sustainable stope spans. This could have a negative impact on both production and stope support requirements.
- The intensity, size and frequency of **thrust structures and potholing** is largely unknown at this stage. This could have a potential negative impact on production estimates and stope support requirements. Commiserate

Item 11(e): Local Structural Model

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° and dipping very steeply south-southeast or north-northwest, are related to the Rustenburg fault system. These structures were reactivated during the intrusion of the Pilanesberg Alkaline Complex. Dykes associated with this Complex intruded along these faults and joints.

Major structures, which occur within the WBJV area, include the Caldera and Elands faults and Chaneng Dyke and a major north-south trending feature, which can be observed across the entire Pilanesberg Complex. These east-west trending structures dip steeply (between 80° and 90°). The magnetics indicate that the Chaneng Dyke dips steeply to the north. This is consistent with similar structures intersected underground on the neighbouring Bafokeng Rasimone Platinum Mine, which all dip steeply northward.

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 outcrop position. There is also a subcrop of the Critical Zone against the main zone rocks.

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. The oldest event appears to be associated with dykes and sills trending at 305 degrees and is of post-BIC age. It 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. 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 phase of deformation may be related to a regional east-west-striking dyke system causing discontinuity on adjacent structures. 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.

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 and 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. 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, and 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.

Potholes

Identification of pothole intersections for the Merensky and UG2 reefs are assisted with interpreted stratigraphic anomalies. Simply, the following factors may indicate potholing:

- Where footwall stratigraphic widths are wider
- Where the Merensky Pyroxenite or UG2 chromitite is bifurcating, split or absent and
- Where the Merensky Reef width is anomalous with regard its normal facies widths.

Merensky Reef potholes have been identified within borehole intersections and the 3D seismic survey conducted by AP. A clear understanding of normal reef facies behaviour has afforded their interpretation. These potholes are defined as areas where normal reef characteristics are destroyed. Pothole areas are hence believed to be un-mineable and are considered as a geological loss. The immediate footwall lithology underlying the Merensky Reef and UG2 Chromitite is often a key identifier of potholing together with variations among deflections of the same borehole. Potholes appear to increase in frequency within the western most areas with the relative decrease in stability of the various lithologies in this area.

Replacement Pegmatites

Pseudo-form replacement bodies exist within the Project Area. A total of at least 12 boreholes of a population of 19 holes drilled in the Project 1A area have intersected iron replacement material ("IRUP").

It is evident from north-east orientated geological cross-sections constructed through the area that this IRUP mostly dominates the upper lithological sequence confined to the Main Zone and some upper lithological units of the Critical Zone. A typical down-dip section profile across Project Area 1A clearly indicates that the area of IRUP influence is sub parallel to the lithology pseudo layering.

The IRUP influence increases towards the Merensky subcrop environment and also further west where the Main Zone lies unconformable on Transvaal dolomites at shallow depths. The regional aeromagnetic image shows the surface expression of the IRUP influence and emphasizes the pseudo-morph shape of this anomaly.

Further south and on the remainder of Project Area 1, these IRUP anomalies occur as islands randomly spaced and are mostly recognizable on the aeromagnetic survey image. With the drilling grid averaging 250m Project Area 1 and in some geologically sensitive areas reduced to 130m, it was not always possible to further delineate the boundaries of these IRUP bodies.

Facies Model

The most pronounced PGE mineralisation in the Project Area 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 pyroxenite. The second important mineralised unit is the UG2 chromitite layer, which is on average 1.50m thick and occurs within the Project Area.

Merensky Reef

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 Area. From logging and sampling information of holes on the WBJV property, it is evident that the footwall mineralization 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 mineralization 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 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 PGE 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 Facies Types

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. 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 mineralization of PGE's. 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 harzburgite 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 0.5m to more than 3m 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.

Figure 18 - UG2 and MR Facies Types

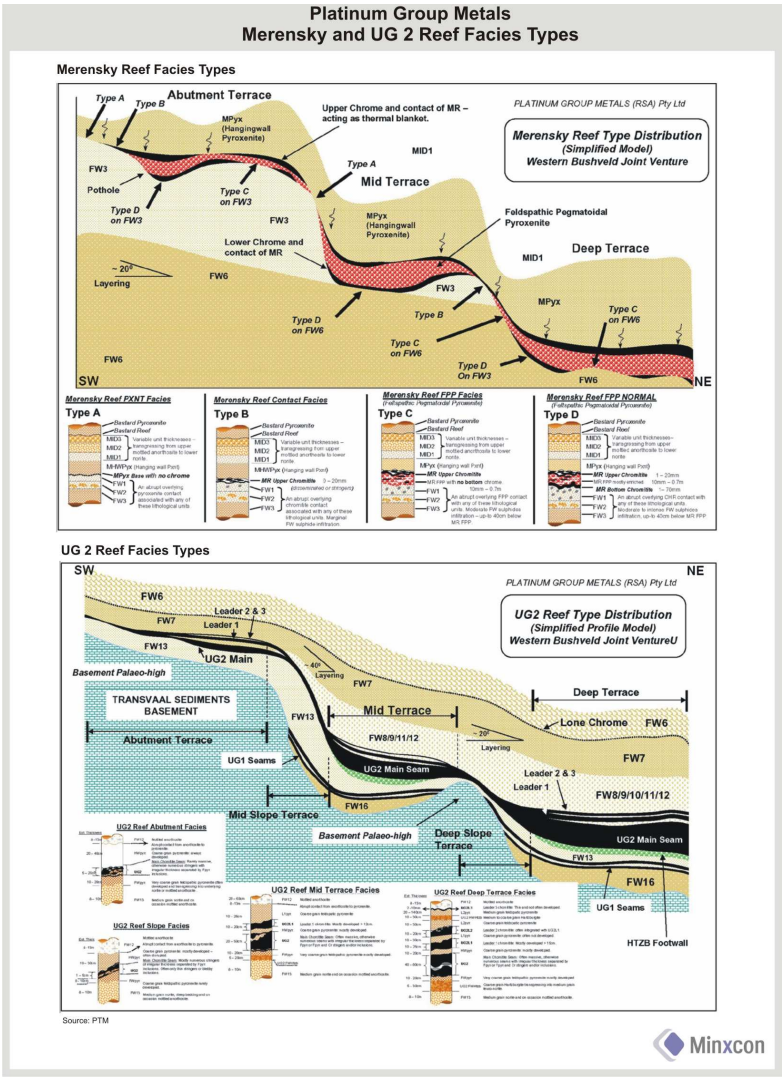
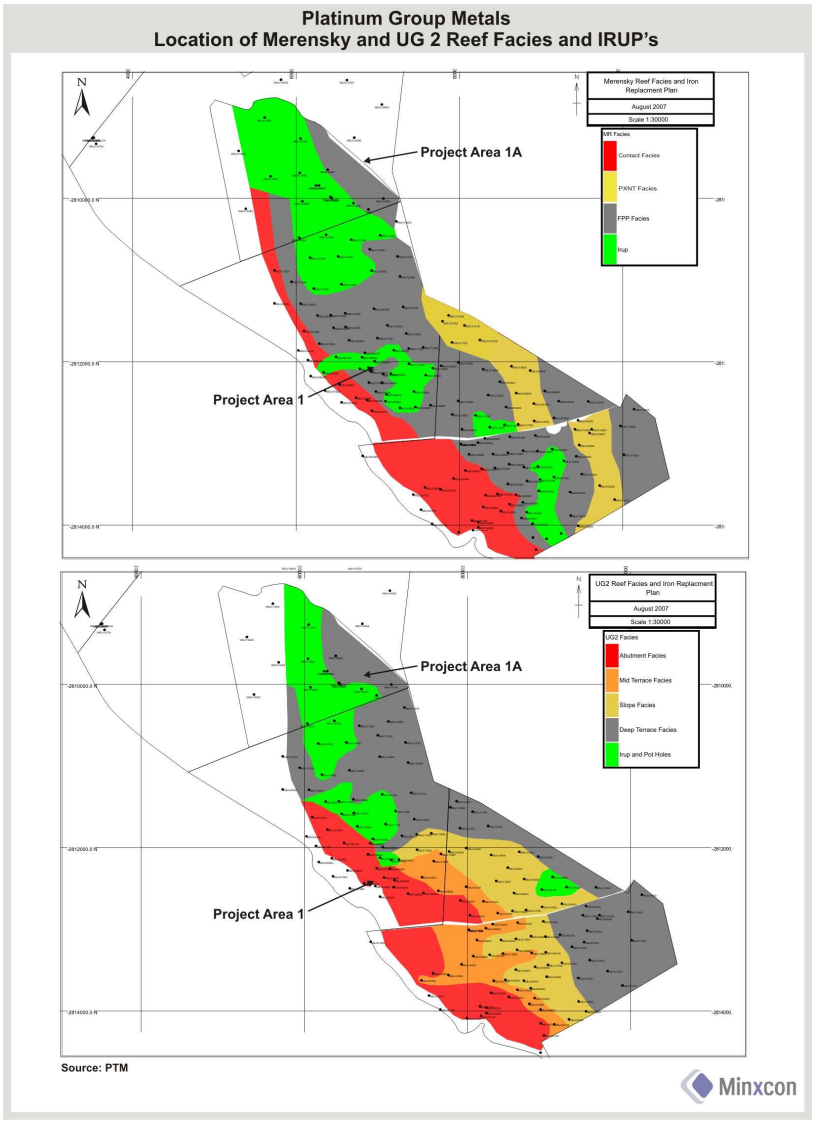


Figure 19 - Location of the MR and UG2 Facies Types in Project Area 1 and 1A



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 Anglo Platinum 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- (Garmin's) 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 Bouguer 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 were 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 north-eastern 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 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 and 1A 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 Chromitite seam - 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 122,361m has been drilled by PTM RSA the Project 1 and 1area (up to WBJV181). Altogether 29,303 field samples have been submitted for assaying along with 2,250 standards and 2,193 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 RSA 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 - Resource cuts

The methodology in determining the Resource cuts is derived from the core intersections. Generally, the economic reefs are about 60cm 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 and extended vertically to accommodate most of the metal content. If this should result in a Resource cut less than 80cm up from the bottom reef contact, it is extended further to 80cm. If the Resource cut is thicker than the proposed 80cm, the last significant reported sample value above 80cm is added to determine the top reef contact.

In the case of the UG2 Reef, the triplets (if and where developed and within 30cm from the top contact) are included in the Resource cut.

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 RSA geologist and transported to the exploration office by PTM RSA 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 RSA. 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 RSA 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 RSA 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 RSA 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 RSA. 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 RSA 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 RSA 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 RSA, 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 PGM’s 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 RSA protocols for quality control are as follows:

- The project geologist (Mr M Rhantho) oversees the sampling process.
- The core yard manager (Mr I Ernst) oversees the core quality control.
- The exploration geologists (Mr T Saindi and Mr T Thapelo) and the sample technician (Mr LJ Selaki) are responsible for the actual sampling process.
- The project geologist oversees the chain of custody.
- The internal Qualified Person (Mr W Visser) verifies both processes and receives the laboratory data.

- 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.
- Together with the project geologist, the Resource geologist determines the initial mining cut.
- The external auditor (Mr N Williams) verifies the sampling process and signs off on the mining cut.
- The second external database auditor (Mr A Deiss and Ms H Sternberg) verifies the SABLE database and highlights QA&QC failures.
- Ms R de Klerk (Maxwell) runs the QA&QC graphs (standards, blanks and duplicates) and reports anomalies and failures to the internal Qualified Person.
- The internal Qualified Person requests re-assay.
- 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:

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

Standards

The following analytical standards were used to assess the accuracy and possible bias of assay values for Platinum (Pt) and Palladium (Pd). Rhodium (Rh) and Gold (Au) were monitored where data for the standards were available, but standards were not failed on Rh and Au alone. (Table 29):

Table 29 - Summary of Reference Materials used

Standard Type	Pt	Pd	Rh	Au
CDN-PGMS-5	Yes	Yes		
CDN-PGMS-6	Yes	Yes		Yes
CDN-PGMS-7	Yes	Yes		Yes
CDN-PGMS-11	Yes	Yes		Yes
AMIS0005	Yes	Yes		
AMIS0007	Yes	Yes		
AMIS0008	Yes	Yes		
AMIS0010	Yes	Yes		
AMIS0014	Yes	Yes		

Since inception, Canadian standards from CDN Resource Laboratories Ltd, Canada were used consisting of CDN-PGMS 5, 6, 7 and 11. The matrix of these standards, with Pt concentrations near 1 g/t and Pd concentrations near 5 g/t, is significantly different to Pt and Pd concentrations for the Bushveld Complex. Contrasting concentrations found for the average grades on the Western Limb of the Bushveld Complex for Merensky and UG2 reefs are shown here:

Merensky Reef:	Pt (4g/t)	Pd (2g/t)
UG2 Reef:	Pt (3g/t)	Pd (1g/t)

Standards made from Bushveld Complex mineralisation were used from borehole WBJV027 onwards. AMIS0005 as well as AMIS0010 is made from UG2 Reef and AMIS0007 from Merensky Reef. These CRM’s are manufactured and sold by African Mineral Standards of Johannesburg.

Assay testing refers to Round Robin programs that comprise collection and preparation of material of varying matrices and grades to provide homogenous 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 batch is rejected when reference material assays are beyond the two standard deviations limit; and any two consecutive batches are rejected when reference material assays are beyond the two standard deviations limit on the same side of the mean. Canadian standards from CDN Resource Laboratories Ltd, Canada was initially used as reference consisting of CDN-PGMS 5, 6, 7 and 11 as no accredited standards for the Bushveld Complex was available.

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 2,250 standard sample values are plotted on a graph for each particular standard and element. The graphs are based on the actual round robin results from the different labs that evaluated the particular standards used on this project. The mean, two standard deviations (Mean+2SDV and Mean-2SDV) and three standard deviations (Mean+3SD and Mean-3SD) are plotted on the graphs as well as the assay values from PTM RSA. These graphs and calculations are available on request in digital format.

CDN PGM-5

All failures are listed in the following tables, including failures subsequently attributed to database errors, selection of wrong standards in the field, sample miss-ordering errors and bias from the laboratory. The tables include data for Rh and Au for the sake of completion.

If a failure is situated within the selected economic mining cut of the Merensky or UG2 reef, it will be reflected in the “Reef” column in the table by either one of two codes (MRMC or UG2MC):

MRMC:	Merensky Reef Mining Cut
UG2MC:	UG2 Reef Mining cut

Failed standards that were not taken within economic mining cut intersections were ignored for the purpose of this evaluation. The following tables display the standards that failed on 3SD using the CDN-PGMS-5 standard.

Table 30 - Standard Failed for Pt on 3SD

BHID	Defl	From	To	Sam_ID	Batch_Number	Mean	Mean+ 3SD	Mean-3SD	Pt Value (g/t)	Reef
WBJV002	D0	463.79	463.79	J2690	2005/03 WBJV002 D0	1.240	1.57	0.91	0	
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002 D0	1.240	1.57	0.91	0	MRMC
WBJV003	D0	533	533	J2714	2005/03 WBJV003 D0	1.240	1.57	0.91	0	
WBJV003	D0	536.42	536.42	J2726	2005/03 WBJV003 D0	1.240	1.57	0.91	0	
WBJV003	D2	548.25	548.25	J2820	2005/04 WBJV003 D2	1.240	1.57	0.91	0.19	
WBJV003	D2	550.32	550.32	J2832	2005/04 WBJV003 D2	1.240	1.57	0.91	5.88	
WBJV005	D0	476.54	476.54	J2868	2005/04 WBJV005 D0	1.240	1.57	0.91	0.2	
WBJV005	D0	484.53	484.53	J2880	2005/04 WBJV005 D0	1.240	1.57	0.91	5.98	UG2MC
WBJV005	D0	494.55	494.55	J2892	2005/04 WBJV005 D0	1.240	1.57	0.91	6.02	
WBJV005	D0	496.9	496.9	J2904	2005/04 WBJV005 D0	1.240	1.57	0.91	6.06	
WBJV006	D0	458.48	458.48	J2910	2005/04 WBJV006 D0	1.240	1.57	0.91	5.91	
WBJV006	D0	460.98	460.98	J2922	2005/04 WBJV006 D0	1.240	1.57	0.91	5.89	
WBJV006	D0	479.72	479.72	J2934	2005/04 WBJV006 D0	1.240	1.57	0.91	6.06	
WBJV006	D0	482.9	482.9	J2946	2005/04 WBJV006 D0	1.240	1.57	0.91	6.06	
WBJV008	D0	243.62	243.62	J2958	2005/04 WBJV008 DO	1.240	1.57	0.91	5.84	MRMC
WBJV006	D1	454.33	454.33	J2970	2005/04 WBJV006 D1	1.240	1.57	0.91	5.98	
WBJV006	D1	456.95	456.95	J2982	2005/04 WBJV006 D1	1.240	1.57	0.91	5.76	MRMC
WBJV008	D0	324.83	324.83	J2994	2005/04 WBJV008 DO	1.240	1.57	0.91	5.56	UG2MC

Table 31 - Standard Failed for Pd on 3SD

BHID	Defl	From	To	Sam_ID	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pd_Value (g/t)	Reef
WBJV002	D0	463.79	463.79	J2690	2005/03 WBJV002D0	5.760	6.66	4.86	0	
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002D0	5.760	6.66	4.86	0	MRMC
WBJV003	D0	533	533	J2714	2005/03 WBJV003D0	5.760	6.66	4.86	0	
WBJV003	D0	536.42	536.42	J2726	2005/03 WBJV003D0	5.760	6.66	4.86	0	
WBJV003	D2	548.25	548.25	J2820	2005/04 WBJV003D2	5.760	6.66	4.86	0.19	
WBJV003	D2	550.32	550.32	J2832	2005/04 WBJV003D2	5.760	6.66	4.86	5.88	
WBJV005	D0	476.54	476.54	J2868	2005/04 WBJV005D0	5.760	6.66	4.86	0.2	
WBJV005	D0	484.53	484.53	J2880	2005/04 WBJV005D0	5.760	6.66	4.86	5.98	UG2MC
WBJV005	D0	494.55	494.55	J2892	2005/04 WBJV005D0	5.760	6.66	4.86	6.02	
WBJV005	D0	496.9	496.9	J2904	2005/04 WBJV005D0	5.760	6.66	4.86	6.06	
WBJV006	D0	458.48	458.48	J2910	2005/04 WBJV006D0	5.760	6.66	4.86	5.91	
WBJV006	D0	460.98	460.98	J2922	2005/04 WBJV006D0	5.760	6.66	4.86	5.89	
WBJV006	D0	479.72	479.72	J2934	2005/04 WBJV006D0	5.760	6.66	4.86	6.06	
WBJV006	D0	482.9	482.9	J2946	2005/04 WBJV006D0	5.760	6.66	4.86	6.06	
WBJV008	D0	243.62	243.62	J2958	2005/04 WBJV008DO	5.760	6.66	4.86	5.84	MRMC
WBJV006	D1	454.33	454.33	J2970	2005/04 WBJV006D1	5.760	6.66	4.86	5.98	
WBJV006	D1	456.95	456.95	J2982	2005/04 WBJV006D1	5.760	6.66	4.86	5.76	MRMC
WBJV008	D0	324.83	324.83	J2994	2005/04 WBJV008DO	5.760	6.66	4.86	5.56	UG2MC

The following tables display the consecutive standards that failed on 2SD using the CDN-PGMS-5 standard.

Table 32 - Standard Failed for Pt on 2SD

BHID	Defl	From	To	Sam_ID	BATCH_NO	Mean	Mean + 2SD	Mean - 2 SD	Pt_Value (g/t)	Reef
WBJV002	D0	463.79	463.79	J2690	2005/03 WBJV002 D0	1.240	1.46	1.02	0	
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002 D0	1.240	1.46	1.02	0	MRMC
WBJV003	D0	533	533	J2714	2005/03 WBJV003 D0	1.240	1.46	1.02	0	
WBJV003	D0	536.42	536.42	J2726	2005/03 WBJV003 D0	1.240	1.46	1.02	0	
WBJV003	D2	548.25	548.25	J2820	2005/04 WBJV003 D2	1.240	1.46	1.02	0.19	
WBJV003	D2	550.32	550.32	J2832	2005/04 WBJV003 D2	1.240	1.46	1.02	5.88	
WBJV005	D0	476.54	476.54	J2868	2005/04 WBJV005 D0	1.240	1.46	1.02	0.2	
WBJV005	D0	484.53	484.53	J2880	2005/04 WBJV005 D0	1.240	1.46	1.02	5.98	UG2MC
WBJV005	D0	494.55	494.55	J2892	2005/04 WBJV005 D0	1.240	1.46	1.02	6.02	
WBJV005	D0	496.9	496.9	J2904	2005/04 WBJV005 D0	1.240	1.46	1.02	6.06	
WBJV006	D0	458.48	458.48	J2910	2005/04 WBJV006 D0	1.240	1.46	1.02	5.91	
WBJV006	D0	460.98	460.98	J2922	2005/04 WBJV006 D0	1.240	1.46	1.02	5.89	
WBJV006	D0	479.72	479.72	J2934	2005/04 WBJV006 D0	1.240	1.46	1.02	6.06	
WBJV006	D0	482.9	482.9	J2946	2005/04 WBJV006 D0	1.240	1.46	1.02	6.06	
WBJV008	D0	243.62	243.62	J2958	2005/04 WBJV008 DO	1.240	1.46	1.02	5.84	MRMC
WBJV006	D1	454.33	454.33	J2970	2005/04 WBJV006 D1	1.240	1.46	1.02	5.98	
WBJV006	D1	456.95	456.95	J2982	2005/04 WBJV006 D1	1.240	1.46	1.02	5.76	MRMC
WBJV008	D0	324.83	324.83	J2994	2005/04 WBJV008 DO	1.240	1.46	1.02	5.56	UG2MC
WBJV002	D0	463.79	463.79	J2690	2005/03 WBJV002 D0	1.240	1.46	1.02	0	
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002 D0	1.240	1.46	1.02	0	MRMC

Table 33 - Standard Failed for Pd on 2SD

BHID	Defl	From	To	Sam_ID	Batch_Number	Mean	Mean + 2SD	Mean - 2 SD	Pd_Value (g/t)	Reef
WBJV002	D0	463.79	463.79	J2690	2005/03 WBJV002 D0	5.760	6.36	5.16	0	
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002 D0	5.760	6.36	5.16	0	MRMC
WBJV003	D0	533	533	J2714	2005/03 WBJV003 D0	5.760	6.36	5.16	0	
WBJV003	D0	536.42	536.42	J2726	2005/03 WBJV003 D0	5.760	6.36	5.16	0	
WBJV003	D2	548.25	548.25	J2820	2005/04 WBJV003 D2	5.760	6.36	5.16	0.19	
WBJV003	D2	550.32	550.32	J2832	2005/04 WBJV003 D2	5.760	6.36	5.16	5.88	
WBJV005	D0	476.54	476.54	J2868	2005/04 WBJV005 D0	5.760	6.36	5.16	0.2	
WBJV005	D0	484.53	484.53	J2880	2005/04 WBJV005 D0	5.760	6.36	5.16	5.98	UG2MC
WBJV005	D0	494.55	494.55	J2892	2005/04 WBJV005 D0	5.760	6.36	5.16	6.02	
WBJV005	D0	496.9	496.9	J2904	2005/04 WBJV005 D0	5.760	6.36	5.16	6.06	
WBJV006	D0	458.48	458.48	J2910	2005/04 WBJV006 D0	5.760	6.36	5.16	5.91	
WBJV006	D0	460.98	460.98	J2922	2005/04 WBJV006 D0	5.760	6.36	5.16	5.89	
WBJV006	D0	479.72	479.72	J2934	2005/04 WBJV006 D0	5.760	6.36	5.16	6.06	
WBJV006	D0	482.9	482.9	J2946	2005/04 WBJV006 D0	5.760	6.36	5.16	6.06	

WBJV008	D0	243.62	243.62	J2958	2005/04 WBJV008 DO	5.760	6.36	5.16	5.84	MRMC
WBJV006	D1	454.33	454.33	J2970	2005/04 WBJV006 D1	5.760	6.36	5.16	5.98	
WBJV006	D1	456.95	456.95	J2982	2005/04 WBJV006 D1	5.760	6.36	5.16	5.76	MRMC
WBJV008	D0	324.83	324.83	J2994	2005/04 WBJV008 DO	5.760	6.36	5.16	5.56	UG2MC
WBJV002	D0	463.79	463.79	J2690	2005/03 WBJV002 D0	5.760	6.36	5.16	0	
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002 D0	5.760	6.36	5.16	0	MRMC

CDN-PGMS-6

No standards failed for Pt on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-6 standard.

No standards failed for Pd on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-6 standard.

CDN-PGMS-7

No standards failed for Pt on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-7 standard.

No standards failed for Pd on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-7 standard.

CDN-PGMS-11

No standards failed for Pt on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-11 standard.

No standards failed for Pd within on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-11 standard.

No standards failed for Au within or on 3SD. No consecutive standards failed on 2SD using the CDN-PGMS-11 standard.

AMIS0005

The following tables display the standards that failed on 3SD using the AMIS0005 standard.

Table 34 - Standard Failed for Pt on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pt_Value (g/t)	Reef
WBJV055	D1	252.84	252.84	P5197	2005/11/WBJV-024	3.380	3.88	2.89	0.410	
WBJV057	D0	181.93	181.93	P5429	2005/11/WBJV-025	3.380	3.88	2.89	0.570	
WBJV099	D2	458.69	458.69	P15626	2006/06/WBJV-040	3.380	3.88	2.89	0.020	UG2MC
WBJV109	D1	568.87	568.87	P17538	2006/07/WBJV-045	3.380	3.88	2.89	2.220	
WBJV089	D1	193.71	193.71	P8831	2006/02/WBJV-031	3.380	3.88	2.89	2.78	

Table 35 - Standard Failed for Pd on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pd_Value (g/t)	Reef
WBJV055	D1	252.84	252.84	P5197	2005/11/WBJV-024	2.230	2.50	1.96	0.240	
WBJV057	D0	181.93	181.93	P5429	2005/11/WBJV-025	2.230	2.50	1.96	0.080	
WBJV065	D0	318.53	318.53	P7043	2005/11/WBJV-025	2.230	2.50	1.96	1.200	
WBJV099	D2	458.69	458.69	P15626	2006/06/WBJV-040	2.230	2.50	1.96	0.030	UG2MC
WBJV109	D1	568.87	568.87	P17538	2006/07/WBJV-045	2.230	2.50	1.96	1.370	
WBJV061	D1	134.6	134.6	P6196	2005/11/WBJV-026	2.230	2.50	1.96		
WBJV089	D1	193.71	193.71	P8831	2006/02/WBJV-031	2.230	2.50	1.96	179	

AMIS0007

The following tables display the standards that failed on 3SD using the AMIS0007 standard.

Table 36 - Standard Failed for Pt on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pd_Value (g/t)	Reef
WBJV104	D1	535.21	535.21	P17100	2006/06/WBJV-042	2.480	2.90	2.06	0.010	

Table 37 - Standard Failed for Pd on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pd_Value (g/t)	Reef
WBJV104	D1	535.21	535.21	P17100	2006/06/WBJV-042	1.50	1.80	1.20	0	
WBJV048	D0	430.3	430.3	P3168	2005/10/RMJ-WBJV-021	1.50	1.80	1.20	0.44	

AMIS0008

The following tables display the standards that failed on 3SD using the AMIS0008 standard.

Table 38 - Standard Failed for Pt on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pd_Value (g/t)	Reef
WBJV178	D4	306.25	306.25	P23632	2007/03/WBJV-073	8.66	9.83	7.49	2.31	
WBJV198	D0	715.05	715.05	P28007	2007/08/WBJV-088	8.66	9.83	7.49	2.78	
WBJV111	D0	466.86	466.86	P20714	2007/02/WBJV-066	8.66	9.83	7.49	10.5	

Table 39 - Standard Failed for Pt on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 3SD	Mean - 3SD	Pd_Value (g/t)	Reef
WBJV178	D4	306.25	306.25	P23632	2007/03/WBJV-073	4.36	5.45	3.26	1.45	
WBJV198	D0	715.05	715.05	P28007	2007/08/WBJV-088	4.36	5.45	3.26	1.61	

AMIS0009

No standards failed for Pt on 3SD. No consecutive standards failed on 2SD using the AMIS0009 standard.

No standards failed for Pd within on 3SD. No consecutive standards failed on 2SD using the AMIS0009 standard.

AMIS0010

The following tables display the standards that failed on 3SD using the AMIS0010 standard.

Table 40 - Standard Failed for Pt on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 2SD	Mean - 2SD	Pt_Value (g/t)	Reef
WBJV123	D0	455.06	455.06	P18464	2006/09/WBJV-052	2.130	2.56	1.76	0.65	
WBJV139	D0	509.41	509.41	P22322	2007/03/WBJV-069	2.130	2.56	1.76	0.00	
WBJV146	D2	480.35	480.35	P13433	2006/10/WBJV-055	2.130	2.56	1.76	2.58	
WBJV178	D1	334.11	334.11	P23484	2007/03/WBJV-073	2.130	2.56	1.76	2.60	

Table 41 - Standard Failed for Pd on 3SD

BHID	Defl	From	To	Sam_Id	Batch_Number	Mean	Mean + 2SD	Mean - 2SD	Pd_Value (g/t)	Reef
WBJV123	D0	455.06	455.06	P18464	2006/09/WBJV-052	1.32	1.59	1.09	0.28	
WBJV139	D0	509.41	509.41	P22322	2007/03/WBJV-069	1.32	1.59	1.09	0.001	

AMIS0014

No standards failed for Pt on 3SD.

No standards failed for Pd on 3SD.

Summary of Standard Results

A failed standard is considered cause for re-assay if it falls within a determined economic mining cut for either the Merensky or UG2 reef (MRMC and UG2MC). In conclusion, the following failed standards listed in ÕF□&ÆR□C"□ □□□□□□□, are located in the economic mining cut. 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) were not included in the final results as the influence is deemed as not of material value.

Of the submitted 2,250 standard samples the total number of standards that failed for Pt and/or Pd based on 3SD deviations equals 39, Of these, only 8 are deemed to be true failures (present within the mining cut) and caused by laboratory problems which constitute a mere 0.36% failure rate.

Table 42 - Summary of Failed Standards

BHID	Defl	From	To	Sam_ ID	Batch Number	StdType	Pt	Pd	Reef	Reason for Failure
WBJV099	D2	458.69	458.69	P15626	2006/06/WBJV-040	AMIS0005	0.020	0.03	UG2MC	Possible Blank
WBJV033	D2	339.31	339.31	P318	200/08/WBJV-013	CDN11	0.080		MRMC	True Lab Failure
WBJV043	D1	579.90	579.90	P527	2005/09/WBJV/P527	CDN11	0.080		UG2MC	True Lab Failure
WBJV006	D1	456.95	456.95	J2982	2005/04 WBJV006 D1	CDN5	5.76	1.32	MRMC	Possible value swap
WBJV008	D0	324.83	324.83	J2994	2005/04 WBJV008 D0	CDN5	5.56	1.19	UG2MC	Possible value swap
WBJV008	D0	243.62	243.62	J2958	2005/04 WBJV008 D0	CDN5	5.84	1.26	MRMC	Possible value swap
WBJV005	D0	484.53	484.53	J2880	2005/04 WBJV005 D0	CDN5	5.98	1.34	UG2MC	Possible value swap
WBJV002	D0	465.74	465.74	J2702	2005/03 WBJV002 D0	CDN5	0.00	0.01	MRMC	Possible Blank

Blanks - QA&QC Results

Blank assay values of 2,193 blanks from PTM RSA were plotted on graphs for each particular element - Platinum, Palladium, Rhodium and Gold. A warning limit is also plotted on the graphs, which is equal to five times the blank background. These graphs and calculations are available on request in digital format.

Table 43 - Failed Blanks -Pt on 2std

BHID	Defl	From	To	ID	BATCH_NO	Pt	Reef	Lab
WBJV041	D1	488.25	488.25	P3490	2005/10/WBJV-023	2.59		SETP
WBJV007	D0	109.03	109.03	O3048	2005/05 WBJV 007 D0	3.75		SETP
WBJV057	D0	180.5	180.5	P5423	2005/11/WBJV-025	0.54		SETP
WBJV113	D1	429.75	429.75	P17672	2006/07/WBJV-046	0.46	UG2MC	SETP
WBJV096	D0	337.49	337.49	P15151	2006/04/WBJV-037	0.24	MRMC	SETP
WBJV008	D0	167.45	167.45	J2988	2005/04 WBJV008 DO	0.36		GEN
WBJV178	D4	307	307	P23637	2007/03/WBJV-073	0.31		SETP

Table 44 - Failed Blanks -Pd on 2std

BHID	Defl	From	To	ID	BATCH_NO	Pd	Reef	Lab
WBJV041	D1	488.25	488.25	P3490	2005/10/WBJV-023	1.55		SETP
WBJV045	D1	563.19	563.19	P3832	2005/11/WBJV-023	0.13		SETP
WBJV007	D0	109.03	109.03	O3048	2005/05 WBJV 007 D0	1.94	UG2MC	SETP
WBJV057	D0	180.5	180.5	P5423	2005/11/WBJV-025	0.14		SETP
WBJV113	D1	429.75	429.75	P17672	2006/07/WBJV-046	0.21	UG2MC	SETP
WBJV008	D0	167.45	167.45	J2988	2005/04 WBJV008 DO	0.88		GEN

Table 45 - Failed Blanks -Rh on 2std

BHID	Defl	From	To	ID	BATCH_NO	Rh	Reef	Lab
WBJV041	D1	488.25	488.25	P3490	2005/10/WBJV-023	0.26		SETP
WBJV007	D0	109.03	109.03	O3048	2005/05 WBJV 007 D0	0.34	UG2MC	SETP
WBJV057	D0	180.5	180.5	P5423	2005/11/WBJV-025	0.1		SETP
WBJV127	D0	490.17	490.17	P16703	2006/08/WBJV-051	0.1		SETP
WBJV172	D0	369	369	P22654	2007/03/WBJV-070	0.13		SETP
WBJV043	D1	578.93	578.93	P521	2005/09/WBJV/P521	0.04		SETP
WBJV112	D1	451.33	451.33	P17861	2006/08/WBJV-47	0.03		SETP
WBJV113	D1	429.75	429.75	P17672	2006/07/WBJV-046	0.06		SETP
WBJV096	D0	337.49	337.49	P15151	2006/04/WBJV-037	0.04	MRMC	SETP

Table 46 - Failed Blanks -Au on 2std

BHID	Defl	From	To	ID	BATCH_NO	Au	Reef	Lab
WBJV003	D1	498.58	498.58	N0168	2005/06 WBJV 007	0.04		SETP
WBJV026	D0	64.03	64.03	N516	2005/08/WBJV-009	0.06		SETP
WBJV007	D0	256.38	256.38	O3048	2005/05 WBJV 007 D0	0.02		SETP
WBJV009	D3	272.56	272.56	O3654	2005/06 WBJV009 D3	0.02	UG2MC	SETP
WBJV090	D2	212.13	212.13	P11742	2006/02/WBJV-032	0.02		SETP
WBJV092	D0	279.25	279.25	P11766	2006/02/WBJV-032	0.02		SETP
WBJV092	D0	294	294	P11790	2006/02/WBJV-032	0.05		SETP
WBJV092	D1	278.57	278.57	P11826	2006/02/WBJV-032	0.02		SETP
WBJV092	D1	285.88	285.88	P11934	2006/02/WBJV-032	0.05		SETP
WBJV092	D2	286.53	286.53	P11945	2006/02/WBJV-032	0.03		SETP
WBJV093	D0	398.96	398.96	P11962	2006/02/WBJV-032	0.04		SETP
WBJV035	D0	478.38	478.38	P1280	2005/09/WBJV-016	0.03		SETP
WBJV093	D0	446	446	P15038	2006/02/WBJV-032	0.02		SETP
WBJV093	D0	458.89	458.89	P15061	2006/02/WBJV-032	0.02		SETP
WBJV112	D0	449.43	449.43	P17782	2006/07/WBJV-046	0.02		SETP
WBJV178	D4	307	307	P23637	2007/03/WBJV-073	0.02		SETP
WBJV180	D2	273.5	273.5	P24795	2007/04/WBJV-075	0.02		SETP
WBJV048	D0	493.08	493.08	P3294	2005/10/RMJ-WBJV-021	0.04		SETP
WBJV041	D1	488.25	488.25	P3490	2005/10/WBJV-023	0.14		SETP
WBJV045	D1	563.19	563.19	P3832	2005/11/WBJV-023	0.02		SETP
WBJV056	D1	253	253	P6298	2005/11/WBJV-026	0.02		SETP
WBJV074	D0	514.85	514.85	P6819	2006/01/WBJV-029	0.02		SETP
WBJV085	D1	516.22	516.22	P8232	2006/01/WBJV-029	0.02		SETP
WBJV152	D0	323	323	P20165	2006/11/WBJV-061	0.02		SETP
WBJV187	D2	562.6	562.6	P26088	2007/05/WBJV-080	0.02		SETP
WBJV198	D0	578.5	578.5	P27809	2007/08/WBJV-088	0.02		SETP
WBJV198	D0	674.75	674.75	P27869	2007/08/WBJV-088	0.03		SETP
WBJV198	D0	807.75	807.75	P28241	2007/08/WBJV-088	0.02		SETP

Of the submitted 2,193 blanks, only six failed, with several failures most likely the result of data entry errors in the field. This constitutes a mere 0.27% failure rate.

Table 47 - Failed Blanks

BHID	Defl	From	To	ID	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
WBJV045	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
WBJV041	D1	488.25	488.25	P3490	2005/10/WBJV-023	2.59	1.55	0.26			SETP
WBJV113	D1	429.8	429.8	P17672	2006/07/WBJV-046		0.210	0.060		UG2MC	SETP

The table below explains briefly, what the possible causes could be for these blanks to have failed.

Table 48 - Reasons for Failed Blanks

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
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)
WBJV041	D1	P3490	2.59	1.55	0.26			SETP	True lab failure
WBJV113	D1	P17672		0.210	0.060		UG2MC	SETP	True lab failure

Assay Validation

Although samples are assayed with reference materials, an assay validation program 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 validation program consists of the following:

- A re-assay program 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 are conducted at Set Point
- Check assays conducted at an independent assaying facility (Genalysis)

Re-assay - QA&QC Results

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 WBJV181). This procedure entailed the following: the failed standard (2) together with the standard (1) submitted before and the standard (3) submitted after the particular failed standard (2) as well as all submitted field samples (pulps) in between standard (1) and standard (3) were re-submitted for re-assaying.

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 (totalling 50 samples (pulps)). Please refer to 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. The new data has been incorporated into the database.

Table 49 - Re-Assayed Values for Failed Standards

BHID	Defl	Sam_ ID	Std_Type	Pt	Pd	Reef	Lab	Re-assay (Pt)	Re-assay (Pd)	Comment
WBJV033	D2	P318	CDN11	0.08		MR	SETP	0.11		Pass
WBJV043	D1	P527	CDN11	0.08		UG2	SETP	0.12		Pass

Laboratory Duplicates - QA&QC Results

The purpose of having field duplicates is that it provides a check on possible sample over selection. The field duplicate contains all levels of error - core or RC cuttings splitting, sample size reduction in the prep lab, sub-sampling at the pulp, plus the analytical error.

Field duplicates were however not used on this project for the very significant reason of the assemblage of the core. Firstly, BQ core has an outer diameter of only 36.5mm. Secondly, it is friable and brittle due to the chrome content making it extremely difficult to quarter the core. It usually ends up in broken pieces and not a solid piece of core.

Due to this problem, the laboratory was asked to regularly assay split pulp samples as a duplicate sample to monitor analytical precision. These graphs and calculations are available on request in digital format. The original analysis vs. the duplicate analysis showed no irregular values. This indicates no sample miss-ordering or nugget effect.

The equations for the trendlines on the graphs illustrating the correlation between the original and duplicate values plotted for Pt, Pd and Au respectively are as follows:

Pt: $y = 0.9946x + 3.8886$
$R^2 = 0.9996$

Pd: $y = 0.9891x + 3.6861$
$R^2 = 0.9997$

Au: $y = 0.9959x + 0.3451$
$R^2 = 0.9969$

The correlation is higher than 99% in all three cases and is deemed as accurate.

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 duplicates are divided in groups of 11 and a new mean and median absolute difference are obtained for Pt and Pd.

The precision is then plotted as percentages calculated using the following formula:

$Pc = 2So/C + 2K$

Where Pc = Precision in percentage

Regression Coefficient So = Intercept = 4.8876

Regression Coefficient K = X Variable = 0.0220

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 reflects the overall low grade of Au in the intersections.

Check Assays

These complete data set consisting of graphs and calculations are available on request in digital format...

It was decided to use Genalysis in Perth as the second laboratory for checks on the assay results from Set Point. A total of 1104 samples were selected and as most of the check sampling sent to Genalysis was within the mining cuts, the lab was requested to add Osmium, Iridium and Ruthenium to the assay process to determine values for these elements. In addition to the extra elements, it was also required that the laboratory determined the specific gravity of each sample.

Due to the above request (assaying for Os, Ir and Ru) it was necessary that the laboratory used a different assay method to ascertain the values for the different elements. The Check Sampling was done using Ni-sulphide collection and not Lead (Pb) collection.

The results from the correlation of the different assayed elements Pt, Pd, Rh, Au and 4PGE between the 2 laboratories are as follows:

Pt: $y = 0.9806x + 0.0188$

$R^2 = 0.9946$

Pd: $y = 1.0062x - 0.0003$

$R^2 = 0.9962$

Rh: $y = 1.2901x - 0.0132$

$R^2 = 0.956$

Au: $y = 1.0253x - 0.0065$

$R^2 = 0.9677$

From these graphs, it was evident that the two laboratories are producing equivalent analyses and confirms the satisfactory performance of Set Point laboratory on the standards.

Adequacy of Sampling Procedures

The QA&QC practice of PTM RSA 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/Resource 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.

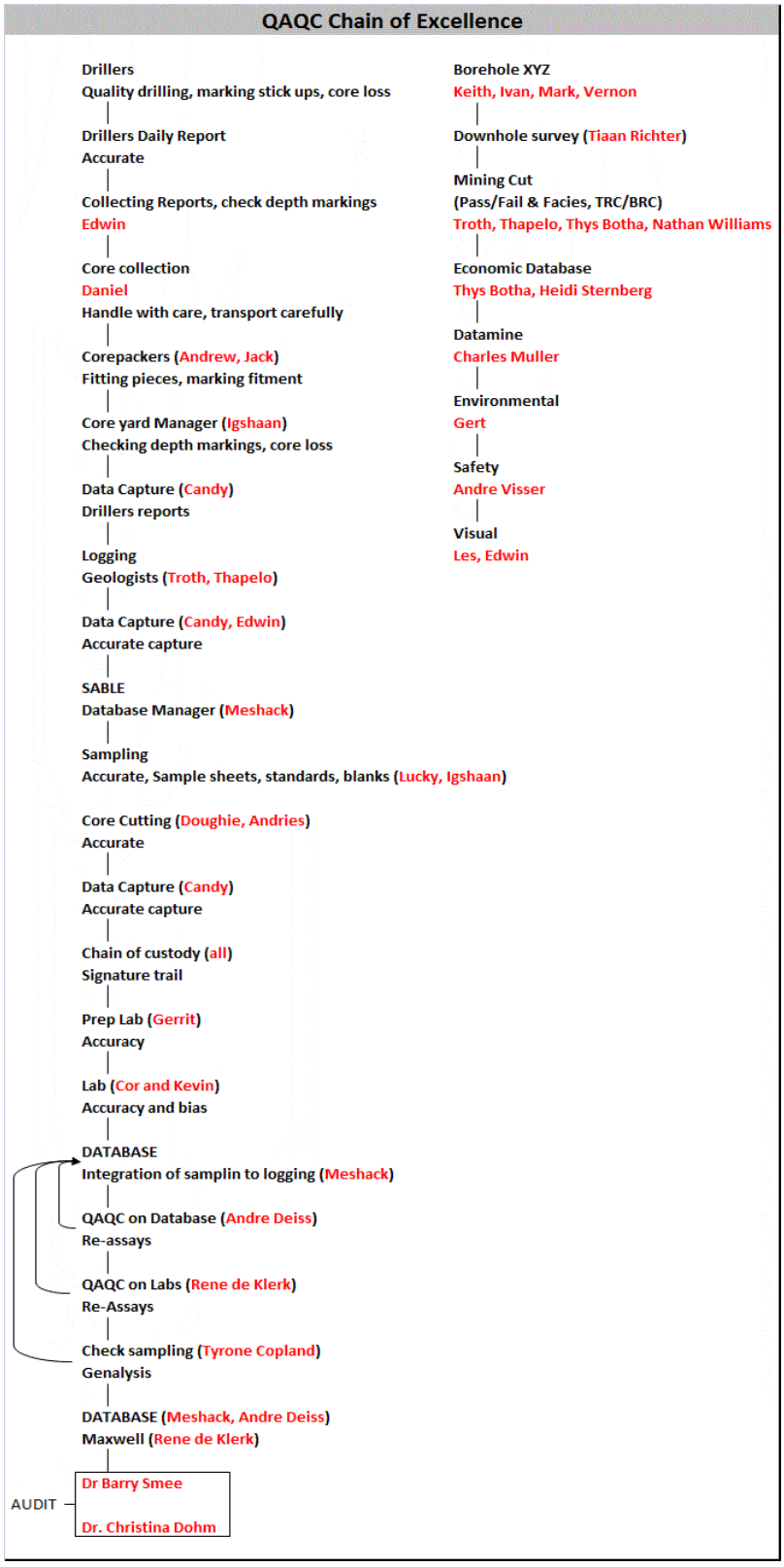
The author of this report (the independent Qualified Person) relied on sub-ordinate qualified persons for the following:

- 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 RSA 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 RSA'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 Qualified Person'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. These auditors also verified the standards, blanks and duplicates within the database as a second check to the QA&QC graphs run by Ms R de Klerk (Maxwell).

Below is the QAQC chain of Excellence Certificate:-

Figure 20 - QAQC Chain of Exellence



Item 15(d): Minor Elements (Ru, Ir and Os)

Assaying for ruthenium (“Ru”), iridium (“Ir”) and osmium (“Os”) is normally not carried out as assaying for Ru, Ir and Os is expensive and time consuming. 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 RSA sent a total of 1 000 samples for assaying to Genalysis which included samples over the Merensky and UG2 mining horizons.

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 that were used were only over the mining cut for the particular borehole. 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.

The graphs below demonstrate the correlation between the different elements.

Merensky Reef

Figure 21 - Ruthenium Correlation Graphs (MR)

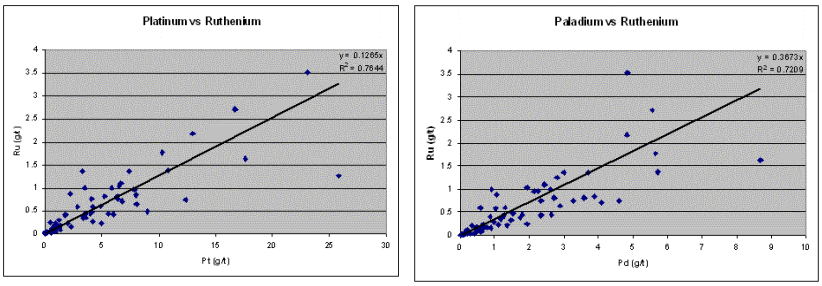


Figure 22 - Iridium Correlation Graphs (MR)

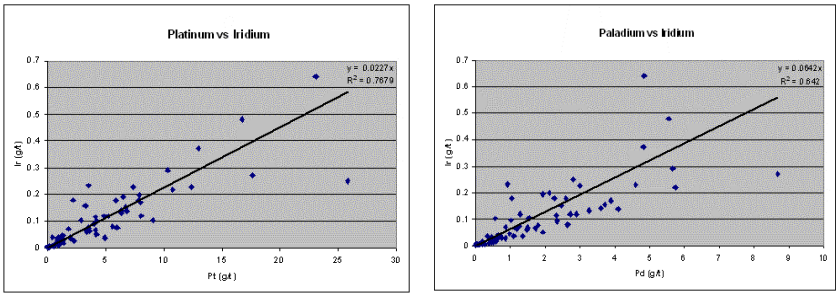
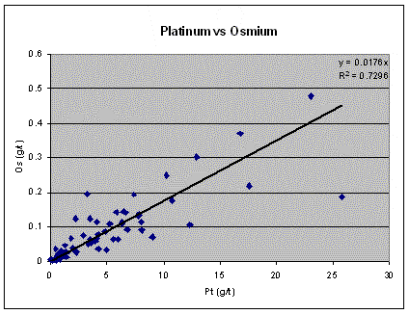
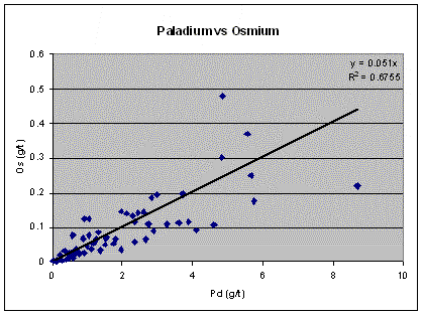


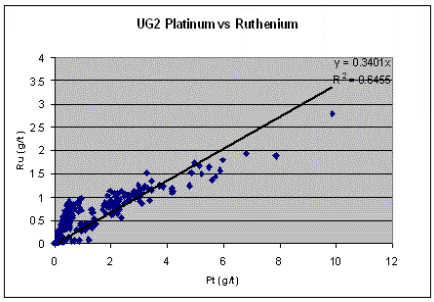
Figure 23 - Osmium Correlation Graphs (MR)





UG2 Reef

Figure 24 - Ruthenium Correlation Graphs (UG2)



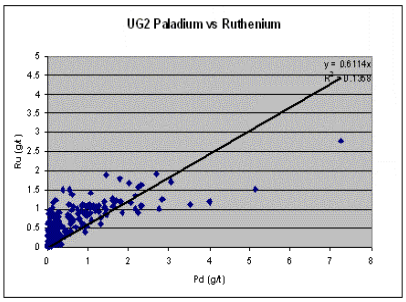
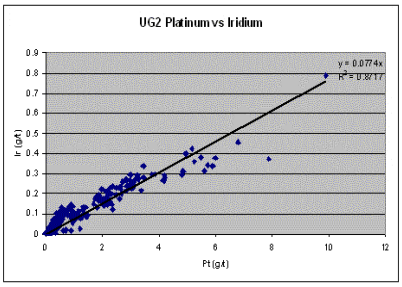


Figure 25 - Iridium Correlation Graphs (UG2)



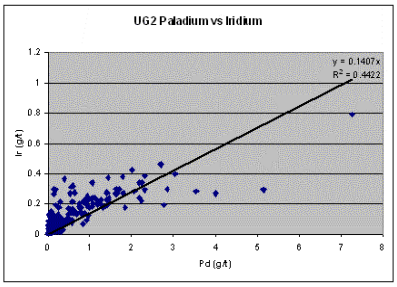
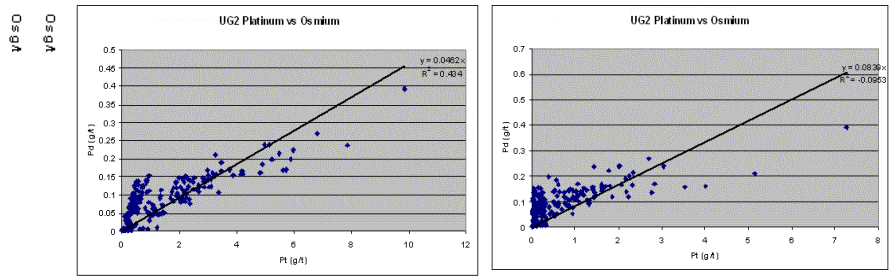


Figure 26 - Osmium Correlation Graphs (UG2)



Results

Using the regressions detailed above, the following grades have been calculated using an arithmetic mean for the minor elements (Ru, Ir, and Os). Whilst Osmium is of geological and technical interest, it does not have commercial value to WBJV Project 1. The Cu and Ni values were kriged:-

Table 50 - Average Potential Grades of Minor Elements in the Project Area

Reef	Cu (%)	Ni (%)	Os (g/t)	Ir (g/t)	Ru (g/t)
UG2	0.009	0.078	0.12	0.20	0.90
MR	0.090	0.200	0.08	0.14	0.57

Notes:
1. MR = Merensky Reef; UG2 = Upper Group No 2 chromitite seam.
2. A COG of 0 g/t has been used for the Os, Ir and Ru
3. A COG of 0% has been used for the Cu and Ni

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, and 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 RSA 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 RSA'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 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 2008 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 Anglo Platinum website includes the following points (Investment Analysts Report 11 March 2005 and the 2008 Annual Report):

- 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 up to 2,900,000 tonnes per year with a built-up head grade of up to 4.3g/t 4E in 2006.
- Mill recovery in 2004 was reported to be 85.83%. There is no new published recovery data.
- For 2008, the production was 175,000 equivalent refined platinum ounces compared to 193,600 ounces in 2007.
- Operating costs per ton milled are stated in Table 51 - all data from the Anglo Platinum annual report and website.

Table 51 - Bafokeng Rasimone Platinum Mine On-mine Cash Costs

Year	Milled (Mtonnes)	g/t 4E Head Grade	R/tonne
2002			R 284
2003			R 329
2004	2.76	4.5	R 372
2005	2.58	4.3	R 378
2006	2.89	4.3	R 385
2007	2.57	4.3	R 492
2008	2.25	4.4	R 630

The adjacent property to the north of the WBJV is Wesizwe **.** 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.

The Wesizwe annual report for the year ending July 2008 published by Wesizwe included a Resource declaration on the Merensky and the UG2 reef horizons. The statement was prepared in accordance with 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.

Table 52 - Mineral Resource - Wesizwe's Pilansberg Project

Reef	Category	Tonnes (Mt)	Grade 4E (g/t)	Content 4E (Moz))
Merensky	Measured	5.142	6.20	1.338
	Indicated	19.937	5.82	1.338
	Inferred	10.937	5.42	1.338
UG2	Indicated	24.788	4.63	1.338
	Inferred	17.652	4.20	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:

Table 53 - Estimated Mineral Resource - Styldrift Project (2005)

Merensky Reef			UG2	
Category	Tonnes(Mt)	Grade 4E (g/t)	Tonnes (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

The published Mineral Resource statement for the 50% attributable interest in Bafokeng Rasimone Platinum Mine, including Styldrift Project, is depicted in Table below.

Table 54 - BRPM including Styldrift Mineral Resource as at 2008

	Merensky		UG2	
	M tonnes	g/t 4E	M tonnes	g/t 4E
Measured	2.5	6.46	8.4	5.34
Indicated	9.6	6.76	38	5.29
Inferred	43.8	6.90	69.3	4.16
Total	**55.9**	**6.86**	**115.7**	**4.62**

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 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 Qualified Person 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

The metallurgical test work that has been conducted on the WBJV Project 1 ore body was initially carried out by SGS Lakefield at the pre-feasibility stage. MINTEK have completed a more extensive programme of test work for the Feasibility Study in a report dated 26 June 2009. Both SGS Lakefield and Mintek are based in Johannesburg, South Africa and have extensive experience with evaluating platinum projects from a metallurgical aspect.

Based on the metallurgical test work completed by SGS Lakefield for the Pre-Feasibility Study, GRD Minproc were commissioned to design and cost a MF2 (mill-float mill-float) circuit to treat the ores from the WBJV Project 1 deposit, either as pure Merensky, pure UG2 or a 20% blend of each in the plant feed. GRD Minproc updated this study financially for the Updated Feasibility Study Technical Report.

The process plant is to treat up to 140 000 tonnes of reef per month ‘name plate’ capacity through the facility, although the project will require the plant to treat up to 160,000 tonnes per month at a recovery discount.

The process plant is expected to be operated on a contract operations basis with a reputable process operator, of which there are a number now working in the platinum industry.

Item 18(a): Metallurgical Test Work

The metallurgical and mineralogical reports completed by SGS are available but are not attached to this Technical Report. The SGS work was based on individual core samples to determine the variability of the ore body.

Mintek were contracted to complete larger scale bulk evaluations on composite samples with rougher, cleaner and lock cycle flotation tests to be completed. The test work is complete at Mintek and the report is available but not attached to this Technical Report. Additional drilling and metallurgical test work is planned during 2010.

Source of Samples

The samples that were presented to the metallurgical laboratories were obtained from drill core only and from the primary hole and subsequent deflections from the primary hole.

The selected drill cores were from the different facies expected to be mined at WBJV at mining width, and were composited where possible on the initial mining areas. Subsequent revisions to the mining plan have shown that selected samples for metallurgical test work are no longer in the first 3 to 5 years of production and additional drilling will be required. Any future drilling is to be from the first 3 to 5 year production window.

No bulk sampling has been possible to obtain larger sized samples.

Predicted Plant Performance

The preliminary SGS test work indicated that the ‘name plate’ capacity metallurgical performance of the plant could achieve the ‘typical’ recoveries from operations within the Western Bushveld Complex of up to 87.5% for Merensky and 82.5% for UG2. The more detailed Mintek test work is supporting these recovery levels for both reef packages.

All available data was evaluated and modelled by Eurus Consulting and the following metallurgical performance is predicted for the concentrator at the finest grind tested of 90-95% passing 75 micron.

UG2 Reef:

Head Grade

3.71 g/t 4E

Concentrate Grade

165 g/t 4E

2.77% Chromite

Recovery

80.0% 4E

40% Cu

30% Ni

The average UG2 mill feed grade (after mine call factor deduction) is expected to be 3.40 g/t4E, which is comparable to the samples sent for metallurgical test work and thus the above results are considered appropriate for the financial modelling.

A final report from Mintek and the financial model has included an improved UG2 recovery at 82.5%, as this is supported by the Mintek investigation.

Merensky Reef:

Head Grade

2.67 g/t 4E

0.073% Cu

0.151% Ni

Concentrate Grade

- 137 g/t 4E
- 3.15% Cu
- 4.22% Ni
- <1.0% Chromite - assume 0.99%

Recovery

- 82.1% 4E
- 68.9% Cu
- 44.6% Ni

The average Merensky mill feed grade (after mine call factor deduction) is expected to be 5.45 g/t 4E which is **not** comparable to the samples sent for metallurgical test work and thus the above results are considered **inappropriate** for the financial modelling. The metallurgical core used for the test work is not regarded as representative of the ore body.

Additional test work will be conducted on new core to be obtained early in 2010, but there has been a consensus meeting where the metallurgical engineers from Mintek and Eurus Consultants have indicated, that assuming that the mineralogical characteristics and the flotation performance characteristics are similar to the low-grade sample evaluated, and then changed metallurgical performance **may** be expected.

There are neighbouring operating and future mines which are processing Merensky reef. The neighbouring BRPM mine had published in 2005 an achieved head grade of 4.47g/t delivered to the processing plant with a recovery of 85.83% for Merensky alone for the production year 2004. Prior to this production year recoveries were poor, as the UG2 open pit was in operation and this downgraded the recovery and is therefore not applicable to the only Merensky operation.

The Wesizwe project has a predicted recovery for Merensky of 88%, although with a number of cautionary statements.

The BRPM published results are regarded as an indication of the concentrator performance that can be achieved. The anticipated head grade to be delivered to the WBJV mine will be 5.45g/t (after MCF correction). Analysing the BRPM performance, the tailings value will be approximately 0.65 g/t. There is a strong relationship between head grade and recovery on platinum operations such that the tailings values remain relatively constant - this means that as the head grade increases, the recovery increases. Applying this metallurgical knowledge to the WBJV Merensky feed, it can be expected that the recovery will be between 87 and 88% with tailings values of approximately 0.73 and 0.67 g/t. The Mintek test work has indicated that an even lower tailings value may be achieved, thus offering upside recovery potential.

On this basis and considering that the samples evaluated at Mintek are not representative of the ore body, it is proposed that the Merensky recovery that may be achieved will be 87.5%. This is the value that was used in the Pre-Feasibility Study and the Feasibility Study. It must be noted that this recovery will not be achieved during periods of lower head grade. This evaluation also assumes that the mineralogical and flotation performance of the different mill feeds is very similar.

Thus, the following **Merensky** performance has been used in the financial model.

Head Grade

- 5.45 g/t 4E (estimated)
- 0.073% Cu
- 0.151% Ni

Concentrate Grade

- 150 g/t 4E
- 3.15% Cu
- 4.22% Ni
- <1.0% Chromite - assume 0.99%

Recovery

- 87.5% 4E
- 68.9% Cu
- 44.6% Ni

This plant performance will be tested during the future metallurgical test work programme.

Plant Performance in excess of 'Name Plate' Capacity

The predicted plant performance during the period when the throughput exceeds to 'name plate' capacity is indicated in Table 55 .

Table 55 - Predicted Plant Recovery in excess of 'name plate' capacity

Tonnage		4E Recovery		Cu Recovery		Ni Recovery	
per Month	per Annum	MR	UG2	MR	UG2	MR	UG2
130,000	1,560,000	87.50%	82.50%	68.90%	40.00%	44.60%	30.00%
135,000	1,620,000	87.50%	82.50%	68.90%	40.00%	44.60%	30.00%
140,000	1,680,000	87.50%	82.50%	68.90%	40.00%	44.60%	30.00%
145,000	1,740,000	87.00%	82.00%	68.50%	39.60%	44.25%	29.65%
150,000	1,800,000	86.50%	81.50%	68.10%	39.20%	43.90%	29.30%
155,000	1,860,000	86.00%	81.00%	67.70%	38.80%	43.55%	28.95%
160,000	1,920,000	85.50%	80.50%	67.30%	38.40%	43.20%	28.60%
165,000	1,980,000	85.00%	80.00%	66.90%	38.00%	42.85%	28.25%

The reduction in process plant recovery is based on the Qualified Persons experience and has not been directly supported by test work. A high-level review of the metallurgical test work completed at Mintek, indicated that the recovery predictions are likely to be conservative and the performance will be bettered.

In addition, a high-level economic review was also completed whereby the processing plant capital was increased by up to 15% with the design recovery being maintained during the life of the mine. This indicated that there is a financial advantage in ensuring that the best possible plant recovery is maintained for the life of the mine, even with the increased capital expenditure. This analysis will be further developed, before the plant design is finalised prior to plant construction. The Updated Feasibility Study has not included this increased capital expenditure, but has included the recovery decrease as throughput increases.

Ongoing Testwork

As previously stated, the Mintek test work has been completed and the final metallurgical report is available.

Future Testwork

Because of the drill core used for the metallurgical test work not being in the first 3 to 5 year mining plan, additional drilling will be conducted in the six areas that are to be mined during the first 3 years of production. During this time, no UG2 will be mined or treated and thus the drilling will be based on recovery of Merensky reef only (this is in the shallow region of the mine and UG2 is poorly developed).

The additional metallurgical drilling will make available up to 50kg of core from each of the six mining areas for geological, geotechnical and metallurgical evaluation. This drilling is expected to be conducted early in 2010 with the metallurgical evaluation completed by late 2010.

The current mining programme has first Merensky ore being produced late in 2011. It is proposed that this ore be evaluated in a pilot plant may be conducted at Mintek late in 2011.

The results from this additional metallurgical test work will be used to review the appropriateness of the process plant design and to optimize the flow sheet as required.

Costs

A provision has been made in the Capital Estimate for the additional drilling of the initial mining areas for the first 3 to 5 years of production of R4m based on 54 intersections (including deflections) of Merensky reef at an average depth of 175m. This will include drill core selection and preparation for metallurgical testing. The information generated from this drilling will also be used for additional geological and geotechnical evaluation and be incorporated into the mine-planning model.

The metallurgical testing of the drill core will be approximately R1.5m and will be based on variability work and composite testing of the different mining areas. This is based on verbal discussion with Mintek and is comparable to the work being conducted at present.

As suggested above, a pilot campaign is to be conducted on the first Merensky reef produced from the mine. There are to be two campaigns conducted at Mintek and these will cost R2.5m each. A provision has been made for R5.0m in the Capital Estimate.

Ultra-Fine Grinding

Ultra-fine grinding of flotation intermediate products is being considered by many of the operating platinum producers to improve overall recovery or final concentrate grade or both. This aspect has not been considered for the WBJV concentrator during the Updated Feasibility Study phase of the project. This aspect may be considered during the future pilot campaign but probably only once production has commenced.

No costing has been included in the capital nor operating costs for ultra-fine grinding as the technology is 'too' new to consider for the initial mine. The associated equipment can be readily retrofitted to the concentrator if there are economic advantages following successful test work.

Item 18(b): Prill Splits

The financial procedure, which has been used to determine the value of the ore body, is based primarily on analysis of contained Platinum, Palladium, Rhodium and Gold (commonly referred to as the 4E's). Other elements of economic interest (Iridium, Ruthenium, Copper and Nickel) were determined on a number of core samples but not on all. A regression analysis was developed, based on these determinations and the results of this regression have been used to approximate the contained Iridium and Ruthenium. Copper and nickel have been modelled and a value in the mill feed has been determined, but not fully modelled and developed via the detailed mine planning exercise.

The more detailed explanation of how the regressions were completed and the copper and nickel values determined is detailed in Item 15(d).

The geological regression results have indicated the following metal splits to be used for the WBJV Project.

Table 56 - Metal Splits in Ore (assumed in Concentrate)

Merensky Mining Area Metal Splits								
	Pt	Pd	Rh	Au	4E's	Ir	Ru	6E's
MR - 4E's	64.00%	27.00%	4.00%	5.00%	100.00%	2.20%	8.95%	111.15%
MR - 6E's	57.58%	24.29%	3.60%	4.50%	89.97%	1.98%	8.05%	100.00%

UG2 Mining Area Metal Splits								
	Pt	Pd	Rh	Au	4E's	Ir	Ru	6E's
MR - 4E's	63.00%	26.00%	10.00%	1.00%	100.00%	5.18%	22.90%	128.08%
MR - 6E's	49.19%	20.30%	7.81%	0.78%	78.08%	4.04%	17.88%	100.00%

The above splits have been used in the financial model to determine the revenue generated by the mine.

This spilt has been based on evaluation of the drill core for each of the elements. It is assumed that the same split is carried forward to the concentrate, but this has not been confirmed, as there has not been a representative concentrate produced during the test work at Mintek. This will be confirmed during future test work.

The geological model has regressed values for Copper and Nickel as indicated below.

Table 57 - Cu & Ni values for different reefs

Reef	Cu (%)	Ni (%)
UG2	0.009	0.078
MR	0.090	0.200

These have not been modelled geologically nor scheduled through the mine planning process. Applying a mining dilution to these anticipated values of 15% plus a Mine Call Factor of 98% is a reasonable way of evaluating the potential head grade to the concentrator. This is indicated in the following table.

Table 58 - Summary of Cu & Ni Values in Plant feed

MR	Cu	Ni
Resource	0.090%	0.200%
Dilution	15%	15%
Mined Grade	0.077%	0.170%
MCF	98%	98%
Plant Feed	0.075%	0.167%

UG2	Cu	Ni
Resource	0.009%	0.078%
Dilution	15%	15%
Mined Grade	0.008%	0.066%
MCF	98%	98%
Plant Feed	0.007%	0.065%

These values have been used in the financial model and are comparable to the samples used for metallurgical test work and are thus accepted as a reasonable indication of the plant feed grade.

Item 18(c): Process Plant

The concentrator design was based on the metallurgical test work completed by SGS Lakefield plus industry experience and is not based on the more appropriate Mintek evaluations. This decision was taken by the project team in the interest of time saving for the project and is regarded as a reasonable approach with the optimization of the flow sheet to be implemented during the detailed design phase of the concentrator.

GRD Minproc were contracted to complete the plant design and for the Updated Feasibility Study, review the capital and operating cost estimates. The process plant design and operating philosophy was not reviewed during the Updated Feasibility Study.

The original plant was based on being fed from a single underground source of reef only with production waste being re-directed at the tipping point. The subsequent mining redesign has extended the plant to accept conveyed reef and waste from underground, distribute the waste to surface bins for discard to the waste dump, crush the underground conveyed reef, accept truck tipped reef from underground and to control the blending of surface stockpiles that will be created during operations.

Operating Philosophy

The concentrator is designed to treat 140,000 tonnes per month of reef (‘name plate’ capacity) with a possible 20% controlled blend of UG2 in Merensky reef or vice versa. The basis of the plant design is that the milling capacity of the plant can process the full 140,000 tpm of Merensky and the flotation plant can treat the full capacity for UG2. The milling power required for MR is higher than that required for UG2, whilst the flotation circuit residence time and other parameters required processing UG2 is greater than for MR and this has been included. This aspect will need to be reviewed to cater for mass-pull and other factors during final process design. The plant will be required to treat up to 160,000 tonnes per month, which is 14% in excess of the ‘name plate’ capacity, albeit at a recovery discount.

This plant is designed to be versatile with two feed bins so that a controlled blend of ore can be processed.

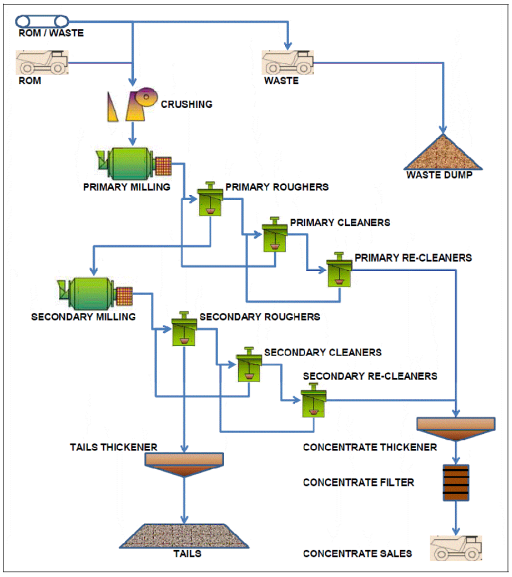
Process Flow Description

The following figure outlines the block flow diagrams for the process flow. The process flow diagrams (PFD’s) and piping and instrumentation diagrams (P&ID’s) are available in the GRD Minproc Process Plant report.

Run of Mine (RoM) material is received from either dump truck or the underground conveyor delivery chute. Dump trucks from the two decline systems will either tip directly into the crusher feed bin or onto the surface stockpile area - stockpiled material will be double handled and reclaimed into the process plant by front-end loader. Material delivered by conveyor will be identified as waste, Merensky reef or UG2 reef. Control systems are to be incorporated such that the three products are not mixed or directed to the incorrect product area.

Waste will be deflected from the conveyor head chute to the waste conveyor and into a storage bin for road transport to the waste dump, (future conveying of this product could be considered).

Figure 27 - Plant Block Flow Diagram



Reef will be directed into the crusher feed bin and the associated pan feeder. The RoM is conveyed to a vibrating grizzly feeder (VGF), from where the oversize is fed to a jaw crusher.

The crusher product combines with the VGF undersize from where the material is conveyed to a reversible conveyor feeding two storage silos allowing UG2 and Merensky ore to be separately stored.

Belt feeders withdraw material separately from each silo. Blending of the two ore types takes place on a conveyor feeding the crushed material to the primary mill surge bin. On the same conveyor a holding bin and vibrating pan feeder is located allowing scats from the primary mill discharge trommel to be returned to the milling circuit.

Material is fed to the primary mill from the mill surge bin by a variable speed belt feeder. The primary milling circuit comprises a grate discharge RoM ball mill with a trommel in closed circuit with hydrocyclones. The cyclone overflow gravitates to the primary roughers flotation circuit. The cyclone underflow gravitates back to the primary mill feed chute.

No allowance has been made for classification of the cyclone overflow, since misplaced UG2 silicates reporting to the cyclone overflow, will be further processed through the secondary milling circuit.

The primary flotation circuit comprises a train of cylindrical tank type, forced air rougher flotation cells operating in series and with gravity flow. Rougher concentrate is cleaned in two stages of flotation and the final concentrate is pumped to the concentrate thickener.

Rougher cleaner tails are returned to the primary mill discharge sump.

Rougher tails are collected and pumped to the secondary mill feed de-watering and classification cyclones. Primary roughers tails will be de-watered and classified in two stages of hydrocyclones before being fed to the secondary mill.

In the first stage cyclone, the higher density chromites will tend to preferentially classify to the underflow, whereas the silicates will report to the overflow. A second stage cyclone will be installed to recover coarse silicates misplaced in the first stage cyclone. In this manner, these misplaced course silicates report back to the secondary mill for fine grinding.

The secondary mill is an overflow ball mill with a trommel and operates in open circuit. The ball mill discharge gravitates to the secondary mill discharge sump and is pumped to the secondary roughers flotation circuit, which is comprised of a train of cylindrical tank type with forced air rougher flotation cells operating in series and with gravity flow.

The secondary rougher concentrate is cleaned in two stages of flotation and the final concentrate is pumped to the concentrate thickener.

Secondary cleaner tails are returned to secondary mill discharge sump.

Secondary rougher tails are collected and pumped to the tails handling circuit before being discarded to the tails storage facility.

Final concentrates from the primary and secondary rougher flotation cleaner circuits are pumped to the concentrate thickener via a trash screen, to prevent foreign trash ingress into the concentrate thickening and filtration circuit.

Thickener underflow, is pumped to an agitated filter feed tank and from the feed tank to the filtration circuit, from where the air dried filter cake discharges onto a conveyor and then onto one of two concentrate stockpiles for loading and dispatch. The filtrate is recycled to the concentrate thickener.

Thickener overflow gravitates to a collection tank and is pumped to the flotation circuits for use as launder wash down water.

Flotation plant tails from the secondary roughers is pumped to the tails thickener via a guard cyclone that prevents coarse and tramp material from entering the thickener.

Thickener underflow together with guard cyclone underflow is pumped to an agitated tails disposal tank from where it is pumped to the tails storage facility.

Thickener overflow gravitates to the process water storage pond and reticulation system.

Tails from the flotation plant is delivered to the tails storage facility (designed by others). Decant water gravitates to the tails return water pond from where it is pumped back to the process water pond and reticulation system.

In order to cater for all eventualities, i.e. treating Merensky ore or UG2 ore separately, or a Merensky/UG2 blend, various reagent make-up and flocculant make-up systems have been provided for in the estimate.

Plant Capital Cost

Outlined below is a summary of the updated capital costs for the processing plant only, and excludes capital costs associated with mining operations and the mine residue disposal facilities.

The basis of the updated capital cost estimate is an EPCM contract, which has been estimated at a base date of 01 September 2009 and is detailed in Addendum 2 to the Process Plant Report.

The total installed capital cost has been estimated at an accuracy of +15%- 15%.

An overall accuracy provision of 7.7% and contingency of 7%, of direct field costs, have been included. No growth allowance or escalation has been allowed for.

All estimates have been prepared based on the currency in which tendered and where applicable converted to South African Rand (ZAR) at the following exchange rates:

USD 1 = ZAR8.00
Euro 1 = ZAR12.0

The total process plant capital cost including; supply, transportation, erection, installation, mechanical completion, dry commissioning, first fills, vendor commissioning assistance, commissioning spares, accuracy provision, EPCM, contingency and strategic/insurance spare costs is outlined in following table.

Table 59 - Updated Process Plant Capital Cost estimate

Discipline	% of Mechanicals	ZAR Cost
Earthworks	9.22%	24 770 713
Civils	13.89%	37 330 193
Structural	12.11%	32 534 198
Platework	5.27%	14 175 527
Mechanical	100	268 749 605
Piping and Valves	10.79%	28 996 002
Electrical and Instrumentation	19.11%	51 360 464
Buildings	1.85%	4 969 604
Miscellaneous	4.53%	12 177 273
Direct Costs		**441 039 170**
Accuracy Provision	7.71% of Direct Cost	34 024 413
Total Direct Costs		**475 063 583**
	% of Direct Costs	
EPCM	14.87%	70 655 223
Contingency	7%	33 254 450
Strategic/Insurance spares		16 386 597
Contract Plant Operator Start Capital		1 189 849
Total Indirect Costs		**121 486 121**
TOTAL PROJECT COST		**596 549 704**

The above estimate includes costs for tailings dam pipeline and return water pipeline and pumps, with associated electrical infrastructure, as per the GRD Minproc report.

This updated capital costs estimate has been included in the overall capital budget for the WBJV mine.

The updated capital cost estimate has increased by 8.3% from the Feasibility Study plant base date of 01 December 2007.

Contingency

The process plant contingency as determined by GRD Minproc has been included in the capital estimate, and an additional provision has been made for possible redesign of the flow sheet subject to the additional test work discussed above.

Plant Operating Cost

This is a summary of the operating costs for the processing plant only, and excludes operating costs associated with mining operations and operation of the mine residue disposal facilities.

The operating costs have been estimated for the process plant at a base date of 01 September 2009 and are detailed in the Process Plant Report.

The operating cost has been estimated at an accuracy of ±15%.

An overall accuracy provision of 7.5% has been included. No growth allowance, escalation or contingency has been provided for.

All estimates have been prepared based on the currency in which tendered and where applicable converted to South African Rand at the following exchange rates:

USD 1 = ZAR8.00
Euro 1 = ZAR12.00

The following table summarises the estimated operating costs.

Table 60 - Summary of Plant Operating Cost

Cost Item	ZAR/annum
Concentrator fixed costs	
Labour	18 882 482
Maintenance spares	15 769 958
Miscellaneous costs	1 591 788
Subtotal	**36 244 228**
Concentrator variable costs	
Power	28 127 401
Plant vehicles	7 187 440
Consumables	45 452 078
Subtotal	**80 766 919**
Accuracy Provision (7.5 %)	**8 775 836**
TOTAL ANNUAL OPERATING COST (including accuracy provision)	**125 786 983**

The financial model has included the fixed costs at R36 244 228 per annum plus the variable costs at R32.70 per tonne milled (excluding power, water and analytical costs). The fixed costs have increased by 14.1% and the variable costs (excluding power, water and analytical costs) have increased by 32.6% since 01 December 2007.

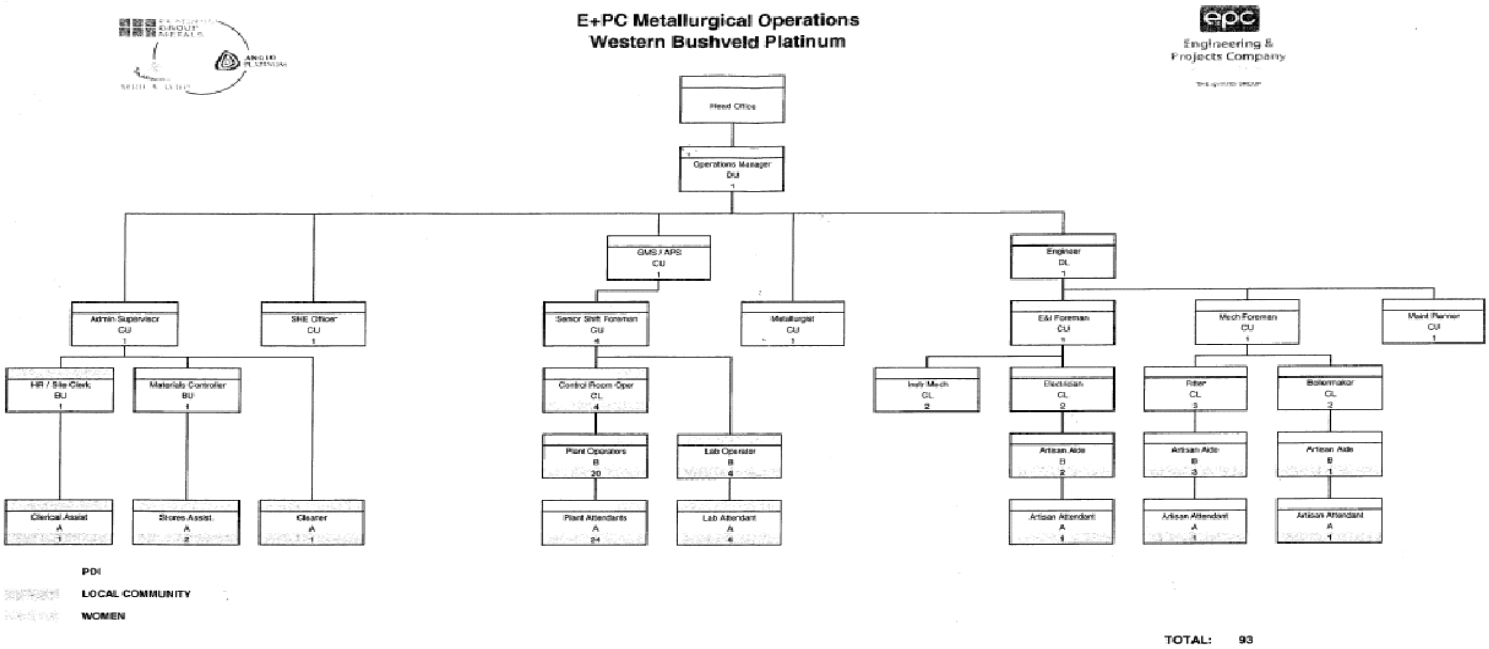
The operating cost power demand is estimated at 10.5MW, which is equal to 50kWhr/tonne treated. This estimate has been removed from the plant operating costs and is stated as a line item in the financial model to allow the effects of the power supply issues to be reflected correctly.

The estimated water demand by GRD Minproc is 0.77kilolitres per tonne treated. This estimate has been removed from the plant operating costs and is stated as a line item in the financial model to indicate the cost of services.

Plant Labour

The following staffing organogram is expected to be implemented by the designated contactor operating the plant.

Figure 28 - Expected Plant Labour Structure



The total staff compliment is to be 93 persons.

Item 18(d): Plant Performance

Process Tonnage

The process plant has been designed to treat 140,000 tonnes per month of reef (as ‘name plate’ capacity) with a controlled blend of up to 20% of UG2 in MR or vice-versa. The plant will be required to treat up to 160,000 tonnes per month as a result of the revised mine plan associated with this Updated Feasibility Study. The processing plant will be able to treat the additional 14% but at the expense of metal recovery, as indicated in Table 55 .

The tonnage and grade profiles as detailed from the mine planning are displayed in the following table - this indicates that there is a change over period between the MR operation and the UG2 operation of about 6 years. During this time, there is a period of 2 years where the reef split exceeds the 20:80 ratio upon which the plant design is based and during this period, ore will need to be stockpiled and re-handled to allow efficient plant operation.

The expected life of mine treatment profile is depicted in the following table.

Table 61 - Milling Production Profile

	Merensky		UG2		To Surface		Mill Feed	
Production Year	Tonnes	Grade g/t	Tonnes	Grade g/t	Tonnes	Grade g/t	Tonnes	Grade g/t
-2	-		-					
-1	12,046	3.48			12,046	3.48	0	
1	224,730	5.17			224,730	5.17	0	
2	899,361	6.16			899,361	6.16	1,135,000	5.94
3	1,662,798	6.30			1,662,798	6.30	1,660,000	6.30
4	1,961,055	5.61			1,961,055	5.61	1,960,000	5.61
5	1,938,830	5.00			1,938,830	5.00	1,940,000	5.00
6	1,934,535	5.36			1,934,535	5.36	1,935,000	5.36
7	1,840,915	5.27			1,840,915	5.27	1,840,000	5.27
8	1,902,559	5.24			1,902,559	5.24	1,905,000	5.24
9	1,808,588	5.43	1,075	3.33	1,809,663	5.43	1,810,000	5.43
10	1,758,302	5.90	116,051	3.63	1,874,353	5.76	1,870,000	5.76
11	1,224,934	5.61	457,090	3.53	1,682,024	5.04	1,680,000	5.05
12	679,346	5.17	1,161,474	3.47	1,840,820	4.10	1,845,000	4.10
13	148,843	4.04	1,685,311	3.49	1,834,154	3.54	1,835,000	3.54
14	13,755	3.06	1,879,975	3.44	1,893,730	3.44	1,895,000	3.44
15			1,901,011	3.46	1,901,011	3.46	1,900,000	3.46
16			1,895,983	3.31	1,895,983	3.31	1,895,000	3.31
17			1,795,050	3.35	1,795,050	3.35	1,795,000	3.35
18			1,604,386	3.28	1,604,386	3.28	1,605,000	3.28
19			728,442	3.26	728,442	3.26	730,000	3.26
20			296,993	3.31	296,993	3.31	295,000	3.31
21			12,496	3.45	12,496	3.45	15,934	3.42
Grand Total	18,010,598	5.51	13,535,336	3.40	31,545,934	4.60	31,545,934	4.60

During operations, it is expected that there will be periods when surface stockpiling of reef will be necessary and the working costs have taken this into account with a re-handling provision of R4.00 per tonne included.

Recovery and Grade Performance

The metallurgical test work being conducted at Mintek has been completed and the expected recovery and grade performances have been confirmed by test work for the plant operating at ‘name plate’ capacity. This performance is expected to be as detailed below at ‘name plate’ capacity.

Merensky:

Concentrate Grade

150 g/t 4E

3.15% Cu

4.22% Ni

<1.0% Chromite - assume 0.99%

Recovery

87.5% 4E

68.9% Cu

44.6% Ni

UG2:

Concentrate Grade

165 g/t 4E

2.77% Chromite

Recovery

82.5% 4E

40% Cu

30% Ni

The data generated from the test work indicates that this performance can be expected from the concentrator once operations have stabilized at ‘name plate’ capacity.

As the plant will be required to exceed the ‘name plate’ capacity, the predicted performance at different levels of throughput are detailed in Table 55 .

There is some uncertainty with the predicted metallurgical performance as indicated above due to the non-representative nature of the samples that were tested and this inappropriate data generated. As a result, additional drilling and metallurgical test, work has been scheduled, including a pilot plant campaign. In addition, the financial analysis has included a sensitivity analysis around lower plant recoveries.

Concentrate Moisture

The use of Larox pressure filters for drying concentrate will ensure that the concentrate moisture will be lower than the penalty limit of 15%. Whilst test work has not been conducted on the final concentrate, the moisture content of Merensky concentrate is expected to be 15% as per industry norms on such concentrate.

Chromite Recovery

The recovery of Chromite has not been included in the process flow sheet as it will only be applicable when the UG2 reef is processed and this can be retrofitted if deemed economically attractive after production year 11.

Item 18(e): Commissioning

The concentrator will be commissioned on stockpiled material from the early mining of Merensky reef and this stockpile will be depleted during this phase.

Item 18(f): Production Ramp-Up

The processing ramp-up in production from a tonnage perspective is not regarded as critical area, as processing will be determined by the mining rate which in the first number of months, will be significantly less than the potential processing rate for the concentrator.

The stabilisation of the concentrator will be required during the first number of months until there is sufficient ore for extended operations. This could result in reduced concentrate grade and recovery, but this aspect has not been factored into the financial model.

Item 18(g): Concentrate Transportation

The concentrate produced on the mine will be required to be delivered to the smelting facility of Anglo Platinum based in Rustenburg. A local transport contractor has been approached and an indicative cost for such delivery is stated as R48.00 per wet tonne. This translates to approximately R1.55 per tonne kilometre, which is regarded as fair and reasonable. This cost has not been escalated since the Feasibility Study.

Item 18(h): Concentrate Off-take Agreement

A draft concentrate off-take agreement is in place between PTM, Wesizwe and Anglo Platinum and this forms part of the Joint Venture agreement. This draft agreement defines the outlines of the treatment contract that will be applied to the WBJV Project 1 mine.

The basic conditions are as indicated below:

- Payable metals in concentrate shall be
  - Platinum - Pt
  - Palladium - Pd
  - Rhodium - Rh
  - Gold - Au
  - Ruthenium - Ru
  - Iridium - Ir
  - Copper - Cu
  - Nickel - Ni

- Penalties shall be applied for the following:
  - Chromitite content above a minimum limit - stated as 1% with a sliding scale
  - Moisture content above a set limit - estimated to be 15%
  - Minimum concentrate 4E's grade - estimated to be 150 g/t
- Charges shall be applied on the following basis
  - Treatment charge per dry tonne received
  - Sampling charges per batch received
- Payment terms shall be
  - Three months after delivery of concentrate

The final terms and condition have not been negotiated between the parties involved but indicative terms have been presented by PTM RSA to Anglo Platinum for comment.

The terms that have been used in the financial model, although **not ratified** by the partners are:

- Payability of all metals - 86%
- Treatment Charges - R500 per dry tonne of concentrate
- Sampling Charges - R2 500 per batch of 200 tonnes receive (1.5 samples per 300 tonnes)
- Moisture penalty - R30 per tonne of contained water above 15%
- Concentrate grade penalty - R50 per 5g/t below threshold of 150 g/t
- Chromite penalty - sliding scale up to R75 000 per tonne contained chromite at 5%. The sliding penalty scale is based on the following table

Table 62 - Chromite Penalty Estimate

Chromitite Penalty Table	
Contained Cr_2O_3 per tonne of concentrate	Penalty per Contained Cr_2O_3 per tonne of concentrate
1.0% to 1.5%	12 000
1.5% to 2.0%	13 200
2.0% to 2.5%	14 600
2.5% to 3.0%	16 100
3.0% to 4.0%	32 200
4.0% to 5.0%	50 000

These smelter terms and conditions have not been updated since the Feasibility Study.

Item 18(i): Analytical & Assay

There will not be an Analytical Laboratory constructed at the WBJV mine site.

Provision has been made for sample preparation facilities at the plant for metallurgical production samples, whilst the mine geologists at the existing facility at the farmhouse will prepare the geological and grade control samples.

Samples will be collected by a commercial laboratory located in the Rustenburg district. An allowance of R2.00 per tonne milled has been made for analytical services in the working cost model for the concentrator plus R3.00 per tonne milled for ore Resource definition and grade control.

No discussions have been held with such a laboratory, but there are a number of independent facilities available.

Item 18(j): Tailings Dam

The tailings dam is located to the east of the project site and to the south of the drainage channel. As such, the access from the Concentrator to the TSF will be directly by road under the ESKOM power lines.

The facility was originally designed for 25 million tonnes but following a number of reviews, this capacity has been increased to 32 million tonnes without changes to the current design.

Design Philosophy

The depositional characteristics of the platinum tailings are based on the physical characteristics and actual particle size distribution of the tailings, which influence:

- Tailings behaviour on deposition,
- Beach formation and profile,
- Rate of drying out, desiccation/consolidation and associated strength gain,
- Particle segregation along the beach,
- Pool control, and
- General tailings dam operations and deposition practices.

For the selected depositional methodology (conventional thickened spigotted self raising), tailings are deposited into the TD basin via a spigot pipeline located on the inner crest of the perimeter starter wall. During commissioning and deposition behind the starter wall the tailings is directed to the base of the inner toe of the starter wall and adjacent to the toe drain and inner blanket drain by flexible hoses. Deposition during this stage of the operation is carefully controlled, monitored and intensely managed, to ensure that the drains are not eroded away by the tailings stream or blinded by depositing “fine” tailings material over them.

Once the tailings has reached the elevation of the starter wall, the forming of ±0.5m high tailings bund walls along the perimeter crest of the starter wall using the deposited tailings and the raising of the spigot pipeline commences - i.e. routine tailings dam operation, depositing in cycles around the TD.

Supernatant and storm water, collected on the tailings dam, is decanted via vertical penstock intakes and a buried penstock pipeline discharging into the RWD/SWD.

During commissioning and the initial development stages of the TD, decanting occurs via temporary/intermediate single penstock inlets, located along the migration path of the pool from the starter wall uphill towards its final location at the double penstock inlets and pool wall. The intermediate inlets are progressively sealed once the pool has migrated to an adjacent upstream inlet.

The tailings dam has been designed as a conventional tailings dam impoundment and is not based on paste technology. Water recovered from the tailings facility will be returned to the processing plant.

The dam is to be rock clad for closure considerations.

Capital Cost

The tailings dam capital cost has been re-estimated by Epoch as part of the Updated Feasibility Study. The Tailings Dam report has not been updated.

Schedules of quantities, for the preparatory works associated with the MRDF, have been reviewed by Epoch and priced using recent budgetary construction rates obtained from a reputable earthworks contractor.

Table 63 - Tailings Dam Capital Cost

Facility	Capital cost (RSA Rands)
Tailings Dam	34,170,010
Return Water Dam & Storm Water Dam	14,973,313
Waste Rock Dump (estimate)	6,000,000
Unmeasured Items (15% of works)	7,371,498
Sub Total	62,514,821
P&G’s (25% of total cost of works)	13,984,677
Total (+25%-15% accuracy)	**76,499,498**

The above capital cost estimate excludes the following, all of which have been catered for in the GRD Minproc or in the project costing:

- The slurry delivery pipeline(s) from the process plant to the TD;
- Access roads between the TD and process plant, shaft;
- The RWD and SWD pump stations;
- Return water pipelines from the RWD and SWD the process plant; and
- All electrical, mechanical and instrumentation equipment.

This capital cost estimate has increased by 46% since the 2008 Feasibility Study as a result of escalation and improved project definition.

Operating Cost

The tailings dam operating costs are based on an annual fixed charge of R500 000 to include the salary of operations staff and consultant reviews plus a variable charge of R0.65 per tonne placed have remained unchanged from the Feasibility Study. These have been included in the financial model.

It is anticipated that the total staffing of the tailings dam by the contractor will be 15 persons.

ITEM 19: MINERAL RESOURCE ESTIMATES

Item 19(a): Standard Reserve and Resource reporting system

Minxcon (Pty) Ltd ("Minxcon") was commissioned by the directors of Platinum Group to undertake the Mineral Resource estimation of the WBJV Project 1 and 1A area. Mr Charles Muller of Minxcon has been closely involved with the Project, in the capacity of an independent Competent Person, since its inception. The scope of this report is to outline and detail the methodology employed of Mineral Resource estimation of Project 1 and 1A of the WBJV, as well as detailing the results of the Mineral Resource estimation exercise. The effective date of the Mineral Resource Estimation is July 2008.

An updated Resource model has been generated and press released with the Updated Feasibility Study but this Updated Resource was NOT used in the Updated Feasibility Study Technical Report.

Item 19(b): Data Sources

The independent author and Qualified Persons (QP) of this report have used the data provided by the representative and internal experts of PTM RSA. This data is derived from historical records for the area as well as information currently compiled by the operating company, which is PTM RSA. The PTM RSA -generated information is under the control and care of Mr WJ Visser Pr.Sci.Nat. 400279/04, who is an employee of PTM RSA and is not independent. The following table summarises the available data that was used for the geostatistical modelling and evaluation of the project:-

Table 64 - Borehole Data

Data	Merensky Reef	UG2 Reef
No. of Borehole intersections Project 1	237	231
No. of Borehole intersections project 1A	68	32

The locations of the boreholes are illustrated in the following figures:-

Figure 29 - Location of MR Boreholes

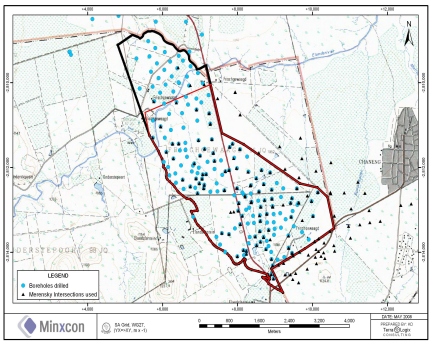
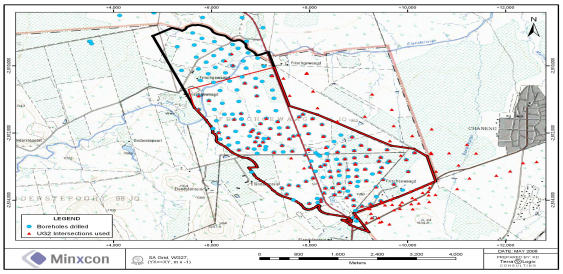


Figure 30 - Location of UG2 Boreholes



Item 19(c): Classical Statistics

Statistical analyses were performed to develop an understanding of the statistical characteristics and sample population distribution relationships. Descriptive statistics in the form of histograms (frequency distributions) and probability plots (used to evaluate the normality of the distribution of a variable) were used to develop an understanding of such statistical relationships. Skewness is a measure of the deviation of the distribution from symmetry (0 - no Skewness). Kurtosis measures the "peakedness" of a distribution (0 - normal distribution).

The descriptive statistics tables are illustrated per Geozone. The Merensky Reef has been subdivided into 8 Geozones. The Geozones were determined using the geological facies plans and grade distribution for both the Merensky and UG2 reefs. The descriptive statistics tables are illustrated per Geozone.

Table 66 - summarise the descriptive statistics for the Merensky Reef.

Table 65 - Merensky Reef Descriptive Statistics Project 1 (min 80 cm cut)

Merensky Reef Domain 1

Variable	Descriptive Statistics (Spreadsheet1)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRMC_CW	83	106.9760	80.19059	265.520	1356.9	36.8357	2.142921	4.98273
DOM1_ALL_MRMC_CM3PG	83	795.9884	16.06004	5615.966	715133.3	845.6555	3.007548	12.71986
DOM1_ALL_MRMC_SG	83	3.2317	3.06000	3.518	0.0	0.0856	0.783957	0.66095
Ln_DOM1_ALL_MRMC_CW	83	4.6285	4.38441	5.582	0.1	0.2800	1.436316	1.57080
Ln_DOM1_ALL_MRMC_CM3PG	83	6.2408	2.77633	8.633	1.0	1.0179	-0.716869	1.61327
Ln_DOM1_ALL_MRMC_SG	83	1.1727	1.11841	1.258	0.0	0.0263	0.708194	0.44205

Merensky Reef Domain 2

Variable	Descriptive Statistics (Spreadsheet3)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2_ALL_MRMC_CW	7	84.50885	83.44655	90.88267	7.8994	2.81059	2.64575	7.000000
DOM2_ALL_MRMC_CM3PG	7	20.82296	3.86816	63.40216	685.7113	26.18609	1.26434	-0.587493
DOM2_ALL_MRMC_SG	7	3.16851	2.85441	3.31678	0.0270	0.16428	-1.31700	1.411725
Ln_DOM2_ALL_MRMC_CW	7	4.43640	4.42421	4.50957	0.0010	0.03226	2.64575	7.000000
Ln_DOM2_ALL_MRMC_CM3PG	7	2.37556	1.35278	4.14950	1.3861	1.17732	1.11371	-0.877684
Ln_DOM2_ALL_MRMC_SG	7	1.15207	1.04887	1.19899	0.0028	0.05325	-1.39439	1.717447

Merensky Reef Domain 3

Variable	Descriptive Statistics (Spreadsheet5)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM3_ALL_MRMC_CW	8	84.91952	80.19059	85.5951	3.651	1.91078	-2.82843	8.00000
DOM3_ALL_MRMC_CM3PG	8	93.24850	18.49183	181.9987	3377.726	58.11821	0.25402	-1.35884
DOM3_ALL_MRMC_SG	8	3.20300	3.05033	3.3266	0.010	0.10167	-0.34776	-0.94332
Ln_DOM3_ALL_MRMC_CW	8	4.44148	4.38441	4.4496	0.001	0.02306	-2.82843	8.00000
Ln_DOM3_ALL_MRMC_CM3PG	8	4.31093	2.91733	5.2040	0.608	0.77968	-0.70933	-0.28484
Ln_DOM3_ALL_MRMC_SG	8	1.16364	1.11525	1.2019	0.001	0.03196	-0.39257	-0.91710

Merensky Reef Domain 4

Variable	Descriptive Statistics (Spreadsheet7)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM4_ALL_MRMC_CW	16	117.7998	85.5951	243.245	2476.6	49.7654	1.684105	1.753551
DOM4_ALL_MRMC_CM3PG	16	767.8015	113.2901	3005.364	687663.3	829.2547	1.746093	2.580283
DOM4_ALL_MRMC_SG	16	3.2640	3.1116	3.524	0.0	0.1118	0.757099	0.468116
Ln_DOM4_ALL_MRMC_CW	16	4.7042	4.4496	5.494	0.1	0.3483	1.403509	0.675938
Ln_DOM4_ALL_MRMC_CM3PG	16	6.1545	4.7300	8.008	1.0	1.0235	0.277543	-0.883739
Ln_DOM4_ALL_MRMC_SG	16	1.1824	1.1351	1.260	0.0	0.0339	0.675959	0.277282

Merensky Reef Domain 5

Variable	Descriptive Statistics (Spreadsheet9)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM5_ALL_MRMC_CW	19	87.24485	80.19059	124.7409	139.294	11.80230	2.176408	4.93976
DOM5_ALL_MRMC_CM3PG	19	56.01191	3.82000	218.1154	3608.383	60.06982	1.450904	1.72278
DOM5_ALL_MRMC_SG	19	3.16692	2.77000	3.3567	0.030	0.17405	-0.839621	0.00711
Ln_DOM5_ALL_MRMC_CW	19	4.46121	4.38441	4.8262	0.015	0.12161	1.923236	3.51410
Ln_DOM5_ALL_MRMC_CM3PG	19	3.36814	1.34025	5.3850	1.660	1.28855	-0.138314	-1.34930
Ln_DOM5_ALL_MRMC_SG	19	1.15128	1.01885	1.2109	0.003	0.05627	-0.942540	0.30777

Merensky Reef Domain 6

Merensky Reef Domain 7

Variable	Descriptive Statistics (Spreadsheet13)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM7_ALL_MRMC_CW	17	94.9938	85.59509	114.1279	123.91	11.1313	0.54548	-1.50290
DOM7_ALL_MRMC_CM3PG	17	264.1820	19.09236	520.8624	18167.21	134.7858	0.23225	-0.29240
DOM7_ALL_MRMC_SG	17	3.1739	2.93844	3.3851	0.01	0.0954	-0.17228	2.57453
Ln_DOM7_ALL_MRMC_CW	17	4.5475	4.44963	4.7373	0.01	0.1147	0.49025	-1.62194
Ln_DOM7_ALL_MRMC_CM3PG	17	5.3833	2.94929	6.2555	0.60	0.7752	-2.03057	5.65327
Ln_DOM7_ALL_MRMC_SG	17	1.1545	1.07788	1.2194	0.00	0.0302	-0.34318	2.72416

Merensky Reef Domain 8

Variable	Descriptive Statistics (Spreadsheet15)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM8_ALL_MRMC_CW	50	130.6750	85.59509	434.633	6104.0	78.1279	3.128220	9.394789
DOM8_ALL_MRMC_CM3PG	50	804.9486	35.07523	3557.912	467542.4	683.7707	2.160852	5.760548
DOM8_ALL_MRMC_SG	50	3.1763	3.16000	3.330	0.0	0.0411	2.523738	5.434229
Ln_DOM8_ALL_MRMC_CW	50	4.7773	4.44963	6.075	0.1	0.3813	2.355565	5.275025
Ln_DOM8_ALL_MRMC_CM3PG	50	6.3807	3.55750	8.177	0.7	0.8416	-0.591247	1.513449
Ln_DOM8_ALL_MRMC_SG	50	1.1556	1.15057	1.203	0.0	0.0127	2.508420	5.330315

Table 1 - Merensky Reef Descriptive Statistics Project 1A (min 80 cm cut)

Variable	Descriptive Statistics (Spreadsheet1)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRMC_CW	56	121.5032	89.10065	231.796	1197.9	34.6110	1.553502	1.56287
DOM1_ALL_MRMC_3PGE_AU	53	6.3690	0.18752	24.228	27.2	5.2126	1.654106	2.60372
DOM1_ALL_MRMC_CM3PG	53	841.3068	19.21424	5616.035	932363.5	965.5897	3.120543	11.99085
Ln_DOM1_ALL_MRMC_CW	56	4.7669	4.48977	5.446	0.1	0.2478	1.189360	0.43140
Ln_DOM1_ALL_MRMC_3PGE_AU	53	1.5271	-1.67387	3.188	0.8	0.8941	-0.950900	2.50386
Ln_DOM1_ALL_MRMC_CM3PG	53	6.2834	2.95565	8.633	1.0	1.0067	-0.585408	1.94254

Variable	Descriptive Statistics (Spreadsheet3)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2_ALL_MRMC_CW	7	105.3041	90.88169	117.7527	117.0780	10.82026	-0.281833	-1.75519
DOM2_ALL_MRMC_3PGE_AU	6	0.2513	0.07629	0.6496	0.0707	0.26591	1.050066	-1.37739
DOM2_ALL_MRMC_CM3PG	6	27.2841	7.65440	66.2510	832.8726	28.85953	0.970799	-1.83650
Ln_DOM2_ALL_MRMC_CW	7	4.6522	4.50956	4.7686	0.0109	0.10436	-0.370529	-1.66969
Ln_DOM2_ALL_MRMC_3PGE_AU	6	-1.8507	-2.57323	-0.4314	1.0527	1.02600	0.972610	-1.77848
Ln_DOM2_ALL_MRMC_CM3PG	6	2.8253	2.03528	4.1935	1.0886	1.04335	0.932970	-1.87239

Variable	Descriptive Statistics (Spreadsheet5)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM3_ALL_MRMC_CW	5	103.4742	94.1539	117.9302	83.39	9.1321	1.120283	1.37683
DOM3_ALL_MRMC_3PGE_AU	5	5.4495	2.7086	8.6322	6.84	2.6162	0.393396	-2.62489
DOM3_ALL_MRMC_CM3PG	5	555.3352	319.4292	911.2676	66050.82	257.0035	0.688428	-1.70237
Ln_DOM3_ALL_MRMC_CW	5	4.6363	4.5449	4.7701	0.01	0.0861	0.981130	1.04733
Ln_DOM3_ALL_MRMC_3PGE_AU	5	1.5986	0.9964	2.1555	0.25	0.4976	0.040031	-2.27103
Ln_DOM3_ALL_MRMC_CM3PG	5	6.2349	5.7665	6.8148	0.21	0.4576	0.367958	-2.33851

The UG2 Reef has been subdivided into 8 Geozones. Table 1 - summarises the descriptive statistics for UG2 Reef. The Geozones were determined using the geological facies plans and grade distribution for the UG2 Reef.

Table 2 - UG2 Reef Descriptive Statistics Project 1(min 80cm width)

UG2 Reef Domain 1

Variable	Descriptive Statistics (Spreadsheet17)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2MC_CW	58	171.0255	85.59509	425.120	6242.6	79.0101	1.360521	1.382789
DOM1_ALL_UG2MC_CM3PG	58	745.5153	86.18525	1827.292	206439.1	454.3557	0.876890	-0.081622
DOM1_ALL_UG2MC_SG	58	3.6594	3.13209	4.083	0.0	0.1742	0.140768	2.045567
Ln_DOM1_ALL_UG2MC_CW	58	5.0532	4.44963	6.052	0.2	0.4108	0.594881	-0.404302
Ln_DOM1_ALL_UG2MC_CM3PG	58	6.4079	4.45650	7.511	0.5	0.7027	-0.816351	1.097747
Ln_DOM1_ALL_UG2MC_SG	58	1.2962	1.14170	1.407	0.0	0.0477	-0.143909	2.345537

UG2 Reef Domain 2

Variable	Descriptive Statistics (Spreadsheet19)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2_ALL_UG2MC_CW	13	117.9749	83.44655	229.942	1765.1	42.0126	1.77031	3.427108
DOM2_ALL_UG2MC_CM3PG	13	456.0514	98.24729	1488.194	142735.9	377.8041	1.75187	4.144320
DOM2_ALL_UG2MC_SG	13	3.6193	3.31670	3.694	0.0	0.1114	-2.27320	4.563512
Ln_DOM2_ALL_UG2MC_CW	13	4.7228	4.42421	5.438	0.1	0.3065	1.14828	0.910055
Ln_DOM2_ALL_UG2MC_CM3PG	13	5.8200	4.58749	7.305	0.7	0.8299	-0.00857	-0.986554
Ln_DOM2_ALL_UG2MC_SG	13	1.2858	1.19897	1.307	0.0	0.0317	-2.30198	4.726997

UG2 Reef Domain 3

Variable	Descriptive Statistics (Spreadsheet21)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM3_ALL_UG2MC_CW	11	81.6706	80.19059	83.4465	2.891	1.70037	0.212762	-2.44444
DOM3_ALL_UG2MC_CM3PG	11	114.7179	39.50519	197.0855	3578.736	59.82254	0.183234	-1.61006
DOM3_ALL_UG2MC_SG	11	3.2668	3.04224	3.5879	0.033	0.18048	0.565788	-1.14967
Ln_DOM3_ALL_UG2MC_CW	11	4.4025	4.38441	4.4242	0.000	0.02078	0.212762	-2.44444
Ln_DOM3_ALL_UG2MC_CM3PG	11	4.5958	3.67643	5.2836	0.355	0.59571	-0.401879	-1.22391
Ln_DOM3_ALL_UG2MC_SG	11	1.1824	1.11260	1.2776	0.003	0.05464	0.513922	-1.23083

UG2 Reef Domain 4

Variable	Descriptive Statistics (Spreadsheet23)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM4_ALL_UG2MC_CW	24	123.1388	80.1906	214.9388	1297.37	36.0191	0.91135	0.67995
DOM4_ALL_UG2MC_CM3PG	24	384.2893	119.8455	661.0321	26897.76	164.0054	0.05095	-1.25581
DOM4_ALL_UG2MC_SG	24	3.6135	3.2066	3.6950	0.01	0.1049	-2.79157	9.67777
Ln_DOM4_ALL_UG2MC_CW	24	4.7748	4.3844	5.3704	0.08	0.2804	0.29855	-0.49628
Ln_DOM4_ALL_UG2MC_CM3PG	24	5.8477	4.7862	6.4938	0.24	0.4896	-0.59031	-0.60379
Ln_DOM4_ALL_UG2MC_SG	24	1.2842	1.1652	1.3070	0.00	0.0302	-2.91959	10.47068

UG2 Reef Domain 5

Variable	Descriptive Statistics (Spreadsheet25)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM5_ALL_UG2MC_CW	11	91.63666	80.19059	171.0733	765.2669	27.66346	2.855710	8.365483
DOM5_ALL_UG2MC_CM3PG	11	44.77714	13.32594	92.0350	621.8021	24.93596	0.879934	0.082791
DOM5_ALL_UG2MC_SG	11	3.41814	3.13280	3.6463	0.0314	0.17723	-0.458135	-0.678988
Ln_DOM5_ALL_UG2MC_CW	11	4.48818	4.38441	5.1421	0.0551	0.23481	2.637062	7.009136
Ln_DOM5_ALL_UG2MC_CM3PG	11	3.65582	2.58971	4.5222	0.3389	0.58212	-0.246954	-0.225238
Ln_DOM5_ALL_UG2MC_SG	11	1.22786	1.14193	1.2937	0.0028	0.05246	-0.539596	-0.597783

UG2 Reef Domain 6

Variable	Descriptive Statistics (Spreadsheet27)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM6_ALL_UG2MC_CW	35	122.7162	80.19059	406.299	3657.57	60.4778	3.499206	14.62972
DOM6_ALL_UG2MC_CM3PG	35	390.8976	65.50446	1209.780	60079.99	245.1122	1.285904	2.38669
DOM6_ALL_UG2MC_SG	35	3.6807	3.35053	4.225	0.03	0.1861	1.560289	2.71444
Ln_DOM6_ALL_UG2MC_CW	35	4.7398	4.38441	6.007	0.12	0.3401	1.917169	5.05411
Ln_DOM6_ALL_UG2MC_CM3PG	35	5.7692	4.18212	7.098	0.46	0.6746	-0.464354	-0.05299
Ln_DOM6_ALL_UG2MC_SG	35	1.3019	1.20912	1.441	0.00	0.0489	1.405408	2.43474

UG2 Reef Domain 7

Variable	Descriptive Statistics (Spreadsheet29)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM7_ALL_UG2MC_CW	23	90.35168	80.19059	132.1983	164.231	12.81525	2.257906	4.969485
DOM7_ALL_UG2MC_CM3PG	23	79.90235	4.81794	429.3997	9131.024	95.55639	2.508069	7.712209
DOM7_ALL_UG2MC_SG	23	3.45833	3.12648	4.2991	0.092	0.30278	1.396723	2.109504
Ln_DOM7_ALL_UG2MC_CW	23	4.49550	4.38441	4.8843	0.016	0.12603	2.022692	3.716046
Ln_DOM7_ALL_UG2MC_CM3PG	23	3.77703	1.57235	6.0624	1.424	1.19343	-0.187320	-0.545965
Ln_DOM7_ALL_UG2MC_SG	23	1.23734	1.13991	1.4584	0.007	0.08361	1.172772	1.428410

UG2 Reef Domain 8

Variable	Descriptive Statistics (Spreadsheet31)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM8_ALL_UG2MC_CW	56	155.8882	91.30143	349.988	2981.22	54.6006	1.57800	2.841227
DOM8_ALL_UG2MC_CM3PG	56	566.3362	85.08193	1478.521	52350.20	228.8017	0.95271	3.632129
DOM8_ALL_UG2MC_SG	56	3.7863	3.47497	4.364	0.05	0.2141	1.01244	-0.024299
Ln_DOM8_ALL_UG2MC_CW	56	4.9986	4.51417	5.858	0.10	0.3100	0.71231	0.281486
Ln_DOM8_ALL_UG2MC_CM3PG	56	6.2453	4.44361	7.299	0.23	0.4782	-1.40623	3.592797
Ln_DOM8_ALL_UG2MC_SG	56	1.3299	1.24558	1.474	0.00	0.0552	0.93859	-0.219194

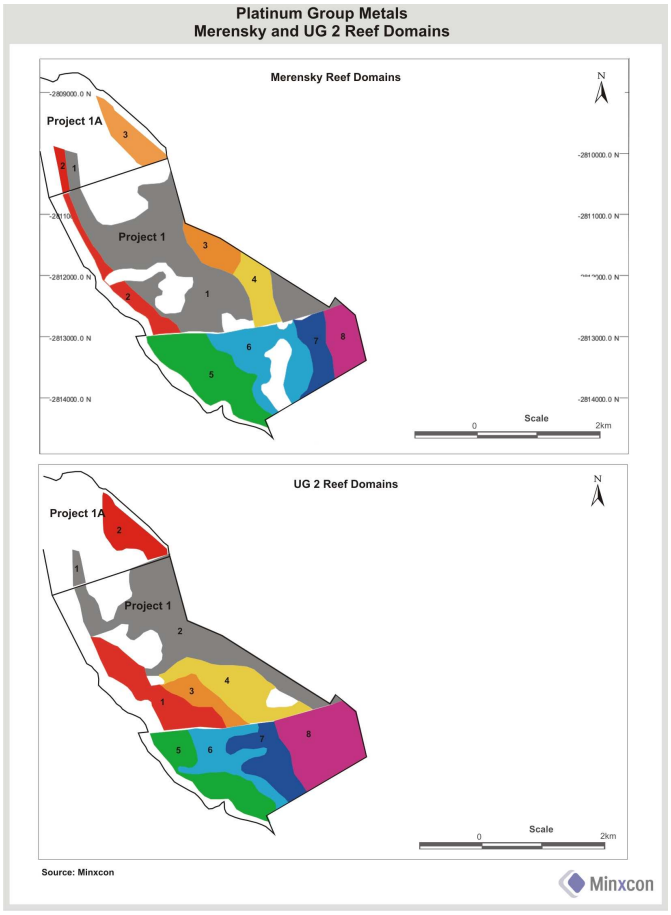
Table 3 - UG2 Reef Descriptive Statistics Project 1A (minimum 80cm width)

Variable	Descriptive Statistics (Spreadsheet7)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2MC_CW	30	185.2333	86.5494	382.324	6982.8	83.5633	0.76317	-0.43567
DOM1_ALL_UG2MC_3PGE_AU	30	4.9007	1.2558	8.455	2.3	1.5126	0.04616	1.13390
DOM1_ALL_UG2MC_CM3PG	30	914.0883	128.9952	1845.916	249544.1	499.5439	0.56137	-1.11444
Ln_DOM1_ALL_UG2MC_CW	30	5.1269	4.4607	5.946	0.2	0.4403	0.22383	-1.20102
Ln_DOM1_ALL_UG2MC_3PGE_AL	30	1.5316	0.2278	2.135	0.1	0.3766	-1.67263	4.61779
Ln_DOM1_ALL_UG2MC_CM3PG	30	6.6585	4.8598	7.521	0.4	0.6050	-0.59048	0.99742

Variable	Descriptive Statistics (Spreadsheet9)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2_ALL_UG2MC_CW	2	103.6644	97.0071	110.3218	88.64	9.4149		
DOM2_ALL_UG2MC_3PGE_AU	2	6.7492	5.6113	7.8871	2.59	1.6092		
DOM2_ALL_UG2MC_CM3PG	2	707.2246	544.3335	870.1156	53067.01	230.3628		
Ln_DOM2_ALL_UG2MC_CW	2	4.6391	4.5748	4.7034	0.01	0.0909		
Ln_DOM2_ALL_UG2MC_3PGE_AL	2	1.8950	1.7248	2.0652	0.06	0.2407		
Ln_DOM2_ALL_UG2MC_CM3PG	2	6.5341	6.2996	6.7686	0.11	0.3317		

The following map illustrates the location of the different Geozones or Domains for both the UG2 and MR:-

Figure 31 - MR and UG2 Domains



The following tables detail the descriptive stats for the three hangingwall (“HW”) and footwall (“FW”) units that were also modelled to achieve the best cut mining width:-

Table 69 - Descriptive Statistics for the MR HW and FW Units

Table 1 - Descriptive Statistics for the MR HW and FW Units

	Descriptive Statistics (Spreadsheet33)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRHW1_CW	162	18.46888	17.82013	19.02113	0.26609	0.515836	-0.24150	-1.62058
DOM1_ALL_MRHW1_CM3PG	162	4.23828	0.71281	33.32364	23.51647	4.849379	3.02714	11.99943
DOM1_ALL_MRHW1_SG	162	3.18115	2.70000	3.34300	0.02738	0.165476	-1.04013	0.31039
Ln_DOM1_ALL_MRHW1_CW	162	2.91570	2.88033	2.94555	0.00079	0.028034	-0.25552	-1.61980
Ln_DOM1_ALL_MRHW1_CM3PG	162	1.00188	-0.33855	3.50627	0.84812	0.920935	0.29616	-0.49139
Ln_DOM1_ALL_MRHW1_SG	162	1.15584	0.99325	1.20687	0.00287	0.053599	-1.14358	0.67809

	Descriptive Statistics (Spreadsheet35)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRHW2_CW	160	18.46794	17.82013	19.02113	0.26936	0.519001	-0.234689	-1.63948
DOM1_ALL_MRHW2_CM3PG	160	3.53386	0.71281	32.25173	20.78837	4.559427	4.148364	20.22906
DOM1_ALL_MRHW2_SG	160	3.16956	2.70000	3.41000	0.03037	0.174257	-0.809906	-0.29225
Ln_DOM1_ALL_MRHW2_CW	160	2.91564	2.88033	2.94555	0.00080	0.028205	-0.248170	-1.63885
Ln_DOM1_ALL_MRHW2_CM3PG	160	0.86415	-0.33855	3.47357	0.67982	0.824515	0.561293	0.57568
Ln_DOM1_ALL_MRHW2_SG	160	1.15204	0.99325	1.22671	0.00318	0.056373	-0.904651	0.01372

	Descriptive Statistics (Spreadsheet37)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRHW3_CW	151	18.46995	17.82013	19.02113	0.26613	0.515878	-0.245154	-1.61959
DOM1_ALL_MRHW3_CM3PG	151	3.52783	0.71281	44.57864	30.03233	5.480176	5.669600	38.44230
DOM1_ALL_MRHW3_SG	151	3.14842	2.70000	3.39100	0.02999	0.173166	-0.554685	-0.72198
Ln_DOM1_ALL_MRHW3_CW	151	2.91576	2.88033	2.94555	0.00079	0.028036	-0.259194	-1.61869
Ln_DOM1_ALL_MRHW3_CM3PG	151	0.81138	-0.33855	3.79725	0.71948	0.848221	0.742504	0.85397
Ln_DOM1_ALL_MRHW3_SG	151	1.14536	0.99325	1.22112	0.00313	0.055985	-0.641986	-0.48422

	Descriptive Statistics (Spreadsheet39)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRFW1_CW	165	18.47024	17.82013	19.0211	0.261	0.51119	-0.251561	-1.59218
DOM1_ALL_MRFW1_CM3PG	165	29.71360	0.71281	548.1472	4741.172	68.85617	4.472123	25.38413
DOM1_ALL_MRFW1_SG	165	2.98488	2.70000	3.3500	0.014	0.11718	0.979442	2.62653
Ln_DOM1_ALL_MRFW1_CW	165	2.91578	2.88033	2.9456	0.001	0.02778	-0.266386	-1.59118
Ln_DOM1_ALL_MRFW1_CM3PG	165	1.80346	-0.33855	6.3065	3.013	1.73593	0.617410	-0.65144
Ln_DOM1_ALL_MRFW1_SG	165	1.09281	0.99325	1.2090	0.001	0.03864	0.768194	2.34603

	Descriptive Statistics (Spreadsheet41)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRFW2_CW	165	18.47024	17.82013	19.0211	0.261	0.51119	-0.251561	-1.59218
DOM1_ALL_MRFW2_CM3PG	165	25.71037	0.71281	271.2871	2791.321	52.83296	2.862832	8.18402
DOM1_ALL_MRFW2_SG	165	2.99549	2.70000	3.3620	0.013	0.11481	1.318433	2.98655
Ln_DOM1_ALL_MRFW2_CW	165	2.91578	2.88033	2.9456	0.001	0.02778	-0.266386	-1.59118
Ln_DOM1_ALL_MRFW2_CM3PG	165	1.66333	-0.33855	5.6032	2.955	1.71888	0.736077	-0.60838
Ln_DOM1_ALL_MRFW2_SG	165	1.09640	0.99325	1.2125	0.001	0.03750	1.134541	2.70145

	Descriptive Statistics (Spreadsheet43)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_MRFW3_CW	160	18.47400	17.82013	19.0211	0.262	0.51208	-0.262462	-1.59177
DOM1_ALL_MRFW3_CM3PG	160	19.15961	0.57565	237.2107	2061.796	45.40700	3.296452	10.96508
DOM1_ALL_MRFW3_SG	160	2.99480	2.70000	3.3500	0.014	0.11854	1.329856	2.59101
Ln_DOM1_ALL_MRFW3_CW	160	2.91598	2.88033	2.9456	0.001	0.02783	-0.277122	-1.59035
Ln_DOM1_ALL_MRFW3_CM3PG	160	1.21295	-0.55225	5.4689	2.817	1.67845	1.102737	0.03036
Ln_DOM1_ALL_MRFW3_SG	160	1.09612	0.99325	1.2090	0.001	0.03868	1.164649	2.33780

Table 2 - Descriptive Statistics for the UG2 HW and FW Units

	Descriptive Statistics (Spreadsheet45)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2HW1_CW	176	18.49394	17.11902	19.02113	0.26222	0.512079	-0.40701	-1.31798
DOM1_ALL_UG2HW1_CM3PG	176	7.54322	0.71281	60.86762	71.93503	8.481452	2.81571	11.54457
DOM1_ALL_UG2HW1_SG	176	3.22583	2.70000	3.60100	0.02566	0.160193	-0.90609	0.53866
Ln_DOM1_ALL_UG2HW1_CW	176	2.91706	2.84019	2.94555	0.00078	0.027857	-0.42927	-1.28607
Ln_DOM1_ALL_UG2HW1_CM3PG	176	1.54939	-0.33855	4.10870	0.93927	0.969158	0.18288	-0.53971
Ln_DOM1_ALL_UG2HW1_SG	176	1.16992	0.99325	1.28121	0.00260	0.050942	-1.03206	0.71852

	Descriptive Statistics (Spreadsheet47)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2HW2_CW	168	18.50583	17.82013	19.02113	0.25616	0.506126	-0.37006	-1.49948
DOM1_ALL_UG2HW2_CM3PG	168	9.58878	0.66718	60.86762	96.19778	9.808047	1.64459	3.96911
DOM1_ALL_UG2HW2_SG	168	3.22876	2.70000	3.60100	0.02512	0.158478	-0.98002	0.66642
Ln_DOM1_ALL_UG2HW2_CW	168	2.91771	2.88033	2.94555	0.00076	0.027498	-0.38546	-1.49400
Ln_DOM1_ALL_UG2HW2_CM3PG	168	1.69702	-0.40470	4.10870	1.27434	1.128866	-0.03329	-1.17520
Ln_DOM1_ALL_UG2HW2_SG	168	1.17086	0.99325	1.28121	0.00254	0.050426	-1.10293	0.84256

	Descriptive Statistics (Spreadsheet49)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2HW3_CW	156	18.49531	17.82013	19.02113	0.2659	0.51569	-0.32605	-1.57745
DOM1_ALL_UG2HW3_CM3PG	156	11.78139	0.64765	47.57185	111.0468	10.53788	0.67093	-0.48512
DOM1_ALL_UG2HW3_SG	156	3.21977	2.70000	3.60100	0.0286	0.16924	-0.97490	0.84358
Ln_DOM1_ALL_UG2HW3_CW	156	2.91713	2.88033	2.94555	0.0008	0.02802	-0.33988	-1.57382
Ln_DOM1_ALL_UG2HW3_CM3PG	156	1.86279	-0.43441	3.86224	1.5582	1.24827	-0.31062	-1.31360
Ln_DOM1_ALL_UG2HW3_SG	156	1.16788	0.99325	1.28121	0.0029	0.05412	-1.12347	1.11995

	Descriptive Statistics (Spreadsheet51)							
Variable	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2FW1_CW	184	18.50904	17.82013	19.0211	0.2433	0.49329	-0.394879	-1.40936
DOM1_ALL_UG2FW1_CM3PG	184	9.20748	0.71281	104.2358	210.4723	14.50766	3.837361	17.96024
DOM1_ALL_UG2FW1_SG	184	3.13748	2.67000	3.6600	0.0513	0.22660	-0.205485	-0.56965
Ln_DOM1_ALL_UG2FW1_CW	184	2.91790	2.88033	2.9456	0.0007	0.02680	-0.412607	-1.40241
Ln_DOM1_ALL_UG2FW1_CM3PG	184	1.55523	-0.33855	4.6467	1.1958	1.09351	0.449648	-0.13128
Ln_DOM1_ALL_UG2FW1_SG	184	1.14078	0.98208	1.2975	0.0053	0.07313	-0.354760	-0.51137

Variable	Descriptive Statistics (Spreadsheet53)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2FW2_CW	183	18.50885	17.82013	19.0211	0.2447	0.49464	-0.392679	-1.41883
DOM1_ALL_UG2FW2_CM3PG	183	8.01522	0.66129	129.6903	259.2453	16.10110	4.646656	25.70772
DOM1_ALL_UG2FW2_SG	183	3.12945	2.70000	3.6215	0.0486	0.22040	-0.143714	-0.54500
Ln_DOM1_ALL_UG2FW2_CW	183	2.91789	2.88033	2.9456	0.0007	0.02687	-0.410157	-1.41203
Ln_DOM1_ALL_UG2FW2_CM3PG	183	1.28263	-0.41356	4.8651	1.2248	1.10671	0.906625	0.62462
Ln_DOM1_ALL_UG2FW2_SG	183	1.13836	0.99325	1.2869	0.0051	0.07113	-0.294885	-0.48600

Variable	Descriptive Statistics (Spreadsheet55)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1_ALL_UG2FW3_CW	174	18.51678	17.82013	19.0211	0.2476	0.49755	-0.414369	-1.42072
DOM1_ALL_UG2FW3_CM3PG	174	6.40186	0.71281	134.4691	262.5193	16.20245	6.079932	42.66781
DOM1_ALL_UG2FW3_SG	174	3.10873	2.70000	3.8100	0.0456	0.21345	0.396442	0.67219
Ln_DOM1_ALL_UG2FW3_CW	174	2.91832	2.88033	2.9456	0.0007	0.02703	-0.431208	-1.41310
Ln_DOM1_ALL_UG2FW3_CM3PG	174	0.99838	-0.33855	4.9013	1.0905	1.04426	1.427952	2.19089
Ln_DOM1_ALL_UG2FW3_SG	174	1.13189	0.99325	1.3376	0.0046	0.06819	0.158572	0.29033

Capping and Cutting

No corrections were made (top cut etc.) to the data as all fall within acceptable limits. The statistical analyses show the expected relationships for these types of reef.

Item 19(d): Mining Considerations

As only Mineral Resources were evaluated, no mine plan was taken into consideration. The reefs were however evaluated over a realistic average diluted mining width as follows:-

Table - Summary of Mining Widths

Reef	Project	Resource Category	Mining Width (cm)
Merensky	Project 1	Measured	108
		Indicated	109
		Inferred	93
UG2	Project 1	Measured	141
		Indicated	134
		Inferred	134

The initial occurrence of payable reef is at 130m below surface. The payable reef then extends to a depth of approximately 630m below surface. There are no near-surface or outcropping reefs of sufficient width, geometry or value to provide an opportunity for opencast mining.

A cut-off grade of 300cm.g/t was selected as a Resource cut-off. The reason for using the 300cm.g/t cut-off is in compliance with responsible engineering practice to simulate probable working cost and flow of ore parameters, in order to report potentially economical Resources.

Reef Compositing Definitions

For the Resource cut width model, the borehole samples were composited into single drill hole intersections according to the pertinent Resource width. Density was incorporated in the compositing phase. The boreholes were subsequently corrected for dip (3D dip model constructed), resulting in true width intersection values. The data was composited according to reef horizon. The following table is a summary of Resource width (“RW”) sample composites with values:-

Table - Merensky Reef - Resource Cut

BHID	FROM	TO	SG	LENGTH	PT	PD	RH	AU	OS	IR	RU	4E	CU%	NI%	RW
	m	m	t/m 3	m	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	%	%	cm
WBJV001D0	447.60	448.65	3.42	1.05	2.82	1.28	0.17	0.17	0.06	0.10	0.42	4.44	0.10	0.19	99.86
WBJV001D2	27.94	28.93	3.51	0.99	3.28	1.49	0.20	0.28	0.07	0.12	0.47	5.26	0.11	0.22	94.15
WBJV010D1	51.42	52.43	3.56	1.01	1.43	0.60	0.24	0.01	0.04	0.06	0.25	2.28	0.00	0.13	96.05
WBJV015D0	389.67	390.73	3.43	1.06	6.50	2.36	0.29	0.38	0.12	0.22	0.86	9.52	0.00	0.09	100.81
WBJV015D1	31.76	33.22	3.34	1.46	2.97	1.25	0.14	0.19	0.07	0.11	0.43	4.56	0.05	0.14	138.85
WBJV030D0	475.89	477.12	3.22	1.23	5.08	2.07	0.27	0.43	0.10	0.18	0.69	7.86	0.12	0.24	116.98
WBJV030D1	21.03	22.21	3.26	1.18	3.16	1.54	0.15	0.33	0.07	0.12	0.46	5.19	0.12	0.27	112.22
WBJV030D2	27.77	28.81	3.32	1.04	0.09	0.04	0.01	0.09	0.02	0.02	0.09	0.23	0.06	0.17	98.91
WBJV033D0	338.61	339.80	3.29	1.19	2.02	1.01	0.11	0.29	0.05	0.08	0.32	3.43	0.09	0.17	113.18
WBJV045D1	62.00	63.19	3.33	1.19	0.01	0.01	0.01	0.01	0.02	0.02	0.06	0.04	0.00	0.04	113.18
WBJV048D0	423.17	424.37	3.27	1.20	0.61	0.58	0.06	0.10	0.03	0.04	0.15	1.36	0.09	0.12	114.13
WBJV048D1	44.36	45.59	3.28	1.23	5.24	1.88	0.22	0.32	0.10	0.18	0.71	7.66	0.09	0.19	116.98
WBJV050D0	530.63	531.75	3.11	1.12	4.30	1.94	0.22	0.30	0.07	0.08	0.53	6.77	0.11	0.22	106.52
WBJV050D1	35.51	36.93	3.20	1.42	4.77	2.22	0.26	0.33	0.10	0.12	0.72	7.58	0.11	0.26	135.05
WBJV104D0	535.67	536.75	3.05	1.08	0.13	0.04	0.01	0.01	0.02	0.02	0.09	0.19	0.01	0.04	102.72
WBJV104D1	60.46	61.64	3.12	1.18	1.00	0.55	0.07	0.08	0.03	0.05	0.20	1.69	0.08	0.11	112.22
WBJV104D2	66.18	67.22	2.97	1.04	0.57	0.30	0.03	0.09	0.03	0.03	0.15	0.99	0.10	0.11	98.91
WBJV109D1	27.98	29.25	3.29	1.27	3.98	1.59	0.27	0.30	0.08	0.14	0.56	6.13	0.07	0.19	120.78
WBJV109D2	33.29	34.67	3.18	1.38	4.48	1.82	0.22	0.48	0.09	0.16	0.62	7.00	0.07	0.17	131.25
WBJV112D0	450.76	453.64	3.19	2.88	3.66	1.31	0.28	0.09	0.08	0.13	0.52	5.35	0.02	0.10	273.90
WBJV112D1	19.88	23.44	3.17	3.56	3.47	1.27	0.26	0.05	0.07	0.12	0.51	5.06	0.03	0.10	338.58
WBJV112D2	23.28	28.13	3.24	4.85	4.92	2.29	0.35	0.24	0.10	0.12	0.70	7.79	0.07	0.18	461.26
WBJV116D0	506.30	507.52	3.11	1.22	2.03	0.89	0.10	0.19	0.04	0.04	0.27	3.20	0.09	0.19	116.03
WBJV116D1	16.19	17.41	3.15	1.22	2.75	1.21	0.13	0.24	0.05	0.06	0.34	4.33	0.08	0.20	116.03
WBJV116D2	21.20	22.35	3.19	1.15	3.48	0.88	0.16	0.32	0.07	0.09	0.50	4.84	0.08	0.17	109.37
WBJV120D0	330.66	331.67	3.11	1.01	0.22	0.11	0.02	0.06	0.01	0.01	0.04	0.41	0.04	0.10	96.06
WBJV120D3	25.06	26.30	3.24	1.24	0.34	0.22	0.03	0.11	0.01	0.01	0.05	0.69	0.06	0.14	117.93
WBJV120RDRIL	19.83	21.03	3.24	1.20	0.30	0.16	0.02	0.11	0.01	0.01	0.05	0.60	0.06	0.14	114.13
WBJV124D0	489.44	490.47	3.20	1.03	6.01	2.55	0.34	0.42	0.14	0.17	0.98	9.32	0.13	0.29	97.96
WBJV124D1	9.34	10.43	3.14	1.09	5.77	1.96	0.40	0.24	0.12	0.15	0.85	8.38	0.13	0.25	103.66
WBJV124D3	29.30	30.50	3.14	1.20	7.14	1.91	0.33	0.31	0.10	0.12	0.71	9.68	0.15	0.23	114.13
WBJV125D0	457.71	458.88	3.12	1.17	3.45	1.60	0.20	0.24	0.07	0.08	0.45	5.48	0.09	0.17	111.27
WBJV125D1	16.90	18.04	3.13	1.14	2.07	1.03	0.12	0.19	0.04	0.04	0.25	3.41	0.07	0.10	108.42
WBJV127D0	446.28	447.44	3.12	1.16	2.38	0.86	0.10	0.21	0.04	0.04	0.23	3.55	0.07	0.17	110.32
WBJV127D1	7.92	9.09	3.15	1.17	2.91	1.41	0.19	0.27	0.06	0.07	0.39	4.78	0.09	0.25	111.27
WBJV127D2	13.65	14.80	3.13	1.15	2.43	0.94	0.11	0.34	0.03	0.04	0.23	3.82	0.08	0.18	109.37
WBJV131D0	548.75	550.00	3.22	1.25	8.65	4.70	0.68	0.70	0.19	0.25	1.43	14.74	0.11	0.34	118.88
WBJV131D1	7.19	8.22	3.28	1.03	7.84	3.50	0.41	0.28	0.10	0.12	0.72	12.02	0.08	0.21	97.96
WBJV131D3	18.19	19.25	3.29	1.06	7.95	3.46	0.62	0.58	0.18	0.23	1.33	12.62	0.13	0.29	100.81
WBJV137D3	51.40	52.46	3.11	1.06	0.95	0.39	0.05	0.08	0.03	0.05	0.19	1.47	0.04	0.09	100.81
WBJV141D0	337.72	338.83	3.11	1.11	1.09	0.41	0.08	0.24	0.04	0.05	0.21	1.82	0.06	0.15	105.57
WBJV141D1	7.60	8.75	3.18	1.15	1.62	0.56	0.10	0.18	0.04	0.07	0.27	2.45	0.07	0.16	109.37
WBJV142D0	409.47	410.85	3.18	1.38	3.43	1.15	0.17	0.22	0.07	0.12	0.49	4.97	0.07	0.17	131.25
WBJV142D1	17.43	19.07	3.18	1.64	4.24	1.76	0.29	0.30	0.09	0.15	0.59	6.58	0.07	0.18	155.97
WBJV143D0	377.31	378.51	3.19	1.20	1.75	0.45	0.13	0.12	0.05	0.07	0.29	2.44	0.06	0.13	114.13
WBJV145D0	523.29	524.44	3.23	1.15	0.72	0.42	0.04	0.14	0.03	0.04	0.16	1.31	0.06	0.14	109.37
WBJV145D1	3.38	4.60	3.22	1.22	0.80	0.55	0.04	0.15	0.03	0.04	0.17	1.54	0.06	0.14	116.03
WBJV145D2	8.66	10.12	3.23	1.46	1.06	0.56	0.06	0.17	0.03	0.05	0.20	1.85	0.06	0.14	138.86
WBJV153D0	513.81	515.15	3.17	1.34	3.81	3.66	0.31	0.48	0.08	0.14	0.53	8.27	0.14	0.30	127.44

WBJV154D0	339.16	340.20	3.04	1.04	0.71	0.41	0.05	0.10	0.03	0.04	0.16	1.27	0.05	0.09	98.91
WBJV154D4	28.97	30.07	3.16	1.10	0.66	0.36	0.05	0.08	0.03	0.04	0.16	1.15	0.05	0.13	104.62
WBJV170D0	258.16	259.28	3.25	1.12	2.59	1.16	0.15	0.24	0.06	0.10	0.39	4.14	0.07	0.16	106.52
WBJV170D1	13.41	14.43	3.24	1.02	3.00	1.09	0.17	0.27	0.07	0.11	0.44	4.52	0.06	0.17	97.01
WBJV170D2	23.75	24.81	3.25	1.06	2.96	0.95	0.27	0.07	0.07	0.11	0.43	4.25	0.03	0.11	100.81
WBJV170D3	33.35	34.50	3.24	1.15	2.45	1.46	0.20	0.20	0.06	0.09	0.37	4.32	0.04	0.11	109.37
WBJV002D0	464.62	465.91	3.52	1.29	3.50	1.73	0.21	0.37	0.07	0.13	0.50	5.80	0.11	0.24	119.61
WBJV006D0	459.98	460.98	3.55	1.00	10.04	4.56	0.55	0.45	0.18	0.33	1.28	15.61	0.14	0.26	92.72
WBJV006D1	96.69	97.69	3.29	1.00	9.77	5.06	0.54	0.74	0.17	0.32	1.25	16.12	0.19	0.40	92.72
WBJV012D0	64.16	65.22	3.09	1.06	0.35	0.13	0.05	0.01	0.02	0.03	0.12	0.55	0.00	0.03	98.28
WBJV016D0	117.60	118.73	3.31	1.13	0.65	0.36	0.03	0.12	0.03	0.04	0.16	1.15	0.03	0.07	104.77
WBJV016D1	27.12	28.20	3.26	1.08	0.30	0.14	0.02	0.09	0.02	0.03	0.11	0.55	0.04	0.11	100.14
WBJV017D0	77.15	78.15	3.08	1.00	0.04	0.02	0.01	0.01	0.02	0.02	0.08	0.08	0.00	0.02	92.72
WBJV017D1	16.65	17.65	3.05	1.00	0.04	0.02	0.01	0.01	0.02	0.02	0.08	0.08	0.00	0.02	92.72
WBJV018D1	30.95	32.12	3.33	1.17	5.87	2.61	0.21	0.58	0.11	0.20	0.78	9.27	0.19	0.35	108.48
WBJV025D0	113.63	114.90	3.19	1.27	0.29	0.19	0.02	0.03	0.02	0.03	0.11	0.53	0.01	0.05	117.75
WBJV025D1	33.36	34.46	3.28	1.10	0.35	0.24	0.01	0.04	0.02	0.03	0.12	0.65	0.01	0.05	101.99
WBJV040D0	384.84	385.84	3.05	1.00	0.02	0.02	0.01	0.04	0.02	0.02	0.06	0.08	0.01	0.05	92.72
WBJV040D1	14.74	15.97	3.13	1.23	1.48	0.73	0.07	0.26	0.04	0.06	0.25	2.53	0.02	0.11	114.04
WBJV042D1	7.74	8.89	3.24	1.15	3.46	1.72	0.21	0.44	0.07	0.12	0.49	5.83	0.13	0.24	106.63
WBJV042D2	14.80	15.84	3.20	1.04	4.75	2.30	0.27	0.45	0.09	0.17	0.65	7.77	0.12	0.26	96.43
WBJV057D0	145.72	146.77	3.25	1.05	2.94	1.10	0.16	0.13	0.04	0.06	0.29	4.33			97.35
WBJV057D1	55.36	56.43	3.06	1.07	1.36	0.48	0.09	0.05	0.03	0.03	0.17	1.97			99.21
WBJV058D0	384.49	385.67	3.25	1.18	4.74	1.66	0.30	0.33	0.11	0.13	0.73	7.03			109.41
WBJV058D1	3.68	4.80	3.25	1.12	7.35	1.48	0.31	0.30	0.10	0.13	0.68	9.43			103.84
WBJV059D0	184.20	185.20	3.35	1.00	0.99	0.26	0.14	0.01	0.05	0.08	0.30	1.40			92.72
WBJV059D1	34.15	35.34	3.24	1.19	0.57	0.27	0.11	0.01	0.04	0.05	0.21	0.96			110.34
WBJV063D0	139.64	140.88	3.06	1.24	0.05	0.02	0.01	0.01	0.02	0.02	0.08	0.09			114.97
WBJV063D1	19.87	20.87	3.09	1.00	0.04	0.02	0.01	0.01	0.02	0.02	0.08	0.08			92.72
WBJV066D0	107.96	109.09	3.21	1.13	0.03	0.02	0.01	0.01	0.02	0.02	0.08	0.08	0.00	0.02	104.77
WBJV066D1	27.72	28.95	3.23	1.23	0.04	0.02	0.01	0.01	0.02	0.02	0.08	0.08	0.01	0.03	114.04
WBJV073D0	146.37	147.58	3.20	1.21	5.13	2.06	0.32	0.38	0.09	0.12	0.66	7.89	0.11	0.25	112.19
WBJV095D0	417.40	419.40	3.25	2.00	2.57	1.14	0.12	0.28	0.06	0.10	0.39	4.11	0.09	0.19	185.44
WBJV095D1	13.03	15.28	3.30	2.25	3.19	1.52	0.15	0.40	0.07	0.12	0.46	5.25	0.12	0.26	208.62
WBJV096D1	60.85	63.35	3.29	2.50	15.34	7.19	1.03	0.66	0.26	0.50	1.92	24.23	0.12	0.28	231.80
WBJV096D2	71.03	73.20	3.28	2.17	10.12	3.87	0.43	0.73	0.18	0.33	1.29	15.15	0.16	0.34	201.20
WBJV101D0	498.65	499.65	3.23	1.00	0.01	0.01	0.01	0.01	0.02	0.02	0.08	0.04	0.00	0.04	92.72
WBJV102D0	408.86	410.20	3.26	1.34	2.36	1.01	0.11	0.21	0.06	0.09	0.36	3.70	0.06	0.15	124.25
WBJV106D0	398.08	399.07	3.17	0.99	4.77	1.94	0.32	0.32	0.09	0.17	0.65	7.35	0.12	0.23	91.79
WBJV106D2	28.75	29.88	3.08	1.13	4.28	1.73	0.24	0.34	0.09	0.15	0.59	6.59	0.11	0.24	104.77
WBJV130D0	483.16	484.44	3.21	1.28	3.25	1.50	0.23	0.23	0.08	0.08	0.51	5.21	0.11	0.24	118.68
WBJV130D1	8.63	9.73	3.20	1.10	6.81	2.47	0.33	0.40	0.11	0.12	0.70	10.01	0.14	0.32	101.99
WBJV130D2	13.43	14.72	3.18	1.29	2.21	0.93	0.14	0.54	0.05	0.08	0.33	3.82	0.14	0.19	119.61
WBJV133D0	512.32	513.33	2.96	1.01	0.01	0.01	0.01	0.01	0.02	0.02	0.02	0.04	0.00	0.04	93.65
WBJV133D1	12.70	13.77	3.00	1.07	0.01	0.01	0.01	0.01	0.02	0.02	0.08	0.04	0.00	0.04	99.21
WBJV133D2	17.65	18.95	3.02	1.30	0.01	0.01	0.01	0.01	0.02	0.02	0.08	0.04	0.00	0.04	120.53
WBJV139D1	11.72	12.76	3.16	1.04	5.82	2.42	0.23	0.79	0.11	0.20	0.78	9.27	0.15	0.25	96.43
WBJV139D2	14.04	15.30	3.12	1.26	7.05	2.49	0.34	0.52	0.13	0.24	0.92	10.39	0.11	0.23	116.82
WBJV008D0	243.00	244.23	3.28	1.23	1.29	0.60	0.09	0.11	0.04	0.06	0.23	2.10	0.05	0.13	109.59
WBJV008D1	19.48	20.52	3.26	1.04	0.50	0.28	0.01	0.11	0.03	0.03	0.14	0.90	0.05	0.10	92.66

WBJV014D1	37.82	38.82	3.07	1.00	0.31	0.15	0.05	0.01	0.02	0.03	0.11	0.52	0.00	0.03	89.10
WBJV022D0	81.16	82.16	3.06	1.00	0.18	0.08	0.03	0.01	0.02	0.02	0.10	0.30	0.00	0.02	89.10
WBJV022D1	22.08	23.21	3.16	1.13	0.04	0.06	0.01	0.01	0.02	0.02	0.08	0.13	0.00	0.03	100.68
WBJV022D2	11.50	12.54	3.26	1.04	0.06	0.03	0.01	0.01	0.02	0.02	0.08	0.12	0.00	0.04	92.66
WBJV026D0	61.36	62.56	3.32	1.20	0.18	0.24	0.03	0.14	0.02	0.02	0.10	0.59	0.07	0.17	106.92
WBJV026D1	11.51	12.51	3.33	1.00	0.98	0.24	0.03	0.11	0.03	0.05	0.19	1.36	0.07	0.16	89.10
WBJV029D1	56.01	57.58	3.42	1.57	3.89	2.32	0.28	0.41	0.08	0.14	0.54	6.91	0.20	0.40	139.89
WBJV053D0	220.50	222.54	3.34	2.04	7.61	2.40	0.48	0.39	0.15	0.18	1.06	10.87	0.11	0.27	181.76
WBJV056D1	36.30	37.35	3.33	1.05	0.80	0.59	0.08	0.25	0.03	0.03	0.21	1.72			93.56
WBJV064D0	228.76	229.86	3.05	1.10	0.04	0.02	0.01	0.01	0.02	0.02	0.08	0.08			98.01
WBJV064D1	18.25	19.26	3.03	1.01	0.13	0.05	0.02	0.01	0.02	0.02	0.09	0.21			89.99
WBJV065D1	8.26	9.61	3.14	1.35	0.05	0.02	0.01	0.01	0.01	0.00	0.02	0.09			120.29
WBJV069D0	199.50	200.93	3.07	1.43	0.02	0.01	0.01	0.01	0.02	0.02	0.08	0.05	0.00	0.02	127.41
WBJV075D0	87.00	88.10	3.10	1.10	0.08	0.03	0.01	0.01	0.02	0.02	0.09	0.13	0.00	0.04	98.01
WBJV076D0	105.15	106.18	3.34	1.03	0.11	0.06	0.01	0.10	0.02	0.02	0.09	0.29	0.02	0.09	91.77
WBJV077D0	219.70	220.84	3.32	1.14	0.18	0.09	0.02	0.02	0.02	0.02	0.10	0.30	0.00	0.05	101.57
WBJV083D0	143.09	144.13	3.21	1.04	0.30	0.18	0.04	0.02	0.02	0.03	0.11	0.55	0.01	0.06	92.67
WBJV083D1	12.71	13.86	3.09	1.15	0.11	0.05	0.01	0.01	0.02	0.02	0.09	0.19	0.00	0.03	102.47
WBJV083D2	18.07	19.11	3.27	1.04	1.22	0.51	0.18	0.02	0.04	0.05	0.22	1.93	0.01	0.06	92.66
WBJV084D0	160.64	161.93	3.25	1.29	3.80	1.37	0.17	0.30	0.05	0.07	0.37	5.64	0.09	0.23	114.94
WBJV085D0	467.14	468.16	3.32	1.02	3.33	1.15	0.15	0.20	0.07	0.12	0.48	4.83	0.09	0.22	90.88
WBJV085D1	16.84	17.84	3.35	1.00	3.34	0.81	0.16	0.22	0.07	0.12	0.48	4.52	0.09	0.18	89.10
WBJV087D0	192.49	193.59	3.24	1.10	3.35	1.54	0.20	0.46	0.07	0.08	0.45	5.55	0.11	0.21	98.01
WBJV087D2	7.29	8.31	3.27	1.02	4.75	1.68	0.24	0.38	0.09	0.17	0.65	7.05	0.11	0.22	90.88
WBJV087D3	12.37	13.50	3.29	1.13	2.83	0.72	0.14	0.16	0.06	0.10	0.42	3.84	0.08	0.15	100.68
WBJV090D0	152.45	153.49	3.23	1.04	0.69	0.64	0.06	0.08	0.03	0.04	0.16	1.47	0.03	0.11	92.67
WBJV090D1	12.37	13.51	3.22	1.14	0.34	0.23	0.02	0.05	0.02	0.03	0.12	0.63	0.02	0.07	101.57
WBJV090D2	17.76	18.77	3.26	1.01	0.82	0.76	0.06	0.09	0.03	0.04	0.18	1.73	0.04	0.10	89.99
WBJV092D0	279.50	280.72	3.17	1.22	1.06	0.52	0.14	0.06	0.03	0.05	0.20	1.77	0.02	0.07	108.70
WBJV092D1	19.07	20.14	3.27	1.07	0.46	0.21	0.04	0.05	0.02	0.03	0.13	0.75	0.04	0.12	95.34
WBJV092D2	24.54	25.62	3.33	1.08	0.84	0.32	0.06	0.13	0.03	0.04	0.18	1.34	0.06	0.15	96.23
WBJV093D0	399.96	401.17	3.25	1.21	1.42	0.71	0.07	0.28	0.04	0.06	0.25	2.47	0.07	0.15	107.81
WBJV100D2	26.12	27.32	3.08	1.20	3.51	1.55	0.21	0.27	0.07	0.13	0.50	5.54	0.07	0.16	106.92
WBJV108D1	42.24	43.25	3.11	1.01	11.82	4.06	0.52	1.17	0.21	0.39	1.50	17.56	0.13	0.29	89.99
WBJV113D0	411.96	412.99	3.08	1.03	0.04	0.03	0.01	0.02	0.01	0.00	0.01	0.11	0.02	0.06	91.77
WBJV113D1	10.40	11.48	3.05	1.08	0.16	0.10	0.02	0.05	0.01	0.01	0.03	0.33	0.03	0.08	96.23
WBJV113D2	18.19	19.44	3.13	1.25	0.07	0.06	0.01	0.02	0.01	0.00	0.02	0.15	0.02	0.07	111.38
WBJV114D0	355.90	357.07	3.03	1.17	4.13	2.06	0.22	0.61	0.08	0.10	0.57	7.02	0.19	0.36	104.25
WBJV115D2	30.00	31.21	3.00	1.21	0.10	0.05	0.01	0.02	0.01	0.01	0.06	0.18	0.02	0.04	107.81
WBJV117D0	345.78	347.03	3.01	1.25	0.02	0.01	0.01	0.01	0.01	0.00	0.00	0.05	0.00	0.03	111.38
WBJV136D0	488.72	490.97	3.23	2.25	7.08	2.56	0.55	0.11	0.13	0.24	0.93	10.30	0.04	0.17	200.48
WBJV136D1	5.57	7.98	3.23	2.41	5.43	2.43	0.50	0.08	0.10	0.19	0.73	8.44	0.02	0.14	214.73
WBJV136D2	10.32	13.05	3.34	2.73	8.11	3.23	0.52	0.49	0.15	0.27	1.05	12.36	0.13	0.29	243.24
WBJV140D1	7.88	8.90	3.25	1.02	10.22	3.97	0.46	0.41	0.18	0.34	1.30	15.06	0.15	0.40	90.88
WBJV140D3	17.75	18.89	3.15	1.14	2.08	1.22	0.12	0.21	0.05	0.08	0.33	3.63	0.10	0.17	101.58
WBJV171D0	299.67	301.62	3.32	1.95	6.52	3.51	0.50	0.26	0.12	0.22	0.86	10.79	0.09	0.19	173.74
WBJV171D1	24.61	26.51	3.28	1.90	13.24	6.71	1.08	0.43	0.23	0.43	1.67	21.46	0.14	0.40	169.29
WBJV171D2	30.39	32.12	3.31	1.73	8.53	3.23	0.59	0.36	0.15	0.28	1.10	12.71	0.12	0.23	154.14
WBJV172D0	379.46	381.44	2.94	1.98	4.43	2.31	0.23	0.33	0.09	0.16	0.61	7.30			176.42
WBJV175D0	240.12	241.30	3.29	1.18	2.33	0.89	0.13	0.20	0.06	0.09	0.36	3.55			105.14

WBJV175D1	34.29	35.29	3.24	1.00	2.27	0.78	0.19	0.11	0.05	0.09	0.35	3.35			89.10
WBJV175D2	50.24	51.54	3.27	1.30	1.16	0.49	0.08	0.14	0.04	0.05	0.22	1.88			115.83
WBJV179D0	334.25	335.46	3.29	1.21	4.20	1.51	0.23	0.36	0.09	0.15	0.58	6.29			107.81

RW = Resource Width

Table - UG2 Reef - Resource Cut

BHID	FROM	TO	SG	LENGTH	PT	PD	RH	AU	OS	IR	RU	4E	CU%	NI%	RW
	m	m	t/m ^{3}	m	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	%	%	cm
WBJV001D0	473.20	475.60	3.65	2.40	0.51	0.17	0.11	0.00	0.05	0.06	0.35	0.79	0.01	0.04	228
WBJV001D1	25.30	27.80	3.59	2.50	0.30	0.11	0.08	0.00	0.05	0.05	0.29	0.50	0.00	0.04	238
WBJV001D2	53.17	55.47	3.53	2.30	0.44	0.16	0.10	0.00	0.05	0.06	0.33	0.71	0.01	0.04	219
WBJV003D0	536.60	537.70	3.67	1.10	2.53	0.85	0.35	0.02	0.12	0.20	0.88	3.75	0.01	0.09	105
WBJV003D1	82.89	83.99	3.71	1.10	1.99	1.18	0.28	0.03	0.10	0.16	0.74	3.48	0.01	0.12	105
WBJV003D2	186.30	187.30	3.63	1.00	0.40	0.12	0.09	0.00	0.05	0.05	0.32	0.62	0.00	0.08	95
WBJV010D1	84.50	86.60	3.75	2.10	0.48	0.23	0.10	0.02	0.05	0.06	0.34	0.83	0.01	0.07	200
WBJV015D0	434.00	435.24	3.78	1.24	2.67	1.14	0.35	0.04	0.13	0.21	0.92	4.20	0.01	0.09	118
WBJV015D1	77.10	78.31	3.75	1.21	2.93	0.97	0.35	0.02	0.13	0.23	0.99	4.26	0.01	0.08	115
WBJV032D0	360.95	362.11	3.74	1.16	2.96	1.18	0.36	0.03	0.13	0.23	0.99	4.52	0.01	0.08	110
WBJV032D1	113.25	114.45	3.73	1.20	3.58	1.27	0.42	0.02	0.15	0.27	1.16	5.29	0.00	0.08	114
WBJV033D1	55.50	56.54	3.35	1.04	0.81	0.27	0.04	0.01	0.06	0.08	0.43	1.13	0.00	0.05	99
WBJV033D2	59.43	60.42	3.45	0.99	1.02	0.63	0.13	0.01	0.07	0.10	0.48	1.79	0.01	0.04	94
WBJV035D0	517.04	519.11	3.73	2.07	1.01	0.23	0.14	0.01	0.07	0.10	0.48	1.38	0.00	0.06	197
WBJV035D1	48.24	50.50	3.64	2.26	0.36	0.21	0.09	0.01	0.05	0.05	0.31	0.67	0.01	0.05	215
WBJV041D0	537.70	539.14	3.12	1.44	0.02	0.02	0.01	0.01	0.04	0.03	0.22	0.06			137
WBJV041D1	60.37	61.58	3.07	1.21	0.06	0.01	0.01	0.01	0.03	0.02	0.03	0.10	0.00	0.03	115
WBJV043D0	574.50	575.50	3.29	1.00	0.66	0.28	0.07	0.01	0.06	0.07	0.39	1.03	0.00	0.06	95
WBJV043D1	63.93	65.18	3.80	1.25	0.48	0.43	0.11	0.01	0.05	0.06	0.34	1.03	0.00	0.06	119
WBJV044D0	500.47	503.45	3.81	2.98	0.52	0.20	0.12	0.01	0.06	0.06	0.35	0.87	0.01	0.06	283
WBJV044D1	30.16	33.25	3.81	3.09	0.46	0.31	0.08	0.01	0.05	0.06	0.34	0.86	0.00	0.06	294
WBJV045D0	573.68	575.41	3.75	1.73	3.16	1.46	0.48	0.01	0.14	0.24	1.05	5.12	0.00	0.06	165
WBJV045D1	73.43	74.94	3.66	1.51	2.52	1.00	0.35	0.01	0.12	0.21	0.87	3.88	0.01	0.08	144
WBJV046D0	544.48	545.78	3.71	1.30	2.74	1.07	0.33	0.02	0.13	0.21	0.94	4.15	0.01	0.08	124
WBJV046D1	64.41	65.75	3.72	1.34	3.05	1.40	0.44	0.06	0.13	0.23	1.02	4.95	0.01	0.07	127
WBJV048D0	478.21	479.94	3.75	1.73	2.08	0.40	0.36	0.01	0.11	0.17	0.76	2.85	0.01	0.05	165
WBJV049D0	550.64	551.85	3.30	1.21	0.24	0.07	0.06	0.01	0.03	0.04	0.15	0.37	0.00	0.02	115
WBJV050D0	591.47	592.67	3.65	1.20	3.35	1.43	0.56	0.04	0.14	0.24	1.07	5.38	0.01	0.09	114
WBJV050D1	96.56	97.67	3.74	1.11	3.18	1.58	0.51	0.05	0.15	0.25	1.10	5.33	0.01	0.07	106
WBJV103D0	446.88	448.47	3.95	1.59	3.23	1.15	0.50	0.03	0.14	0.25	1.06	4.91	0.01	0.07	151
WBJV104D0	564.46	567.02	3.64	2.56	1.21	0.46	0.20	0.01	0.08	0.11	0.53	1.88	0.01	0.07	243
WBJV104D1	89.00	91.04	3.83	2.04	2.16	0.91	0.40	0.02	0.11	0.17	0.78	3.48	0.01	0.09	194
WBJV104D2	94.93	97.36	3.60	2.43	1.54	0.65	0.24	0.02	0.09	0.13	0.62	2.45	0.01	0.07	231
WBJV109D1	92.65	94.70	3.59	2.05	2.43	0.64	0.33	0.02	0.12	0.19	0.85	3.41	0.01	0.09	195
WBJV109D2	98.18	99.61	3.75	1.43	3.54	1.60	0.46	0.03	0.15	0.27	1.15	5.64	0.04	0.08	136
WBJV112D0	502.14	503.21	3.86	1.07	2.61	0.83	0.40	0.02	0.12	0.20	0.90	3.86	0.01	0.08	102
WBJV112D2	77.07	78.27	3.60	1.20	1.75	0.83	0.33	0.01	0.10	0.15	0.67	2.92	0.01	0.08	114
WBJV116D0	562.88	564.05	3.90	1.17	2.42	1.53	0.35	0.03	0.12	0.19	0.85	4.33	0.02	0.08	111
WBJV116D1	72.37	73.43	4.01	1.06	3.12	1.24	0.50	0.02	0.14	0.24	1.04	4.88	0.01	0.09	101
WBJV116D2	78.49	79.67	4.24	1.18	2.92	0.88	0.57	0.02	0.13	0.22	0.98	4.39	0.01	0.09	112
WBJV120D0	375.10	379.12	3.94	4.02	2.58	1.04	0.39	0.01	0.12	0.20	0.89	4.02	0.01	0.07	382
WBJV120D3	68.25	69.88	3.90	1.63	3.23	1.46	0.52	0.02	0.14	0.25	1.07	5.23	0.01	0.08	155

WBJV120RDRIL	65.03	66.73	3.92	1.70	2.61	0.77	0.39	0.02	0.12	0.20	0.90	3.79	0.02	0.09	162
WBJV124D0	540.29	541.77	3.96	1.48	2.67	1.13	0.48	0.01	0.13	0.21	0.92	4.30	0.00	0.09	141
WBJV124D1	60.00	61.38	3.98	1.38	3.15	1.41	0.56	0.02	0.14	0.24	1.05	5.15	0.01	0.08	131
WBJV124D3	80.60	81.80	4.09	1.20	3.17	1.01	0.52	0.01	0.14	0.24	1.05	4.72	0.01	0.08	114
WBJV128D1	52.17	53.89	3.95	1.72	2.79	1.40	0.49	0.02	0.13	0.22	0.95	4.70	0.01	0.08	164
WBJV128D2	56.64	58.91	3.90	2.27	1.93	0.84	0.38	0.02	0.10	0.16	0.72	3.17	0.01	0.10	216
WBJV128D3	62.93	64.99	3.98	2.06	4.77	2.89	0.74	0.06	0.19	0.35	1.47	8.46	0.01	0.10	196
WBJV137D0	522.93	524.26	3.89	1.33	2.79	1.09	0.46	0.02	0.13	0.22	0.95	4.35	0.01	0.08	126
WBJV137D1	103.77	105.16	3.99	1.39	2.89	1.14	0.44	0.02	0.13	0.22	0.97	4.48	0.01	0.09	132
WBJV141D0	381.98	383.07	4.05	1.09	3.35	0.84	0.57	0.01	0.15	0.25	1.10	4.76	0.01	0.09	104
WBJV141D1	52.43	53.41	4.13	0.98	3.83	1.40	0.62	0.02	0.16	0.29	1.22	5.88	0.01	0.08	93
WBJV142D0	462.47	466.02	3.96	3.55	2.96	1.42	0.42	0.04	0.13	0.23	0.99	4.84	0.03	0.10	338
WBJV142D1	70.83	73.60	4.06	2.77	2.72	1.26	0.47	0.04	0.13	0.21	0.93	4.49	0.02	0.10	263
WBJV143D1	27.28	28.36	3.21	1.08	1.07	0.48	0.17	0.01	0.07	0.10	0.49	1.73	0.01	0.04	103
WBJV145D0	558.51	559.61	3.13	1.10	1.03	0.54	0.06	0.07	0.07	0.10	0.49	1.70	0.02	0.05	105
WBJV145D1	39.72	40.83	3.94	1.11	1.88	0.67	0.33	0.02	0.10	0.15	0.71	2.90	0.01	0.10	106
WBJV145D2	43.26	44.36	3.39	1.10	1.27	0.45	0.19	0.03	0.08	0.11	0.55	1.94	0.01	0.07	105
WBJV153D0	549.98	550.89	4.26	0.91	3.26	1.07	0.54	0.04	0.14	0.25	1.07	4.91	0.01	0.11	87
WBJV153D1	45.09	46.42	4.08	1.33	2.99	1.39	0.48	0.05	0.14	0.23	1.00	4.91	0.03	0.13	126
WBJV154D4	53.31	54.39	3.13	1.08	0.82	0.28	0.15	0.01	0.07	0.08	0.43	1.26	0.00	0.03	103
WBJV156D0	685.20	686.58	3.93	1.38	2.30	0.54	0.38	0.02	0.11	0.18	0.82	3.24	0.00	0.07	131
WBJV156D1	66.60	68.83	3.96	2.23	3.43	1.21	0.45	0.02	0.15	0.26	1.12	5.11	0.01	0.08	212
WBJV170D0	267.32	268.32	3.45	1.00	2.89	1.09	0.47	0.02	0.13	0.22	0.98	4.46	0.00	0.04	95
WBJV170D1	22.77	23.93	3.74	1.16	3.31	1.38	0.51	0.02	0.15	0.25	1.09	5.22	0.00	0.07	110
WBJV178D0	331.13	332.56	3.68	1.43	2.48	0.86	0.39	0.02	0.12	0.20	0.87	3.74			136
WBJV178D1	51.44	53.35	3.70	1.91	4.74	2.28	0.70	0.04	0.19	0.35	1.46	7.76			182
WBJV002D0	555.90	557.60	3.80	1.70	2.02	0.72	0.28	0.01	0.10	0.16	0.75	3.04	0.00	0.12	158
WBJV002D1	105.11	106.11	3.79	1.00	2.12	0.74	0.30	0.01	0.11	0.17	0.77	3.17	0.01	0.10	93
WBJV002D2	16.66	17.86	3.81	1.20	2.21	0.73	0.31	0.01	0.11	0.18	0.80	3.25	0.01	0.09	111
WBJV005D0	483.90	485.70	3.79	1.80	0.50	0.19	0.11	0.00	0.05	0.06	0.35	0.80	0.00	0.08	167
WBJV012D0	69.90	71.00	3.32	1.10	0.11	0.05	0.02	0.01	0.04	0.03	0.24	0.19	0.00	0.03	102
WBJV013D1	124.11	125.21	3.81	1.10	0.26	0.07	0.07	0.01	0.05	0.04	0.28	0.41	0.01	0.06	102
WBJV016D1	41.90	43.30	3.80	1.40	2.14	0.55	0.29	0.02	0.11	0.17	0.78	3.00	0.00	0.07	130
WBJV018D1	45.02	46.42	3.54	1.40	1.67	0.68	0.25	0.02	0.09	0.14	0.66	2.62	0.00	0.09	130
WBJV020D0	96.30	97.50	3.57	1.20	0.55	0.04	0.10	0.01	0.06	0.06	0.36	0.70	0.01	0.06	111
WBJV020D1	26.50	27.60	3.67	1.10	1.32	0.12	0.29	0.01	0.08	0.12	0.56	1.74	0.01	0.07	102
WBJV021D0	280.50	281.70	3.67	1.20	3.80	1.66	0.41	0.05	0.16	0.28	1.22	5.91	0.03	0.10	111
WBJV021D1	89.85	90.85	3.66	1.00	2.18	0.75	0.26	0.03	0.11	0.17	0.79	3.22	0.01	0.11	93
WBJV024D0	282.96	283.96	3.24	1.00	0.79	0.44	0.09	0.02	0.06	0.08	0.42	1.34	0.00	0.05	93
WBJV024D1	63.00	64.00	3.41	1.00	0.96	0.56	0.10	0.03	0.07	0.09	0.47	1.65	0.01	0.07	93
WBJV025D0	121.48	123.17	3.74	1.69	2.76	0.85	0.33	0.02	0.13	0.21	0.94	3.97	0.02	0.10	157
WBJV025D1	40.20	42.68	3.77	2.48	3.40	2.63	0.37	0.08	0.15	0.26	1.11	6.49	0.02	0.12	230
WBJV027D1	58.03	59.03	3.13	1.00	0.23	0.09	0.05	0.01	0.05	0.04	0.28	0.38	0.00	0.04	93
WBJV027D2	99.33	100.35	3.64	1.02	0.29	0.11	0.06	0.01	0.05	0.05	0.29	0.47	0.01	0.07	95
WBJV028D0	221.91	224.65	3.74	2.74	3.11	1.59	0.37	0.05	0.14	0.24	1.03	5.12	0.01	0.09	254
WBJV028D1	71.64	74.21	3.76	2.57	4.37	2.46	0.38	0.07	0.18	0.32	1.37	7.29	0.01	0.11	238
WBJV034D0	478.44	479.60	3.37	1.16	0.18	0.11	0.04	0.01	0.04	0.04	0.26	0.34	0.00	0.07	108
WBJV034D1	48.34	49.34	3.24	1.00	0.16	0.05	0.05	0.01	0.04	0.04	0.26	0.27	0.00	0.04	93
WBJV037D0	46.06	47.06	3.71	1.00	2.90	1.16	0.35	0.05	0.13	0.22	0.98	4.47	0.01	0.07	93
WBJV039D0	136.99	137.99	3.37	1.00	0.50	0.14	0.06	0.01	0.05	0.06	0.34	0.71	0.00	0.04	93

WBJV040D0	433.12	434.14	3.12	1.02	0.04	0.02	0.01	0.01	0.04	0.03	0.22	0.08	0.00	0.03	95
WBJV040D1	61.75	62.75	3.20	1.00	0.21	0.11	0.05	0.01	0.05	0.04	0.27	0.38	0.00	0.04	93
WBJV047D0	47.52	48.52	3.20	1.00	0.42	0.27	0.06	0.01	0.05	0.06	0.33	0.77	0.00	0.04	93
WBJV054D0	337.40	338.47	3.40	1.07	1.02	0.35	0.13	0.02	0.05	0.07	0.29	1.52	0.00	0.11	99
WBJV054D1	17.38	18.50	3.29	1.12	1.48	0.66	0.23	0.02	0.06	0.11	0.49	2.38			104
WBJV055D1	28.85	30.00	3.65	1.15	0.50	0.05	0.14	0.01	0.09	0.08	0.55	0.69	0.00	0.07	107
WBJV057D1	71.70	72.80	3.42	1.10	0.90	0.30	0.14	0.01	0.04	0.06	0.29	1.35			102
WBJV058D1	37.95	39.12	3.56	1.17	0.37	0.14	0.12	0.01	0.08	0.06	0.43	0.63			108
WBJV059D0	200.62	203.22	3.77	2.60	0.46	0.13	0.14	0.01	0.08	0.08	0.53	0.74			241
WBJV068D0	267.02	268.27	3.61	1.25	1.79	0.82	0.30	0.02	0.09	0.15	0.68	2.93	0.00	0.07	116
WBJV068D1	26.38	27.74	3.77	1.36	2.74	1.00	0.45	0.01	0.11	0.20	0.88	4.20	0.00	0.06	126
WBJV070RDRIL	52.23	53.73	3.37	1.50	0.18	0.04	0.05	0.01	0.04	0.04	0.26	0.28	0.00	0.05	139
WBJV071D0	54.51	55.62	3.47	1.11	0.11	0.09	0.04	0.01	0.04	0.03	0.24	0.25	0.00	0.04	103
WBJV073D0	159.02	160.17	3.72	1.15	3.00	1.33	0.53	0.03	0.12	0.22	0.99	4.89	0.01	0.07	107
WBJV073D1	98.22	99.32	3.66	1.10	3.38	1.42	0.61	0.04	0.15	0.26	1.10	5.45	0.01	0.07	102
WBJV078D0	72.25	73.66	3.26	1.41	0.39	0.27	0.08	0.01	0.05	0.05	0.32	0.76	0.00	0.04	131
WBJV086D1	30.70	32.96	3.79	2.26	0.51	0.25	0.16	0.01	0.10	0.09	0.61	0.93	0.00	0.05	210
WBJV089D1	121.72	122.72	3.64	1.00	2.04	0.22	0.09	0.01	0.10	0.17	0.75	2.36	0.00	0.05	93
WBJV096D0	417.84	418.83	3.13	0.99	0.57	0.22	0.09	0.01	0.06	0.06	0.36	0.88	0.00	0.03	92
WBJV099D0	453.90	455.04	3.54	1.14	1.38	1.28	0.26	0.04	0.08	0.12	0.58	2.96	0.01	0.07	106
WBJV102D0	467.13	468.23	3.40	1.10	0.97	0.62	0.14	0.02	0.07	0.09	0.47	1.76	0.01	0.09	102
WBJV102D2	122.34	123.49	3.07	1.15	0.72	0.24	0.13	0.02	0.06	0.08	0.40	1.10	0.00	0.03	107
WBJV122D0	473.29	474.93	3.78	1.64	2.45	1.10	0.48	0.02	0.12	0.19	0.86	4.05	0.01	0.09	152
WBJV130D1	84.70	85.74	3.91	1.04	2.63	0.94	0.38	0.03	0.12	0.21	0.91	3.98	0.01	0.09	96
WBJV130D2	88.69	89.85	3.86	1.16	2.59	0.71	0.41	0.01	0.12	0.20	0.90	3.72	0.01	0.07	108
WBJV133D0	527.26	528.47	3.01	1.21	0.67	0.18	0.09	0.01	0.06	0.07	0.39	0.95	0.00	0.03	112
WBJV133D1	27.36	28.36	2.96	1.00	0.41	0.16	0.06	0.01	0.05	0.05	0.32	0.64	0.00	0.03	93
WBJV139D2	23.40	24.40	3.08	1.00	0.02	0.02	0.01	0.01	0.04	0.03	0.22	0.06	0.00	0.03	93
WBJV007D0	255.70	256.80	3.73	1.10	2.27	0.66	0.22	0.03	0.11	0.18	0.81	3.18	0.00	0.07	98
WBJV008D0	324.10	325.30	3.52	1.20	0.60	0.24	0.12	0.01	0.06	0.07	0.37	0.97	0.00	0.06	107
WBJV008D1	102.35	103.55	3.71	1.20	1.51	0.60	0.19	0.01	0.09	0.13	0.61	2.32	0.00	0.10	107
WBJV009D0	279.70	281.10	3.77	1.40	0.37	0.09	0.08	0.01	0.05	0.05	0.31	0.55	0.00	0.07	125
WBJV009D3	46.10	47.50	3.74	1.40	0.61	0.17	0.13	0.01	0.06	0.07	0.37	0.92	0.00	0.06	125
WBJV014D0	247.40	248.40	3.69	1.00	0.33	0.10	0.08	0.01	0.05	0.05	0.30	0.52	0.00	0.07	89
WBJV014D1	47.20	48.20	3.44	1.00	0.17	0.05	0.03	0.01	0.04	0.04	0.26	0.26	0.00	0.04	89
WBJV022D0	99.13	100.95	3.79	1.82	0.14	0.06	0.04	0.01	0.04	0.04	0.25	0.25	0.00	0.05	162
WBJV022D1	38.69	40.61	3.55	1.92	0.38	0.08	0.08	0.01	0.05	0.05	0.31	0.55	0.00	0.06	171
WBJV023D0	201.75	204.50	3.64	2.75	1.85	0.61	0.28	0.02	0.10	0.15	0.70	2.76	0.01	0.07	245
WBJV038D2	67.00	69.10	3.02	2.10	0.19	0.07	0.02	0.01	0.04	0.04	0.26	0.29			187
WBJV052D0	190.63	191.72	3.47	1.09	0.60	0.19	0.05	0.01	0.04	0.05	0.19	0.85	0.00	0.05	97
WBJV053D1	45.90	46.90	3.52	1.00	0.26	0.10	0.07	0.01	0.05	0.05	0.32	0.44	0.00	0.07	89
WBJV053D2	53.49	54.71	3.71	1.22	0.36	0.30	0.10	0.02	0.06	0.05	0.42	0.77	0.00	0.06	109
WBJV060D0	248.46	249.78	3.58	1.32	2.30	0.81	0.38	0.02	0.09	0.16	0.72	3.51			118
WBJV060D1	49.37	50.86	3.76	1.49	3.87	0.87	0.48	0.03	0.16	0.27	1.24	5.24			133
WBJV064D0	242.53	245.02	3.65	2.49	0.55	0.11	0.14	0.01	0.08	0.08	0.50	0.81			222
WBJV064D1	33.84	35.79	3.67	1.95	0.54	0.13	0.15	0.01	0.10	0.09	0.64	0.82			174
WBJV065D0	315.77	316.91	3.25	1.14	0.15	0.09	0.05	0.01	0.03	0.02	0.17	0.29			102
WBJV065D1	31.19	32.37	3.20	1.18	0.07	0.03	0.02	0.01	0.03	0.02	0.06	0.13			105
WBJV067D0	375.25	378.34	3.72	3.09	3.48	1.36	0.50	0.02	0.15	0.26	1.10	5.36	0.00	0.07	275
WBJV083D0	181.10	182.37	3.76	1.27	0.62	0.10	0.18	0.01	0.06	0.07	0.38	0.90	0.00	0.04	113

WBJV083D1	51.31	52.69	3.81	1.38	0.51	0.33	0.15	0.01	0.06	0.06	0.35	1.01	0.00	0.06	123
WBJV084D1	69.97	71.09	3.75	1.12	2.93	0.98	0.38	0.03	0.13	0.23	1.00	4.32	0.01	0.08	100
WBJV085D0	508.65	510.16	3.74	1.51	2.76	0.86	0.42	0.01	0.13	0.21	0.94	4.05	0.00	0.06	135
WBJV085D1	57.62	59.16	3.72	1.54	3.16	1.20	0.46	0.01	0.14	0.24	1.05	4.83	0.00	0.06	137
WBJV100D0	407.06	409.70	3.50	2.64	1.67	0.67	0.24	0.01	0.09	0.14	0.65	2.59	0.01	0.06	235
WBJV105D0	450.36	451.44	3.39	1.08	1.40	0.45	0.20	0.01	0.08	0.12	0.58	2.06	0.01	0.06	96
WBJV105D2	96.31	97.31	3.14	1.00	0.33	0.06	0.08	0.02	0.05	0.05	0.30	0.49	0.01	0.06	89
WBJV108D0	421.76	423.27	3.62	1.51	1.94	0.53	0.32	0.02	0.10	0.16	0.72	2.80	0.01	0.08	135
WBJV108D2	96.84	98.00	3.68	1.16	3.06	1.19	0.36	0.03	0.14	0.23	1.02	4.65	0.02	0.09	103
WBJV113D0	429.19	430.46	3.39	1.27	1.36	0.30	0.18	0.01	0.08	0.12	0.57	1.85	0.01	0.06	113
WBJV113D1	26.10	28.00	3.28	1.90	1.33	0.29	0.17	0.01	0.08	0.12	0.56	1.80	0.01	0.06	169
WBJV113D2	33.46	34.65	3.24	1.19	0.79	0.24	0.06	0.01	0.06	0.08	0.42	1.10	0.00	0.05	106
WBJV115D0	435.31	438.35	3.72	3.04	1.39	0.33	0.21	0.01	0.08	0.12	0.58	1.95	0.01	0.07	271
WBJV115D1	49.76	53.06	4.10	3.30	3.64	1.59	0.52	0.04	0.16	0.27	1.17	5.79	0.01	0.08	294
WBJV115D2	65.66	69.39	4.20	3.73	3.49	1.53	0.51	0.03	0.15	0.26	1.13	5.55	0.01	0.09	332
WBJV117D0	369.19	370.39	3.43	1.20	1.88	0.58	0.27	0.01	0.10	0.15	0.71	2.75	0.01	0.07	107
WBJV118D0	478.33	479.61	3.54	1.28	1.20	0.59	0.22	0.01	0.08	0.11	0.53	2.02	0.01	0.07	114
WBJV118D1	49.53	50.73	3.42	1.20	1.58	0.33	0.21	0.01	0.09	0.13	0.63	2.14	0.00	0.06	107
WBJV121D0	387.17	388.70	4.07	1.53	3.89	2.31	0.68	0.07	0.16	0.29	1.24	6.95	0.02	0.11	136
WBJV129D2	62.68	63.89	3.89	1.21	2.77	0.98	0.40	0.02	0.13	0.22	0.95	4.16	0.00	0.08	108
WBJV138D0	390.62	392.03	3.96	1.41	2.73	0.83	0.45	0.01	0.13	0.21	0.93	4.02	0.01	0.08	126
WBJV138D2	57.23	59.01	3.99	1.78	3.10	0.71	0.45	0.01	0.14	0.24	1.03	4.27	0.00	0.09	159
WBJV140D1	35.62	36.89	3.29	1.27	1.17	0.38	0.19	0.01	0.08	0.11	0.52	1.75	0.00	0.06	113
WBJV140D2	40.65	41.75	3.49	1.10	1.83	0.60	0.30	0.01	0.10	0.15	0.70	2.74	0.00	0.06	98
WBJV171D1	72.03	73.08	3.71	1.05	2.22	0.60	0.37	0.02	0.11	0.18	0.80	3.22	0.02	0.09	94
WBJV171D2	77.82	79.04	3.51	1.22	1.98	1.21	0.25	0.04	0.10	0.16	0.74	3.48	0.02	0.08	109
WBJV174D0	387.65	389.14	3.58	1.49	2.70	1.26	0.46	0.02	0.13	0.21	0.93	4.45			133
WBJV174D2	70.06	71.50	3.77	1.44	3.17	1.12	0.53	0.02	0.14	0.24	1.05	4.84			128
WBJV177D0	328.06	329.29	3.31	1.23	1.17	0.40	0.15	0.06	0.08	0.11	0.52	1.78			110
WBJV179D0	350.25	352.20	3.65	1.95	2.45	1.28	0.43	0.03	0.12	0.19	0.86	4.19			174

Item 19(e): Spatial Statistics

Variography

Variograms are a useful tool for investigating the spatial relationships of samples. Variograms for channel width (cm), Pt, Pd, Rh, Au, 3PGE_Au, Os, Ir, Ru contents (cm.g/t), and Cu and Ni percents (%) were modelled during the estimation process. All variograms are omni-directional spherical semi-variograms.

Table 75 - and Table 74 - summarise the 4E content variograms, which have a modelled grade continuity range of ~180 - 300m for the Merensky Reef and a modelled grade continuity range of ~200 - 420 m for the UG2 Reef. The nugget effect is on average 37% of the sill or population variance for the Merensky Reef and 45% for the UG2 Reef. This is slightly high but it is expected that these ranges will improve as more data becomes available. No top-cuts were used for the generation of the experimental variograms. Parameters for the remaining elements are available but were omitted from the report

Table - Variogram Parameters Project 1A

Reef	Parameter	Geozone	Nugget (%)	Sill (%)	Range1 (m)	Range2 (m)	Range3 (m)	Sill 2 (%)	Range1 (m)	Range2 (m)	Range3 (m)
MRMC	CW	1	31	100	558	558	1	100			
MRMC	CW	2	34	74	190	190	1	100	593	593	1
MRMC	CW	3	31	100	558	558	1	100			
UG2MC	CW	1	39	100	813	813	1	100			
UG2MC	CW	2	39	100	812	812	1	100			
MRMC	4E (cmg/t)	1	27	100	370	370	1	100			
MRMC	4E (cmg/t)	2	31	70	134	134	1	100	299	299	1
MRMC	4E (cmg/t)	3	27	100	370	370	1	100			
UG2MC	4E (cmg/t)	1	49	100	259	259	1	100	744	744	1
UG2MC	4E (cmg/t)	2	49	100	258	258	1	100	744	743	1

Table - Variogram Parameters Project 1

Reef	Parameter	Geozone	Nugget (%)	Sill 1 (%)	Range1 (m)	Range2 (m)	Range3 (m)	Sill 2 (%)	Range1 (m)	Range2 (m)	Range3 (m)
MRMC	CW	1	31	100	558	558	1	100			
MRMC	CW	2	34	74	190	190	1	100	593	593	593
MRMC	CW	3	32	78	140	140	1	100	212	212	212
MRMC	CW	4	31	80	227	227	1	100	426	426	426
MRMC	CW	5	30	79	151	151	1	100	375	375	375
MRMC	CW	6	19	100	469	469	1	100			
MRMC	CW	7	20	100	300	300	1	100			
MRMC	CW	8	31	100	408	408	1	100			
UG2MC	CW	1	32	78	254	254	1	100	766	766	766
UG2MC	CW	2	10	100	395	395	1	100			
UG2MC	CW	3	30	100	449	449	1	100			
UG2MC	CW	4	36	100	260	260	1	100			
UG2MC	CW	5	35	100	393	393	1	100			
UG2MC	CW	6	24	100	325	325	1	100			
UG2MC	CW	7	41	100	268	268	1	100			
UG2MC	CW	8	29	100	342	342	1	100			
MRMC	CM3PG	1	35	100	521	521	1	100			
MRMC	CM3PG	2	42	80	247	247	1	100	390	390	1
MRMC	CM3PG	3	33	69	145	145	1	100	252	252	1
MRMC	CM3PG	4	28	74	272	272	1	100	565	565	1
MRMC	CM3PG	5	31	100	315	315	1	100			
MRMC	CM3PG	6	33	100	460	460	1	100			
MRMC	CM3PG	7	34	100	250	250	1	100			
MRMC	CM3PG	8	25	100	569	569	1	100			
UG2MC	CM3PG	1	40	81	259	259	1	100	744	744	744
UG2MC	CM3PG	2	35	100	469	469	1	100			
UG2MC	CM3PG	3	42	100	460	460	1	100			
UG2MC	CM3PG	4	28	100	242	242	1	100			
UG2MC	CM3PG	5	49	100	530	530	1	100			
UG2MC	CM3PG	6	38	100	257	257	1	100			
UG2MC	CM3PG	7	34	80	255	255	1	100	837	837	837
UG2MC	CM3PG	8	17	100	320	320	1	100			

Item 19(f): Modelling Overview

The geological channel width was modelled and estimated to supplement the Resource cut model and Mineral Resource estimates in the main geological report.

Three-dimensional (“3D”) wireframes (DTM's) were constructed from reef intersections representing the reef in 3D space. The reef wireframes were filled with a seam block model. The dip and dip-direction have been interpolated into the seam model. The dip was used to correct the reef intersection widths and tonnes. The first 50m of the ore body is considered to represent a weathered zone and is discarded in the modelling and estimation procedures.

2D simple and ordinary kriging methodologies were utilized using the in-house geostatistical package ‘Res’ with Datamine as the platform package.

Statistical analysis provided a basis for final data verification and was used to establish specific information on population distributions and checks for anomalous values.

Variograms were modelled for the channel width (cm), Pt, Pd, Rh, Au, 3PGE_Au, Os, Ir, Ru contents (cm.g/t), and Cu and Ni percents (%).

Resource cut width (cm) and channel width (cm) and Pt, Pd, Rh, Au, 3PGE_Au, Os, Ir, & Ru contents, and elements (Cu %, and Ni %) were estimated for both the Merensky and UG2 reefs.

Modelling Parameters

The available borehole data was obtained from PTM RSA. In the evaluation process, the metal content (4E cm.g/t) and reef width (cm) values are used. For the Resource cut model the reef width refers to the corrected Resource cut 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 Resource cut width values.

Grade Estimation

Full reef composite data - Resource cut width (cm) and Pt, Pd, Rh, Au, 3PGE_Au, Os, Ir, and Ru contents (cm.g/t), elements (Cu %, and Ni %) and SG were estimated for both the Merensky and UG2 reefs.

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 (cm.g/t) and Resource cut width (cm) values. Detailed checks were carried out 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, all blocks within a specific Geozone have been assigned a global mean (declustered mean) for that Geozone. The data available for Project 1 was combined with the additional boreholes for Project 1A and realistic means were generated for Project 1A estimation.

The following parameters were used in the kriging process for both Project areas:-

1. Full reef composite data - channel width / Resource cut width (cm) and content (Pt, Pd, Rh, Au, 3PGE_Au, Os, Ir, Ru) elements (Cu %, and Ni %) and SG;
2. 200m x 200m x 1m block size. Block models were constructed using split cells and not subcells due to the size of the parent blocks (200 X 200 X 1m);
3. Discretisation 5 x 5 x 1 for each 200m x 200m x 1m block;
4. First search volume - 750 m
    a. Minimum number of samples 4,
    b. Maximum number of samples 40;
5. Second search volume
    a. Minimum number of samples 2,
    b. Maximum number of samples 40;
6. Third search volume
    a. Minimum number of samples 1,
    b. Maximum number of samples 20;
7. Interpolation methods - simple kriging, ordinary kriging;
8. Local / global mean values used in the simple kriging process;

The first 50m of the ore body is considered to represent a weathered zone and is discarded in the modelling and estimation procedures. Figure 24 shows the block model plots (Project 1 & 1A) for the different parameters.

The kriged estimates were post-processed to calculate the information effect, dispersion variance and grade tonnage intervals. The 4E cut-off values used ranged from 100 - 600 cm.g/t. A dip model was created from the reef wireframe, and each individual block has an interpolated dip value and dip direction.

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 SMU's 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.

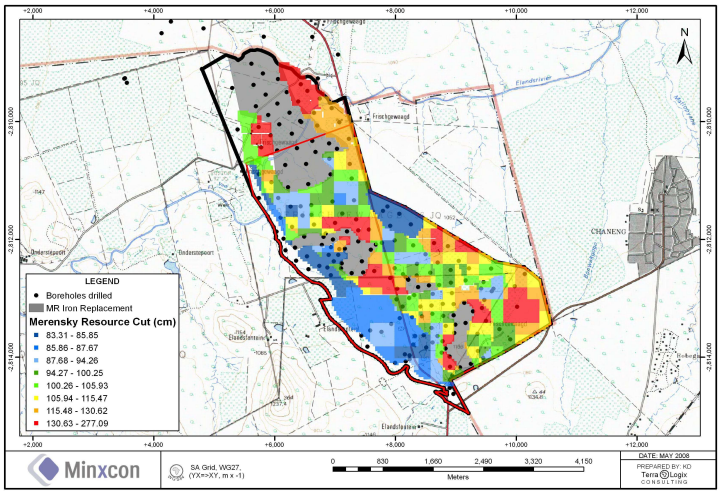
An SMU of 20m x 50m was selected with an expected future underground sampling configuration on a 20m x 20m grid. Information effects were calculated based on the SMU and the expected future production underground sampling configuration.

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. 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 SMU's) 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 cm.g/t.

Item 19(g): Model Plans and Sections

Figure 32 - MR and UG2 Width Plots - Resource Cut Model



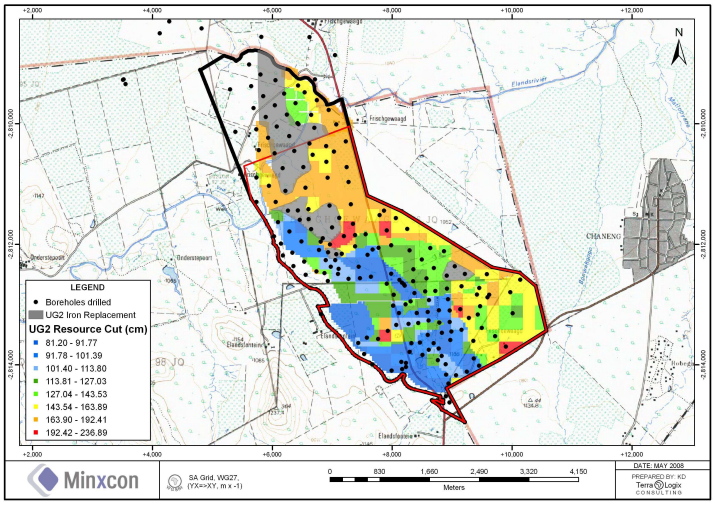
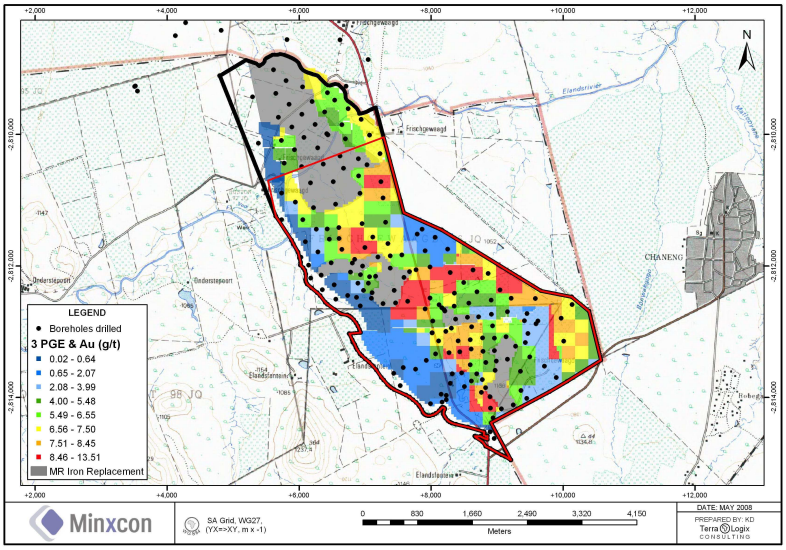


Figure 33 - UG2 and MR 4E Grades - Resource Cut Model



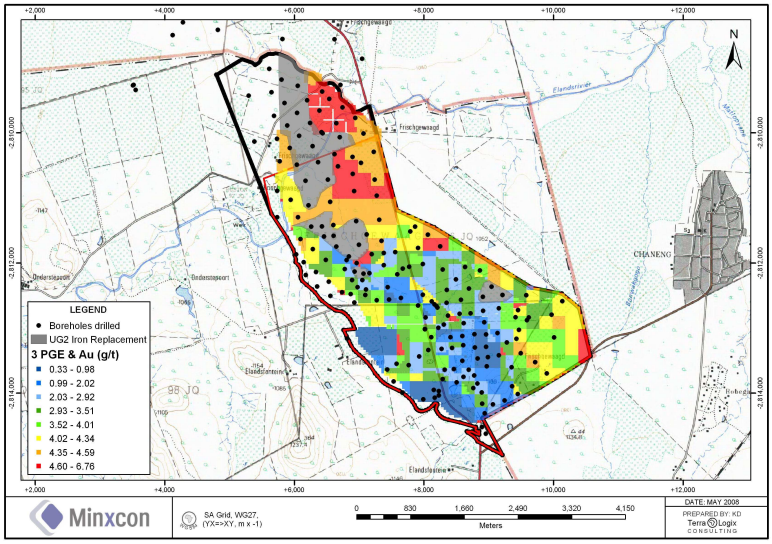
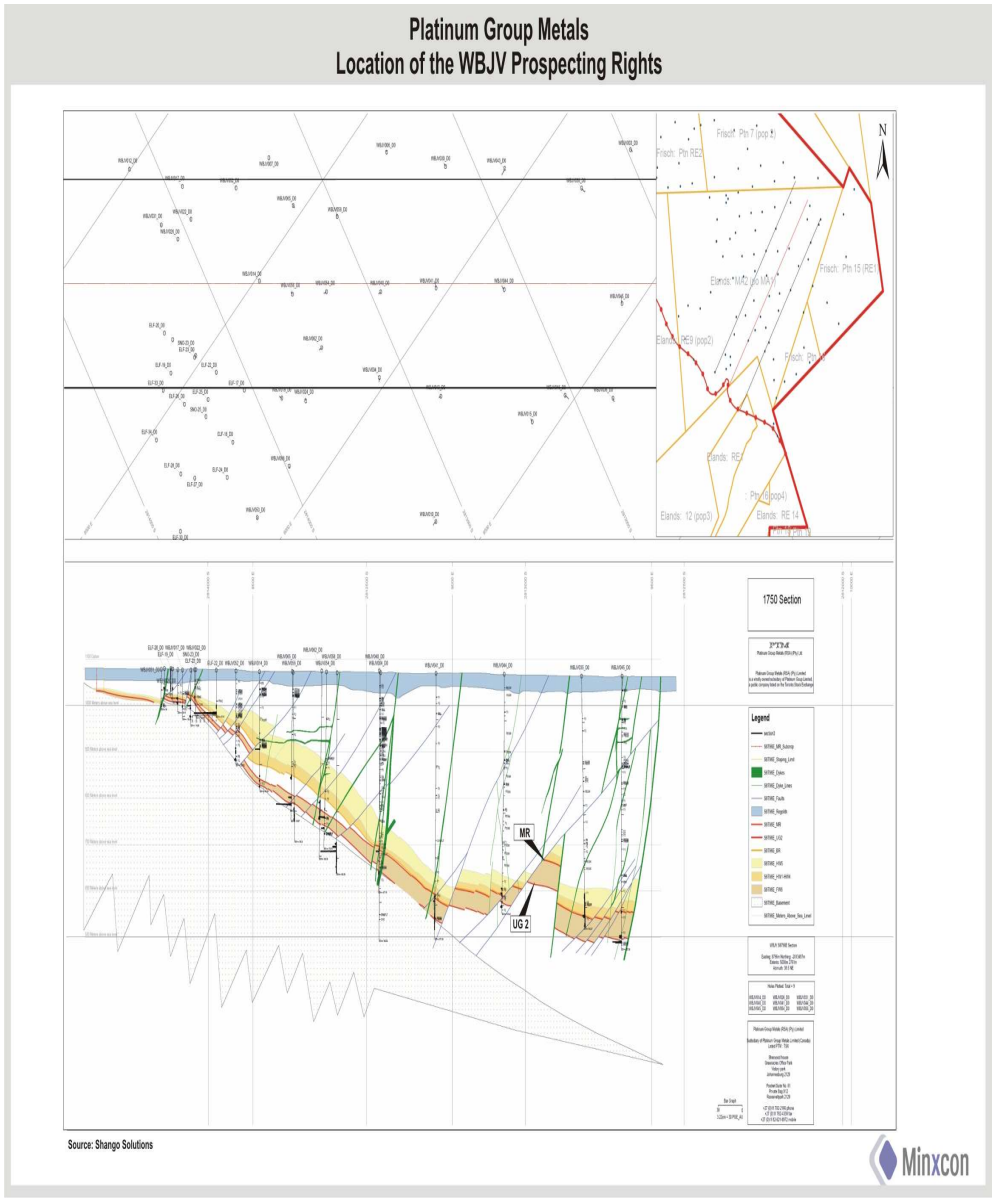


Figure 34 - Cross Section through Project 1



Item 19(h): 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 semi-variogram range and minimum of twenty 1m composited samples
    b. Indicated: at least 3 boreholes within semi-variogram range and a minimum of twelve 1m composite samples
    c. Inferred: less than 3 boreholes within the semi-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 (semi-variogram range)
    a. Measured : at least within 60% of semi - variogram range
    b. Indicated : within semi-variogram range
    c. Inferred: further than semi-variogram range
6. Lower Confidence Limit (blocks)
    a. Measured: < 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: > 40%
    b. Indicated: 20 - 40%
    c. Inferred: <20%
8. Deviation from lower 90% confidence limit (data distribution within Resource area considered for classification)
    a. Measured: <10% deviation from the mean
    b. Indicated: 10 - 20% deviation from the mean
    c. Inferred: >20% deviation from the mean

Using the above criteria, the Merensky and UG2 reefs within the Project Area were classified into Measured, Indicated and Inferred Mineral Resource categories. Measured, Indicated and Inferred Mineral Resources are classified, under the SAMREC Code, as follows.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which volume and/or tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and sampling and assumed but not verified geologically and/or through analysis of grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that may be limited in scope or of uncertain quality and reliability.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

The Indicated Mineral Resource has sufficient confidence for mine design, mine planning, and/or economic studies.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource, but has a higher level of confidence than that applying to an Inferred Mineral Resource.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and grade continuity.

A Measured Mineral Resource has sufficient confidence for mine design, mine planning, production planning, and/or detailed economic studies.

A Measured Mineral Resource requires that the nature, quality, amount and distribution of data are such as to leave no reasonable doubt in the opinion of the Competent Person(s), that the tonnage and grade of the mineralization can be estimated to within close limits and that any variation within these limits would not materially affect potential economic viability.

This category requires a high level of confidence in, and understanding of, the geology and the controls on mineralization.

‘Resource Cut’ Mineral Resources

Major Elements (4E)

Table - Resource Cut Mineral Resources (Measured) - Merensky Reef & UG2 Project 1 (100% WBJV Area)

Cut-Off 4E	Tonnage	Geo Loss	Tonnage	Grade 4E	Content 4E		Resource Cut Width
cmg/t	t	%	t	g/t	g	Moz	cm
MR Measured Project 1							
0	8 079 815	14	6 948 641	6.71	46 636 165	1.499	108
100	7 570 302	14	6 510 460	7.08	46 089 380	1.482	108
200	7 073 137	14	6 082 898	7.43	45 215 208	1.454	108
300	6 384 754	14	5 490 888	7.94	43 594 640	1.402	108
400	5 595 560	14	4 812 182	8.55	41 136 636	1.323	108
500	4 775 183	14	4 106 657	9.24	37 940 742	1.220	108
600	3 990 992	14	3 432 253	9.99	34 279 923	1.102	108
UG2 Measured Project 1							
0	11 214 700	23	8 635 319	3.31	28 598 095	0.919	141
100	10 324 150	23	7 949 596	3.54	28 132 887	0.904	141
200	9 577 904	23	7 374 986	3.70	27 282 257	0.877	141
300	8 492 261	23	6 539 041	3.91	25 586 430	0.823	141
400	6 927 889	23	5 334 475	4.23	22 566 945	0.726	141
500	5 225 893	23	4 023 938	4.62	18 588 189	0.598	141
600	3 708 056	23	2 855 203	5.05	14 405 165	0.463	141

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 MR	64%	5.08	27%	2.14	4%	0.318	5%	0.398
Project 1 UG2	63%	2.46	26%	1.02	10%	0.39	1%	0.04

Table - Resource Cut Mineral Resources (Indicated) - Merensky Reef & UG2 Project 1 (100% WBJV Area)

Cut-Off 4E	Tonnage	Geo Loss	Tonnage	Grade 4E	Content 4E		Resource Cut Width
cmg/t	t	%	t	g/t	g	Moz	cm
MR Indicated Project 1							
0	20 200 910	14	17 372 783	5.29	91 876 308	2.954	109
100	16 032 610	14	13 788 045	6.52	89 901 112	2.890	109
200	14 303 610	14	12 301 105	7.11	87 484 927	2.813	109
300	12 574 180	14	10 813 795	7.75	83 795 242	2.694	109
400	10 876 640	14	9 353 910	8.44	78 962 503	2.539	109
500	9 328 631	14	8 022 623	9.16	73 469 181	2.362	109
600	7 971 142	14	6 855 182	9.88	67 716 517	2.177	109
UG2 Indicated Project 1							
0	30 986 290	23	23 859 443	3.43	81 767 529	2.629	134
100	28 510 310	23	21 952 939	3.67	80 585 286	2.591	134
200	26 326 710	23	20 271 567	3.85	78 121 246	2.512	134
300	22 680 620	23	17 464 077	4.13	72 192 078	2.321	134
400	17 944 430	23	13 817 211	4.51	62 344 873	2.004	134
500	13 493 680	23	10 390 134	4.93	51 250 913	1.648	134
600	9 827 620	23	7 567 267	5.38	40 712 239	1.309	134

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 MR	64%	4.96	27%	2.09	4%	0.31	5%	0.39
Project 1 UG2	63%	2.60	26%	1.08	10%	0.41	1%	0.04

Table - Resource Cut Mineral Resources (Inferred) - Merensky Reef & UG2 Project 1 (100% WBJV Area)

Cut-Off 4E	Tonnage	Geo Loss	Tonnage	Grade 4E	Content 4E		Resource Cut Width
cmg/t	t	%	t	g/t	g	Moz	cm
MR Inferred Project 1							
0	1 849 534	14	1 590 599	1.64	2 614 920	0.084	93
100	471 251	14	405 276	5.17	2 096 171	0.067	93
200	313 467	14	269 582	6.93	1 867 380	0.060	93
300	251 896	14	216 630	7.95	1 721 176	0.055	93
400	205 623	14	176 836	8.90	1 573 589	0.051	93
500	166 983	14	143 606	9.87	1 417 955	0.046	93
600	135 003	14	116 103	10.87	1 261 782	0.041	93
UG2 Inferred Project 1							
0	4 052 591	23	3 120 495	3.56	11 106 229	0.357	134
100	3 306 020	23	2 545 635	4.23	10 777 291	0.346	134
200	3 160 682	23	2 433 725	4.35	10 596 819	0.341	134
300	3 002 029	23	2 311 562	4.47	10 334 004	0.332	134
400	2 690 964	23	2 072 042	4.70	9 734 111	0.313	134
500	2 256 234	23	1 737 300	5.04	8 749 344	0.281	134
600	1 785 266	23	1 374 655	5.46	7 502 749	0.241	134

Table - Resource Cut Mineral Resources (Inferred) - Merensky Reef & UG2 Project 1A (100% WBJV Area)

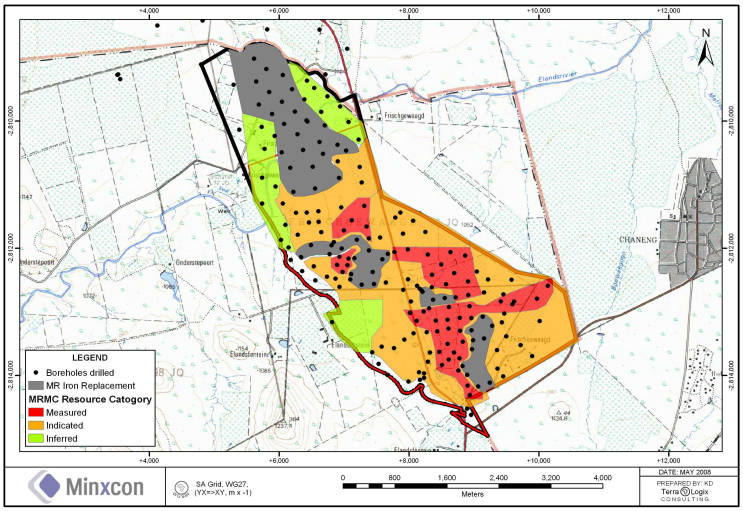
Cut-Off 4E	Tonnage	Geo Loss	Tonnage	Grade 4E	Content 4E		Resource Cut Width
cmg/t	t	%	t	g/t	g	Moz	cm
MR Inferred Project 1a							
0	2 797 251	14	2 405 635	5.22	12 567 057	0.404	115
100	2 382 003	14	2 048 522	6.10	12 490 183	0.402	115
200	2 342 388	14	2 014 454	6.18	12 441 314	0.400	115
300	2 175 732	14	1 871 129	6.48	12 127 388	0.390	115
400	1 890 982	14	1 626 245	7.00	11 390 901	0.366	115
500	1 560 803	14	1 342 291	7.67	10 298 330	0.331	115
600	1 244 445	14	1 070 223	8.43	9 020 630	0.290	115
UG2 Inferred Project 1a							
0	3 476 682	23	2 989 946	4.98	14 888 600	0.479	157
100	3 476 682	23	2 989 946	4.98	14 888 600	0.479	157
200	3 476 306	23	2 989 623	4.98	14 888 199	0.479	157
300	3 457 089	23	2 973 097	5.00	14 858 499	0.478	157
400	3 325 753	23	2 860 148	5.10	14 590 132	0.469	157
500	2 986 850	23	2 568 691	5.34	13 722 902	0.441	157
600	2 472 994	23	2 126 775	5.71	12 138 423	0.390	157

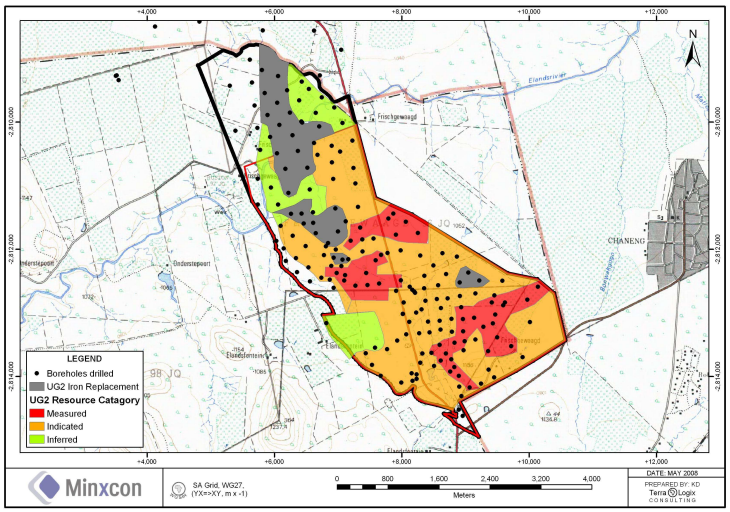
Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 MR	64%	5.09	27%	2.15	4%	0.32	5%	0.32
Project 1 UG2	63%	2.82	26%	1.16	10%	0.45	1%	0.04
Project 1A MR	64%	4.15	27%	1.75	4%	0.26	5%	0.32
Project 1A UG2	63%	3.15	26%	1.30	10%	0.50	1%	0.05

Notes:
1. MR = Merensky Reef; UG2 = Upper Group No. 2 chromitite seam; PGE = Platinum Group Elements.
2. The cut-offs for Inferred Mineral Resources have been established by a qualified person after a review of potential operating costs and other factors.
3. Results have been rounded for reporting purposes.
4. A 14% Geological Loss was applied to the MR, and 23% Geological Loss to the UG2.
5. Mineral Resources have been quoted as INCLUSIVE of Mineral Reserves.
6. 4E pertains to 3PGE + Au grade.

Figure 35 - Location of Measured, Indicated and Inferred Resources

Z





Item 19(i): Conclusions and Recommendations

Measured, Indicated and Inferred Mineral Resource estimates have been calculated for the Merensky Reef and UG2 Chromitite seam from available borehole information. The Merensky Reef and UG2 Chromitite seam were both divided into eight distinct geozones based on facies with specific lithological and mineralised characteristics.

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 Chromitite 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.

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 regular QA&QC process carried out by PTM RSA 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 drilling will have to be carried out in order to increase the confidence in the Resource estimate in the old Project Area 1A.

For the Inferred Mineral Resource category to be potentially upgradeable, infill drilling needs to be carried out. 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.

Whilst no further work was planned for Project Area 1 as it was understood that sufficient Indicated and Measured Resources had been delineated to commence a Feasibility Study. Additional drilling has occurred in the ore body and this has resulted in an updated Resource statement being press released, but this updated Resource was **NOT** used for the Updated Feasibility Study Technical Report.

Item 19(j): Audits and Reviews

The Mineral Resource estimation process was reviewed by two independent geostatisticians, namely Ms Carina Lemmer of Geological and Geostatistical Services (August 2006) and Mr D Gray of Snowden (July 2007). The independent reviews referred to were completed on the mineral Resource estimate completed to support the Pre-Feasibility Study and were subsequently considered and addressed in the Updated Feasibility Study. The following sections summarise the findings of these reviews:-

Review by C Lemmer

Ms Lemmer challenged the fact that Mr Muller used the SK method of estimation rather than the OK method. Mr Muller carried out the modelling using the OK method of estimation and found the following:-

- MR Measured Resources have a 0.82% higher grade for SK vs. OK, -5.4% for tonnes and -0.05% for metal content;
- MR Indicated Resources show a -3.2% lower grade for SK vs. OK and -0.19% difference in metal content;

She concluded that the differences (using the SK vs. OK method) at a 300cm.g/t cut-off is not significant, especially so for the Measured and Indicated Resources.

Ms Lemmer suggested that as the histograms created by Mr Muller had normal distributions superimposed on them, that the distribution curves be removed if the histograms were to be included in the Updated Feasibility Study Report.

Furthermore, Ms Lemmer was concerned about the substantial amount of kurtosis figures listed in the stats columns that were negative, which she felt could not be correct. Mr Muller undertook to look into the origin of the negative numbers, which could have resulted from distributions that were not properly defined with gaps between intervals and limited data. Negative values represent problem distributions and Mr Muller stated that care should be taken in any statistical analysis for those parameters.

Ms Lemmer found the variograms to be problematic with the currently relatively sparse data and the number of geological domains. She recommended that Mr Muller experiment with calculating log variograms standardised to a sill of one. When back transforming the variogram spherical fits to real space, the adjustment is so slight that one might even use the log variogram fits when kriging real space variables. She suggested that improved variograms would lend considerable credibility to the Updated Feasibility Study Document. Mr Muller agreed to the comments regarding variograms and stated that he had used log variograms to determine the nugget and ranges where possible.

With regard to post processing Ms Lemmer considered that post processing based on a log normal assumption was not appropriate for the project. She stated that if it could not be changed, then it would be prudent to point it out in the Updated Feasibility Study and comment on the fact that there would be upside to the recoverable Resources when applying a COG. Mr Muller stated that although not 100% log normal, the population is closer to a log normal than a normal distribution and the net effect on results are expected to be minimal.

Ms Lemmer was concerned about the use of an SMU that would fit into the panels. She suggested that the use of an SME of 25m x 25m instead of 30m x 40m would show an improvement in recoverable Resources. Mr Muller commented that he had considered the 25m x 25m SMU and that the net effect was minimal.

Finally, Ms Lemmer commented that the criterion of 90% lower confidence limit was based on a lognormal assumption and was therefore not appropriate. Mr Muller undertook to remove the criteria per block basis.

Review by D Gray (Snowden)

The following comments were made by Snowden regarding the Resource modelling at the WBJV by PTM RSA (Muller):-

- Snowden was unable to verify the statistic provided by PTM RSA from the drill hole data provided and suggested that consideration of classical statistics for the individual reef and footwall layers would yield considerably different statistical properties. Snowden commented that the footwall mineralisation typically has higher skewed distribution compared to the more normal distributions commonly observed for the MR population. They suggested that the impact of these populations on each other should be noted and tested. Snowden noted that the footwall mineralisation is not well developed and is highly skewed tot the low value end. They also noted that there were a significant proportion of assay values that have values of <0.02g/t, and that there was uncertainty as to whether these are trace values or values assigned due to insufficient sample.
- Snowden recommended that in addition to the 3PGE+Au and the individual prill grades were to be modelled for validation purposes and for providing a more robust estimate of the prill split proportions.
- Snowden noted that the use of SK for sparsely spaced datasets is common practice. They note that estimation parameters used are likely to over smooth the estimate, providing a false sense of the true variability in the Resource. Variogram ranges did not exceed 600m. Snowden suggested that a detailed kriging neighbourhood analysis be completed with the available data and selected variogram models in order to optimise the Resource estimation parameters.
- Snowden suggested that conditional simulation be considered as an alternative to the post processing technique employed by Mr Muller as conditional simulation has the ability to provide a quantitative error on the estimate per block and which incorporates the ore body’s spatial continuity. Snowden noted that there was no substitute for the collection of additional data.
- With regard to Resource estimate validation, Snowden suggested they are documented;
- With regard to the Resource Classification, Snowden recommended that a comprehensive risk assessment of key criteria influencing the Resource Estimate be considered so as to conclusively support the Resource classification. WRT the current classification, the Indicated Resources are delineated in areas where the closest drill hole is approximately 1km away. Similarly, the shallow Indicated Resource areas are questioned with respect to their poor continuity between drill holes. Considering the high degree of variability that this Resource displays in its stratigraphic sequence, reef behaviour and classical statistics, Snowden consider the classification optimistic.

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(k): Reserve Calculation

The historical Resource statement for the orebody is contained below. Only Indicated and Measured Resources are included in the overall mine plan.

Table -Merensky Reef Resource Statement

Cut-Off 3PGE_Au	Tonnage	Geo Loss	Tonnage	4E	Content 4E		Reef Width
cmg/t	t	%		g/t	g	Moz	cm
MR Measured							
0	8,079,815	14	6,948,641	6.71	46,636,165	1.499	108
MR Indicated							
0	20,200,910	14	17,372,783	5.29	91,876,308	2.954	109
MR TOTAL							
0	28,280,725	14	24,321,424	5.70	138,512,473	4.453	109

Table -UG2 Reef Resource Statement

Cut-Off 3PGE_Au	Tonnage	Geo Loss	Tonnage	4E	Content 4E		Reef Width
cmg/t	t	%		g/t	g	Moz	cm
UG2 Measured							
0	11,214,700	23	8,635,319	3.31	28,598,095	0.919	141
UG2 Indicated							
0	30,986,290	23	23,859,443	3.43	81,767,529	2.629	134
UG2 Total							
0	42,200,990	23	32,494,762	3.40	110,365,624	3.548	136

A planning pay limit was established for both reefs using estimates of working costs and revenue. This resulted in a stope face pay limit of 3.5 g/t for both Merensky and UG2.

The Mineral Reserve statement has been calculated based on the outcome of the Updated Feasibility Study economic evaluation and are detailed in the following table:

Table - Mineral Reserve Statement

Mineral Reserves 4E

Merensky				UG2			
Tonnage	4E	Content 4E		Tonnage	4E	Content 4E	
t	g/t	Tonne	Moz	t	g/t	Tonne	Moz
Merensky Proved				UG2 Proved			
6,677,304	5.61	37.478	1.205	5,086,535	3.37	17.126	0.551
Merensky Probable				UG2 Probable			
11,333,294	5.44	61.677	1.983	8,448,801	3.41	28.831	0.927
Total Merensky Mineral Reserves				Total UG2 Mineral Reserves			
18,010,598	5.51	99.155	3.188	13,535,336	3.40	45.957	1.478

To convert the above Reserve Statement to individual elements, the Resource Prill Splits have been applied to the 4E content to create the list of individual elements depicted in Table 83 . The individual element list as depicted in this table do not constitute a Reserve on an individual basis, only when referred to as 4E is it possible to report the Reserve.

Table - Listing of Individual Elements from 4E Reserve Statement

Metal	Reef	Category	Prill Split	Tonnage Tonnes	g/t 4E	4E Content Tonne	4E Content Moz
Pt	Merensky	From Proved 4E Reserve	64%	6,677,304	3.59	23.99	0.771
		From Probable 4E Reserve	64%	11,333,294	3.48	39.47	1.269
		Total	**64%**	**18,010,598**	**3.52**	**63.46**	**2.040**
	UG2	From Proved 4E Reserve	63%	5,086,535	2.12	10.79	0.347
		From Probable 4E Reserve	63%	8,448,801	2.15	18.16	0.584
		Total	**63%**	**13,535,336**	**2.14**	**28.95**	**0.931**
Pd	Merensky	From Proved 4E Reserve	27%	6,677,304	1.52	10.12	0.325
		From Probable 4E Reserve	27%	11,333,294	1.47	16.65	0.535
		Total	**27%**	**18,010,598**	**1.49**	**26.77**	**0.861**
	UG2	From Proved 4E Reserve	26%	5,086,535	0.88	4.45	0.143
		From Probable 4E Reserve	26%	8,448,801	0.89	7.50	0.241
		Total	**26%**	**13,535,336**	**0.88**	**11.95**	**0.384**
Rh	Merensky	From Proved 4E Reserve	4%	6,677,304	0.22	1.50	0.048
		From Probable 4E Reserve	4%	11,333,294	0.22	2.47	0.079
		Total	**4%**	**18,010,598**	**0.22**	**3.97**	**0.128**
	UG2	From Proved 4E Reserve	10%	5,086,535	0.34	1.71	0.055
		From Probable 4E Reserve	10%	8,448,801	0.34	2.88	0.093
		Total	**10%**	**13,535,336**	**0.34**	**4.60**	**0.148**
Au	Merensky	From Proved 4E Reserve	5%	6,677,304	0.28	1.87	0.060
		From Probable 4E Reserve	5%	11,333,294	0.27	3.08	0.099
		Total	**5%**	**18,010,598**	**0.28**	**4.96**	**0.159**
	UG2	From Proved 4E Reserve	1%	5,086,535	0.03	0.17	0.006
		From Probable 4E Reserve	1%	8,448,801	0.03	0.29	0.009
		Total	**1%**	**13,535,336**	**0.03**	**0.46**	**0.015**

The prill splits are the same percentages as for the Measured and Indicated Resources. The effective date of the Reserve estimate is June 30, 2008.

As the result of the updated Resource statement being released (which was not used for this Technical Report), it is anticipated that a further iteration of the mine plan will be completed in the near future, with a modified Mineral Reserve statement being evaluated.

ITEM 20: OTHER RELEVANT DATA AND INFORMATION

The economic viability of mineral Resources declared in this Updated Feasibility Study has been demonstrated and thus the Reserve has been estimated based on the financial and working cost estimates are applied to the Resource in Item 19.

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 Reserve 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 Qualified Person 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 Qualified Person, reasonable and realistic prospects for eventual economic extraction.

The definitions of each of the Reserves and Resource categories can be found under Item 19(h).

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 figure 17. The structure then allows for specific structurally related blocks, to be allocated a Resource estimate.

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

ITEM 21: ADDITIONAL REQUIREMENT FOR MINE DEVELOPMENT

A Feasibility Study on this project was completed and published in July 2008, with a Revised Feasibility Study published in June 2009. Since the release of the Feasibility Study, an optimization process has taken place resulting in changes to various design aspects of the planned mine. These changes have in turn resulted in different financial outcomes for the project.

The main changes to access methods and production rates are as follows:-

- Block development previously 30 metres in the footwall has been sited closer to reef at 20 metres in the footwall.
- Multiple orepasses per raise line have been replaced by a single ore pass.
- Loading of reef tonnes by LHD to truck has been replaced by loading from the stope orepasses directly on to trucks.
- Reef drive travelling ways on each level have been eliminated.

- The overall constraint of 140,000 reef tonnes per month has been removed. The resulting maximum tonnage is approximately 55,000 reef tonnes per decline system, or 160,000 reef tonnes per month overall.
- More UG 2 reef is mined as a result of better extraction rates.

The overall on-reef / hand held methods described below have not changed significantly from the previous study. The assumption made in this Technical Report mining section is that all development necessary to access reef is completed in time to meet the mine production schedule. A further assumption is that all requirements, as defined by this Technical Report stoping section (men, material, ventilation and services), are delivered to the stope or timber bay in the crosscut as required. All rock broken on the reef horizon and scraped to the in-stope ore pass is removed immediately.

Doubts over South African electric power generation authority, ESKOM and its ability to provide the necessary bulk grid power have been somewhat relieved and this has resulted in reduced dependence upon self generated power requirements.

Item 21(a): Geological & Geotechnical Factors affecting On-Reef Mine Design

A detailed description of the geology of the total mine area is contained in Item 19.

The main findings in the geological and rock engineering investigations that influenced on-reef mine design are discussed below:

- The dip of the reefs, combined with the narrow reef widths, preclude mechanized, on-reef mining methods
- The Merensky reef has an average dip of 19 degrees with variations from 9 degrees to 30 degrees and an average stoping width over the life of the mine of 116cm
- The UG2 reef has an average dip of 16 degrees, varying from flat to 33 degrees. The average stope width of the UG2 over the life of the mine is 155cm
- Certain mining blocks (such as Block 17), have the potential for increased mechanization if the currently predicted dip, structure and width are confirmed by additional drilling, either from surface or from underground with stope definition drilling
- The average dip of the reef also influenced the choice of level spacing - to have a reasonable back length between levels while not having excessive raise lengths to develop, a vertical level spacing of 60 metres was selected giving an average back length of 184 metres
- Significant faults and dykes intersect the orebody and subdivide the deposit into a number of discrete mining blocks, each of which requires access development on different mining levels. The resultant blocks of ground left un-mined add to the regional stability of the mine

After application of appropriate pay limits, the Merensky reef contains twice as much recoverable metal as the UG2 and is therefore the primary target. The parting between the Merensky reef and the UG2 varies from contact in the south to 70 metres in the north. Mining of both reefs generally only occurs when the parting is greater than 30 metres.

Item 21(b): Rock Engineering

Regional Stability

The regional and panel pillar design is based upon a detailed understanding of the structure and nature of the rockmass resulting from inspection and evaluation of representative exploration drilling core, coupled with numerical modeling of planned mining layouts. Geotechnical information obtained from neighbouring mines and knowledge of similar operations elsewhere in the Bushveld Complex has also been incorporated.

On the Merensky reef horizon, twenty-five meter wide stope panels will be advanced on breast between up to seven meter wide in-panel pillars sited immediately down-dip of the strike scraper gullies. These pillars will operate within their elastic limits and will carry the full overburden load down to a depth of 300m below surface. Below this depth, five to six metre wide yield pillars and three metre wide crush pillars have been designed. On the UG2 horizon, panel lengths vary from 20 to 25 metres depending on site specific rock engineering recommendations. Pillars are designed to carry the full overburden load down to a depth of 300m below surface. Below this depth, six to eight metre wide yield pillars and three metre wide crush pillars will be left.

In view of the structural complexity of the orebody, and the provisions that have been made in the mining model to acknowledge the significant loss of Reserves attributable to these geological losses, no attempt has been made in this study to include additional support in the form of regular regional pillars as it is believed that the natural geological losses will adequately meet the regional support requirement. This assertion has been tested, and verified, by numerically modeling.

Local Stope Stability

Based upon detailed geotechnical assessment of the hangingwall, a maximum resistance of 185kN/m ^{2} is considered adequate to meet local stope support requirements for the majority of planned operations. Because the support will need to be stiff and preferably pro-active, large (180mm to 200mm diameter) pre-stressed elongate units will be used. These units will be spaced 1.5m apart on dip and strike, together with breaker lines of three-stick cluster props installed 1.5m apart on dip at 6m strike intervals (i.e. every fourth row). These breaker lines will also serve to contain over-runs should collapses occur in the back-area. In-stope bolting on a 1.5m x 1.5m pattern has also been included in the overall stope support design strategy for this project. Strike and dip gullies will be supported by means of three-stick cluster props installed at 1.5m centers along their edges, and a 1m pattern of tensioned rockbolts across the hangingwall span.

Localised problem areas requiring additional support may be addressed by either increasing the elongate and/or rock bolt support density, or, where conditions so dictate, leaving small internal pillars. In situations of potential massive beam failure, or of excessive ride in more steeply dipping stoping environments, stiff grout-based or composite concrete packs may be needed to augment elongate units that are prone to buckling-type failure under such circumstances. These eventualities have not been factored into the stope support design for this feasibility exercise.

Stability of Access Tunnels

Mining layouts comprise horizontal access tunnels sited 20m in the footwall of the reef. This is well within geotechnical criteria that have been determined for the project by numerical modeling. Separate and independent footwall access will in general be provided for Merensky and UG2 infrastructure respectively. Consideration will need to be given to alternative access strategies in situations where the close proximity of the reefs renders this approach either impossible, impractical or wasteful.

Evaluation of the rockmass quality based upon borehole information, suggests that not only will access tunnels be sited in a variety of differing rock types, but the rock types themselves are likely to exhibit a wide spread of quality and competence characteristics. As a result, support demand is likely to range across a broad spectrum from little or no support to intense support. This is likely to be exacerbated by moderately high (horizontal) stress considerations at the deeper levels. Standard tunnel support design charts will be used to match prevailing and anticipated rockmass condition ratings with corresponding support requirements. To cater for this anticipated wide range of tunnel and development support requirements, a system of support categories has been devised, which can be applied to estimate support requirements based upon available geotechnical information.

Primary access to the mine is to be provided by a series of declined shafts that will target the south, central and north sections of the mine respectively. A preliminary geotechnical evaluation of the rockmass through which the declines will pass, and of the surface area in which the boxcuts and portals will be situated, was undertaken by examining relevant sections of core from existing nearby boreholes. This indicated that extremely weathered material extends down to a depth of 29m below surface. Below this, however, there is a rapid transition to intact, unweathered and competent rock. Based upon this information, it is expected that the boxcuts will need to be excavated to a depth of at least 35m to provide for stable portal undercuts.

The decline tunnels themselves will, for the most part, generally be situated in a very competent rockmass. However, they will, on occasion, pass through alternating bands of good quality, moderate quality and poor quality rock. Support requirements for these different sections will vary from minimal bolt patterns to moderately intense bolt patterns with appropriate fabric linings. Extremely weathered lamprophyre dyke intrusions are noted, and such intersections will require reinforced shotcrete applications or concrete lining.

Eleven raise-bored ventilation winzes of various sizes will be situated across the property to service mining operations. A similar evaluation of rockmass conditions at these sites was undertaken by examining the core from existing nearby boreholes. The depth of weathered material that will require consolidation was determined for each of the individual holes. These depths range from about 11m to 28m and up to as much as 55m in one instance. This information has been factored into the engineering design of these holes. Weak zones that will occur within the barrels were also identified and characterized by examination of the drill core. Because of the prohibitive cost implications of providing access into the winzes for support purposes, winze sites will be evaluated and chosen to minimize the potential for shaft sidewall failure, and, in the event that such failure does occur and that it adversely affects the design performance of the shaft, the affected shaft will simply be abandoned and an alternative replacement shaft will be drilled at a lesser cost than would have been required to support the original shaft.

Industry-accepted rock mechanics criteria relating to the size, spacing, layout and juxtaposition of tunnels, chambers, dams, bins and passes have been incorporated into the detailed design of major service excavations associated with the declined shafts. Appropriate support measures will need to be instituted in those instances where the host rock is weak or structurally disturbed. Nevertheless, no untoward geotechnical problems are anticipated provided that stoping activities in the vicinity of such excavations do not interfere with their integrity.

Multi-reef Mining

The separation between the Merensky reef and the UG2 reef varies from only a few metres in the shallow areas close to the outcrop, increasing to many tens of metres down-dip and across the property. Superimposed, multi-reef mining of both the Merensky reef and the UG2 reef will occur in only a few limited areas of the mine, and here, depending upon the separation distance between the two horizons, additional consideration will need to be given to aspects such as the layout and siting of the footwall access tunnels, the order and sequences of stoping operations on the respective reef horizons, as well as any special precautionary stope support requirements.

Independent access layouts will be provided for the Merensky and portions of the UG2 reefs, but where possible the Merensky infrastructure is used to access the UG2. . In practice, this aspect will require careful consideration to optimize mining operations, and access efficiencies, and to avoid full or partial sterilization of some Reserves. The study also envisages exploitation of the Merensky reef as the primary objective of the project strategy. Mining of the payable areas of UG2 will take place on a less substantive scale. There is thus a danger that early mining on the Merensky horizon may be undertaken with less than due regard for possible mining of the UG2 reef in the future, perhaps resulting in partial or complete sterilisation of portions of this Reserve.

Geotechnical Overview

The exploration drilling program has revealed that the orebody is structurally complex, and exhibits a large degree of variability in terms of key geotechnical aspects such as stratigraphy, reef separation, reef dip, rockmass integrity, faulting, jointing, and weathering. In practice, this will no doubt elicit a similar range of differing mining strategies to deal with the changing circumstances occasioned by this variability. Nevertheless, the basic mining strategies adopted and proposed in this feasibility study are considered to be entirely representative of the strategies that are likely to be adopted in practice, and are entirely appropriate to this level of study.

Item 21(c): Stope Design Features

Based on preferred practice on adjoining mines a system of scattered breast stoping was decided on for the WBJV. This method is described below:-

- Panels are 25 metres long for Merensky reef and 20 to 25 metres long for UG2 reef measured on dip, advancing in the direction of strike
- Strike gullies are mined at between 5 and16 degrees above strike for purposes of drainage and fault negotiation
- In situ reef support pillars are left below the strike gullies
- The in-situ pillars vary in width dependent on depth below surface. For Merensky reef, the pillar width varies between 3 and 7 metres and for the UG2 reef, the width varies from 4 to 9 metres.

The pillar dimensions for different depths on the Merensky reef horizon are depicted below.

Table - Pillar Dimensions

Depth	SG	Vertical Stress	Mining Height	Holing Width	Pillar Length	Pillar Width	Pillar W:H Ratio	Effect Pillar Width	Panel Span	Panel Length	Bulk Strngth	Pillar Stmgth	Pillar Stress	Safety Factor	% Extract
50	3.0	1.5	1.30	2.0	6.0	4.0	3.1	4.8	25	29	40	72	14	5.1	90
100	3.0	2.9	1.30	2.0	6.0	4.0	3.1	4.8	25	29	40	72	28	2.5	90
150	3.0	4.4	1.30	2.0	6.0	4.0	3.1	4.8	25	29	40	72	43	1.7	90
200	3.0	5.9	1.30	2.0	6.0	5.0	3.8	5.5	25	30	40	77	47	1.6	88
250	3.0	7.4	1.30	2.0	6.0	6.0	4.6	6.0	25	31	40	80	51	1.6	85
300	3.0	8.8	1.30	2.0	6.0	7.0	5.4	6.5	25	32	40	84	54	1.6	84
350	3.0	10.3	1.30	2.0	6.0	5.2	4.0	5.6	25	30	40	78	80	1.0	87
400	3.0	11.8	1.30	2.0	6.0	5.8	4.5	5.9	25	31	40	80	83	1.0	86
450	3.0	13.2	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	165	0.4	92
500	3.0	14.7	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	183	0.4	92
550	3.0	16.2	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	201	0.3	92
600	3.0	17.7	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	220	0.3	92
650	3.0	19.1	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	238	0.3	92
700	3.0	20.6	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	256	0.3	92
750	3.0	22.1	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	275	0.2	92
800	3.0	23.5	1.30	2.0	6.0	3.0	2.3	4.0	25	28	40	66	293	0.2	92

The pillars are not continuous on strike allowing for the flow of ventilation between panels.

Additional support in the form of 200mm elongates (pipe sticks) and roof bolts are included in the mine design. The roof bolts are drilled using customized rigs to ensure vertical drilling of the roof bolt holes from a position of safety for the operator.

The panels and gullies are to be drilled by means of 2.2kW electric hand held rock drills rather than with conventional pneumatic rock drills. This technology has been refined, mostly by AngloGold Ashanti and Angloplats, over the last eight years, to the point where whole shafts are only using electric drills for stoping purposes. As a result of visits to sites using the electric rock drills and discussion with operators, it was decided to include these rock drills in the stope planning and infrastructure design. There are significant power consumption benefits from this technology and their inclusion has significantly reduced the power consumption for the entire mine. Whilst this technology is relatively new, the project team does not regard their inclusion as a risk to the project. Hydraulic rockdrills can also be considered for the mine in preference to reverting to compressed air.

Numerous published papers on the implementation of electric rock drills, as well as extensive discussions with the intended suppliers, were the basis for the inclusion of electric stope rock drills.

The blast holes are charged with ‘Anfo’ type explosives and are detonated using capped fuse and igniter cord. The use of shock tube technology will be considered once it proves itself economically viable.

Broken rock is cleaned by scraper winch to the strike gullies and then scraped on strike, to the centre gully (raise). In the centre gully, the reef is either tipped directly into an ore pass to the level below, or scraped down the centre gully to the first ore pass available.

Raises are situated 150 metres apart on strike and are accessed from the strike haulage by cross cut and inclined travelling way.

A working face is planned, equipped and resourced to advance at 11 metres per month. An average raise line would contain 10 to 12 panels and for every 10 panels that are equipped for mining, approximately 5 will be available at any one time.

For the Merensky reef, if each working panel advances at a rate of 11 metres per month, the raise line will produce 1,375m ^{2} (5 panels x 25m x 11m). This equates to approximately 5,250 panel tonnes per producing raise line.

A production unit (half level) consists of two such producing raise lines, one ledging and equipping raise line and one raise line being vamped and reclaimed. The ventilation model is based on supplying through

ventilation to four raise lines per production unit. In addition the development necessary to sustain production on that half level must also be ventilated, equipped and resourced.

Each full section can be expected to produce 10,500 panel reef tonnes per month with 14 producing sections capable of producing 147,000 panel tonnes per month. With ledging accounting for an additional 8%, total reef tonnes from 14 sections will amount to approximately 160,000 before dilution. With the extra 18% dilution factored into the mining equation, the theoretical capacity of the mine is 188,000 tons per month. However, at any one time resources allocated to production will be involved in re-establishment of panels. Therefore, the diluted reef tonnage is expected to be 160,000 tonnes per month. The total production plan and cost estimate for the mine is based on this relationship.

Item 21(d): Ventilation

The mine will be divided into discrete ventilation districts incorporating various mining blocks, these will be created by naturally occurring geological features. In total there will be eleven districts each returning air to surface by dedicated raise bored ventilation (RBH) shafts equipped with fans. RBHs and fans have been located and sized to cater for the ultimate production with sufficient allowance for concurrent development.

Multi-blast development will be accommodated during initial decline and haulage development. Thereafter blasting will be on fixed-time-blast basis with maximum two blasts per day. For stoping the ventilation strategy will be to re-use air in a cascade system with most air re-used but with some leaked back through worked out areas.
The conveyor decline will be used as an intake airway but will be equipped with an early warning monitoring system. Mobile refuge chambers will be used during capital development of the decline systems with permanent refuge chambers established in line with production requirements. Early Warning Systems throughout the mine will be in compliance with the South African Mine Health and Safety Act.

Each of the three decline clusters will have the capacity to handle approximately 250 kg/s ventilation air, which based on the production schedule, will be sufficient to dilute diesel exhaust and other pollutants and to remove heat without the need for refrigeration, although heat tolerance screening of underground workers will be required.

Raised bored holes and main fan installations will be needed early in the establishment of the mining blocks and CAPEX will need to be committed to ensure that fans are installed immediately after the raise bore holes are complete. The estimate for CAPEX of major ventilation components is approximately R100m, this figure includes main and auxiliary equipment but excludes the cost of the raised bored holes and has been included in the financial model. Main and auxiliary fans will be relatively large power consumers, for the ultimate condition, total fan absorbed power will be approximately 6.0 MW.

Due to the shallow depth of the mine, not exceeding 650m below surface, it will not be necessary to consider refrigeration of the intake air.

Item 21(e): Resource to Reserve Conversion - Basic Grade Equation

The historical Resource statement for the orebody is contained below. Only Indicated and Measured Resources are included in the overall mine plan.

Mineral Resource Statement

Table - Mineral Resource Statement - Merensky

Cut-Off 3PGE_Au	Tonnage	Geo Loss	Tonnage	4E	Content 4E		Reef Width
cmg/t	t	%		g/t	g	Moz	cm
MR Measured							
0	8,079,815	14	6,948,641	6.71	46,636,165	1.499	108
MR Indicated							
0	20,200,910	14	17,372,783	5.29	91,876,308	2.954	109
MR TOTAL							
0	28,280,725	14	24,321,424	5.70	138,512,473	4.453	109

Table - Mineral Resource Statement - UG2

Cut-Off 3PGE_Au	Tonnage	Geo Loss	Tonnage	4E	Content 4E		Reef Width
cmg/t	t	%		g/t	g	Moz	cm
UG2 Measured							
0	11,214,700	23	8,635,319	3.31	28,598,095	0.919	141
UG2 Indicated							
0	30,986,290	23	23,859,443	3.43	81,767,529	2.629	134
UG2 Total							
0	42,200,990	23	32,494,762	3.40	110,365,624	3.548	136

A planning pay limit was established for both reefs using estimates of working costs and revenue. This resulted in a stope face pay limit of 3.5 g/t for both Merensky and UG2.

Table - Planning Pay Limit Basis

Planning Pay Limit

Merensky	
Concentrator	88%
Smelter Recovery	86%
UG2 levy	100%
Revenue per gram	220.000
Cost per tonne milled [R]	460
MCF	98%

Tons Mill

Source	Tonnes	Val	Content	%
Stope Face	757	3.50	2650	75.70%
Sundries	205	0.40	82	20.50%
To Mill	962			96.20%
Dev to Mill	38	2.71	103	3.80%
MCF 98%	1000	2.84	2835	100%
To Plant	1000	2.78	2779	100%
Product	1000	2.09	2091	100%

Total Cost	460000

UG2	
Concentrator	83%
Payability	86%
UG2 levy	99%
Revenue per gram	240.000
Cost per tonne milled [R]	460
MCF	98%

Tons Mill

Source	Tonnes	Val	Content	%
Stope Face	759	3.51	2662	75.90%
Sundries	206.5	0.25	52	20.65%
To Mill	965.5			96.55%
Dev to Mill	34.5	2.04	70	3.45%
MCF 98%	1000	2.78	2784	100%
To Plant	1000	2.73	2729	100%
Product	1000	1.92	1917	100%

Total Cost	460000

Using these cut off grades, mining blocks for inclusion in the mine plan were identified. This resulted in the exclusion of 5.09 million tonnes of Merensky reef and 17.3 million tonnes of UG2. Based on the knowledge of each mining block, a geological loss factor was applied to each of these blocks. This resulted in the exclusion 2.6 million tonnes of Merensky reef and 2.3 million tonnes of UG2.

Systematic pillar support resulted in the exclusion of 2.6 million of Merensky reef and 2.5 million tonnes of UG2.

Given the selected mining method for both reefs, hand held breast stoping, dilution factors were established as shown below.

Table 88 - Production Modifying Factors for conversion to Reserve

Modifying Factors

Merensky Reef

	Value	Sg
Hangingwall1 (20cm)	0.22	3.16
Best Cut (Min 80cm)		
Footwall 1 (20cm)	1.32	3.0
Footwall2 (20cm)	1.18	3.0
Footwall3 (20cm)	0.97	3.0

Overbreak @ 4% of Best Cut		CMGT
	HW	4.4
	FW	26.4
Average Value		0.77 gt

UG2 Reef

	Value	Sg
Hangingwall1 (20cm)	0.11	3.16
Best Cut (Min 80cm)		
Footwall 1 (20cm)	0.38	3.0
Footwall2 (20cm)	0.38	3.0
Footwall3 (20cm)	0.38	3.0

Overbreak @ 4% of Best Cut		CMGT
	HW	2.2
	FW	7.6
Average Value		0.245 gt

Stoping Sundry (Modifying Factors) Average Value

	%	gt	Calc	cm
Overbreak	4	0.77	3.08	5
ASG	7.45	0.54	4.023	135
Winch Bed	0.6	0.69	0.414	100
Faulting	5	0	0	
Re-Raising	2	0.69	1.38	100
Total	19.05	0.467	8.897	

%	gt	Calc	cm
4	0.24	0.96	6
5.26	0.38	1.99804	97
0.6	0.38	0.228	100
5	0	0	
2	0.38	0.76	100
16.9	0.23	3.94604	

The average diluting effect on stope tonnes for Merensky reef is 19.05% and for UG2 16.9%.

The whole process above is summarised in the Basic Grade Equation depicted in the following tables.

Table 89 - Basic Grade Equation - Merensky & UG2

	Merensky 4E	Factor	Tonnes (000)	g/t	4E Content (Kg)	Copper g/t	Nickel g/t
Geological Information Measured & Indicated	Average Reef Dip (Degrees)	19					
	Mining width (cm) - Mining Blocks	116					
	Average Relative Density	3.24					
	Geological Losses:						
	Potholes	4%					
	Faults	10%					
	Replacement pegmatite	25%					
	Resource (after geological losses)	39%	24321	5.70	138,512	0.09%	0.20%
Mining Information	Pay limit	3.5 g/t	5,094	0.36	1,840		
	Mining Blocks		19,228	7.11	136,673	900	2000
	Mining loss in Mining Blocks (-)	12.2%	1,967	8.01	15,762		
	Extraction Ratio:						
	Gully Pillars (-)	15.7%	2532	8.01	20,291		
	Regional Pillars	Superimposed on geological losses					
			14730		100,620		
	Balance of Mining Blocks to Mine		14730	6.83	100,620	900	2000
	Planned dilution:						
	ASG (% Stoping)	7.5%	1,268	0.54	6845		
	Winch beds (% Stoping)	0.6%	102	0.69	708		
	Reef development FW (% Milled)	4.2%	40	0.54	215		
	Additional dilution:						
	Off reef mining loss incl Flts (% Stoping)	5%	851	0	0.0		
	Off reef development (re-dev) (% Stoping)	2%	339	0.69	234.2		
	Overbreak (% Stoping)	4%	681	0.77	524.3		
	RIF / PIH Content loss (- From Stope Faces)	1%			-977		
	Calculated Tonnes Milled		18,011	5.62	101,178		
	Scheduling Efficiency		0.00%	0.00%	0.00%		
	Surpac Tonnes Milled (Reported)		18,011	5.62	101,178	740	1645
	Mineral Reserves (Proved & Probable) MCF	98%	18,011	5.51	99,155		
						% To Mill	0.074%	0.164%

	UG2 4E	Factor	Tonnes (000)	g/t	4E Content (Kg)	Copper g/t	Nickel g/t
Geological Information Measured & Indicated	Average Reef Dip (Degrees)	16					
	Mining width (cm) - Mining Blocks	155					
	Average Relative Density	3.66					
	Geological Losses:						
	Potholes	8%					
	Faults	15%					
	Replacement pegmatite	18%					
	Resource (after geological losses)	41%	32,495	3.40	110,366	0.009%	0.078%
Mining Information	Pay limit	3.5 g/t	17,256	2.72	46,870		
	Mining Blocks		15,238	4.17	63,495	90	780
	Mining loss in Mining Blocks (-)	11.6%	1,576	4.37	6,883		
	Extraction Ratio:						
	Gully Pillars (-)	16.4%	2,232	4.37	9,748		
	Regional Pillars	Superimposed on geological losses					
			11,430		46,865		
	Balance of Mining Blocks to Mine		11,430	4.10	46,865	90	780
	Planned dilution:						
	ASG (% Stoping)	5.3%	566	0.38	215.0		
	Winch beds (% Stoping)	0.6%	79	0.38	30.1		
	Reef development FW (% Milled)	2.1%	16	0.23	3.7		
	Additional dilution:						
	Off reef mining loss incl Flts (% Stoping)	5%	657	0	0.0		
	Off reef development (re-dev) (% Stoping)	2%	262	0.38	99.6		
	Overbreak (% Stoping)	4%	526	0.24	126.1		
	RIF / PIH Content Loss (- From Stope Faces)	1%			-459		
	Calculated Tonnes Milled		13,535	3.46	46,880		
	Scheduling Efficiency		0.00%	0.03%	0.03%		
	Surpac Tonnes Milled (Reported)		13,535	3.46	46,895	76	659
	Mineral Reserves (Proved & Probable) MCF	98%	13,535	3.40	45,957		
						% To Mill	0.008%	0.066%

Mineral Reserves are reported as inclusive of uneconomic material and dilution material delivered for treatment or dispatched from the mine without treatment, using the modifying factors as indicated in the basic grade equation.

Table 90 - Mineral Reserve Statement

Mineral Reserves 4E

Merensky				UG2			
Tonnage	4E	Content 4E		Tonnage	4E	Content 4E	
t	g/t	Tonne	Moz	t	g/t	Tonne	Moz
Merensky Proved				UG2 Proved			
6,677,304	5.61	37.478	1.205	5,086,535	3.37	17.126	0.551
Merensky Probable				UG2 Probable			
11,333,294	5.44	61.677	1.983	8,448,801	3.41	28.831	0.927
Total Merensky Mineral Reserves				Total UG2 Mineral Reserves			
18,010,598	5.51	99.155	3.188	13,535,336	3.40	45.957	1.478

Prill Splits	Pt	Pt (g/t)	Pd	Pd (g/t)	Rh	Rh (g/t)	Au	Au (g/t)
Project 1 & 1A MR	64%	3.52	27%	1.49	4%	0.22	5%	0.28
Project 1 & 1A UG2	63%	2.15	26%	0.88	10%	0.34	1%	0.03

Table 91 - Listing of Individual Elements from 4E Reserve Statement

Metal	Reef	Category	Prill Split	Tonnage Tonnes	g/t 4E	4E Content Tonne	4E Content Moz
Pt	Merensky	From Proved 4E Reserve	64%	6,677,304	3.59	23.99	0.771
		From Probable 4E Reserve	64%	11,333,294	3.48	39.47	1.269
		Total	**64%**	**18,010,598**	**3.52**	**63.46**	**2.040**
	UG2	From Proved 4E Reserve	63%	5,086,535	2.12	10.79	0.347
		From Probable 4E Reserve	63%	8,448,801	2.15	18.16	0.584
		Total	**63%**	**13,535,336**	**2.14**	**28.95**	**0.931**
Pd	Merensky	From Proved 4E Reserve	27%	6,677,304	1.52	10.12	0.325
		From Probable 4E Reserve	27%	11,333,294	1.47	16.65	0.535
		Total	**27%**	**18,010,598**	**1.49**	**26.77**	**0.861**
	UG2	From Proved 4E Reserve	26%	5,086,535	0.88	4.45	0.143
		From Probable 4E Reserve	26%	8,448,801	0.89	7.50	0.241
		Total	**26%**	**13,535,336**	**0.88**	**11.95**	**0.384**
Rh	Merensky	From Proved 4E Reserve	4%	6,677,304	0.22	1.50	0.048
		From Probable 4E Reserve	4%	11,333,294	0.22	2.47	0.079
		Total	**4%**	**18,010,598**	**0.22**	**3.97**	**0.128**
	UG2	From Proved 4E Reserve	10%	5,086,535	0.34	1.71	0.055
		From Probable 4E Reserve	10%	8,448,801	0.34	2.88	0.093
		Total	**10%**	**13,535,336**	**0.34**	**4.60**	**0.148**
Au	Merensky	From Proved 4E Reserve	5%	6,677,304	0.28	1.87	0.060
		From Probable 4E Reserve	5%	11,333,294	0.27	3.08	0.099
		Total	**5%**	**18,010,598**	**0.28**	**4.96**	**0.159**
	UG2	From Proved 4E Reserve	1%	5,086,535	0.03	0.17	0.006
		From Probable 4E Reserve	1%	8,448,801	0.03	0.29	0.009
		Total	**1%**	**13,535,336**	**0.03**	**0.46**	**0.015**

Item 21(f): Basic Mining Equation

For every four stope panels ventilated and equipped, one panel per day is blasted.

Stope panels are 25 metres long.

Panels are included in the mine plan when they are ventilated (raises holed) and resourced (all equipment for the panel and raise have been paid for in the capital estimate) and sufficient timing has been allowed in the production schedule for ledging and equipping.

Panels are advanced by an effective 5.5 metres / month in the mine plan.

This is derived as follows:

- For every panel planned to produce in any month, an additional panel, fully equipped is considered unavailable for geological or other reasons
- A stoping crew, consisting of 20 men, is allocated to two producing panels
- Depending on the length of back, a raise line will generally have 2 or 3 crews operating
- The crew blasts one panel per day, each panel 12 times per month. At a face advance per blast of 0.9 metres, this equates to 11 metres per working panel per month. For scheduling purposes, allowing for the fact that only half of the equipped panels are blasted each month, the overall face advance planned is 5.5 metre / month. This means that the mine must blast one out of four equipped panels per day. For a standard length raise, not more than three panels per day are blasted
- An average raise line with 10 panels will produce approximately 5,300 stope tonnes per month (25 metre panel x 5.5m advance/month x 10 panels x 3.9 tonnes per square metre)
- Each crew will have the capacity to produce 2,150 tonnes, or 550 m^2 per month at 26 m^2 per stope man per month
- 70 stope crews are planned at full production
- 70 stope crews have the potential to produce 38,500 m^2 per month

At peak production 35,000 m^2 are planned to be mined per month. Of these, 2,800 m^2 come from ledging, which is resourced separately. This means that the 70 stope crews must produce 32,200 m^2 per month giving an expected output of 460 m^2 per crew per month.

An excess capacity of 20% exists between the theoretical output of the stoping crews and the planned output. This capacity is utilized in re-raising and re-establishment of non-producing panels. Re-raising tonnage constitutes 2% of total stope tonnes.

A production unit will have two raises stoping, one raise vamping / reclaiming and one raise ledging / equipping and can theoretically produce up to 14,000 reef tonnes per month, including ledging and reef development. Fourteen production units are planned and costed in the financial model at an average of 11,200 reef tonnes per month.

For an average stope back of approximately 190 metres, with footwall haulage 20 metres below reef, it takes

- 12 months to develop the crosscut and hole the raise to the level above
- Month 13 is utilized to strip the raise and complete box holing
- Months 14 and 16 are used to ledge the raise to a position 6 metre from the original raise sidewall on both sides of the raise
- Months 17, 18 and 19 are used to equip the raise, two panels per month on each side of the raise, from the bottom up

This allows limited stoping: 550 m^2 /month in month 18: 1,100 m^2 per month in month 19 and 1 650 m^2 per month in month 20.

Each production unit will have one reef raising / ledging / stope equipping crew capable of ledging and equipping the necessary panels per month from the 14 th month after the crosscut breakaway position is reached.

When the need for equipped panels diminishes, this crew will do reclamation and vamping. The duty for this crew is calculated as follows:-

- The mine needs to develop 230 reef raise metres per month to sustain production
- One raise metre on reef gives 150 square metres of mining based on the strike spacing of 150 metres between raises. If stoping is reducing the available Reserve by 35,000 m^2 per month, it is necessary to develop 230 reef metres per month or 17 reef metres per production unit. At 4.2 metres / man / month, this is 4 people
- Ledging is carried out 6 metres either side of the raise
- 35,000 m^2 requires 35,000 x 12 / 150 = 2,800 m^2 ledging or 200 m^2 per section. At 20 m^2 per man, this requires 10 men
- 35,000 m^2 at 5.5 metres per month overall face advance over life of raise, requires 220 equipped panels to sustain production
- Each panel contains 1,725 m^2
- Each month, twenty (35,000 / 1,725) panels are depleted and must be replaced with equipped panels. This is just over one panel per section per month. 5 people are allocated to each section for this equipping

The total crew requirement per section for reef raising / ledging and equipping is 20 with a mine total of 280.
Re-raising and re-establishment of off reef panels is catered for in the 19% excess allowance in the stoping crews.

Allowing for the 15% relief category, the total on-reef labour that will be required and has been costed is 1,775. The labour productivity determined as 'm ^{2} per total reef employee costed' is 19.

The high relief percentage is to allow for a 25 day per month work cycle. Continuous operations have not been included in the production plan.

To comply with the requirements of a 45 hour work week, each employee can only work 7.5 hours per day. To comply with the requirements of a 90 hour, 11 day fortnight the employee can work 8.2 hours per day. This requires an additional 7% to allow the necessary rotation. The remainder of the relief category is for annual leave, illness and other valid reasons for failure to be at work. This is regarded by the project team as a reasonable approach.

A summary of the labour requirements for on-reef production is contained in the following section.

Item 21(g): Mining Labour

Raising, Ledging & equipping labour		
Miners	14	One per crew improved safety and productivity
Team leaders	28	One per crew per shift
Winch erecting	28	Winch transport on reef, two per crew
Rock drill Operators	84	Six per crew for raising ledging and equipping.
Winch drivers	56	Two per shift per crew
General	70	Day shift workers as required
Leave / cycle relief	40	
Total	**320**	
Stoping labour		
Miners	84	One miner per crew on day shift, One per section on night shift
Team leaders	140	One per crew per shift
Rock drill operators	280	four per crew
Drill assistants	70	one for gully
Generals	280	four per crew dayshift as required
Winch drivers	280	three per crew night shift, one day shift
Mono winch drivers	70	one per crew day shift
Tip attendant	70	one per crew
Leave/cycle relief	181	
Total	**1,455**	

The total staff required for on reef mining will be 1,775 persons at peak production.

Operating Cycles

Each stoping crew under the day shift control of a qualified miner, is allocated two adjacent (or close to each other) stope panels per month. Each day, one panel is supported and prepared for drilling and the other is drilled and blasted.

Cleaning is done mainly on night shift but winch drivers are also allocated to day shift. This allows for the clean up of any remaining reef, but also allows for cleaning on day shift to make up for lost blasts. The day shift winch drivers also prepare the cleaning rigs for night shift.

In the preparation cycle, permanent support (elongates) are installed as per mine standard. Temporary support in the form of mechanical props has been dispensed with in the stope design. Each crew is allocated a stope support drill rig operated remotely. Roof bolts are installed on a predetermined systematic pattern during the preparation cycle. When support is complete, the face can be marked off by the miner with time available to ensure that the marking off is done correctly and according to mine standard.

On the second panel, the drilling crew drills the face that has been prepared on the previous shift. No time is wasted waiting for the completion of the cleaning, marking of the holes and supporting of the face, nor is there any need to start drilling before a face has been adequately prepared. The effective drilling shift is greatly increased. Combined with the greater attention given to marking the holes, better face advance per blast should be achieved in a safer environment. Better stoping width control is also possible as well as adhering to the best stoping cut as recommended by the geologists.

In the event of a missed blast by the crew, the capacity exists in the cleaning and drilling labour allocations to make up the lost production.

70 stope crews are planned at full production. To achieve the target, each crew must achieve 460 m^2 per month at 21 m^2 per man per month.

Item 21(h): Production Planning

A production planning summary is included as Appendix 1 - the following section references the page numbers in Appendix 1 and is referred to as ‘ ***A1pg*** ’.

The orebody is accessed from surface by a system of multiple mechanized declines.
Two of the declines, the North and Central declines, share a common portal in the centre of the mines surface infrastructure area. The third decline, the South decline has a portal some 700 metres from the Central portal to the south.

The advance rates and dimensions used for each of these declines at various stages, as well as the advance rates and dimensions on the horizontal levels serviced by the declines are indicated on A1pgs 1 to 6.

The rates used are based on the design and assessment of Wardrop Engineering, which have extensive experience in the field of mechanized underground development. The primary target of the declines is the Merensky reef. Payable Merensky reef occurs from depths some 150 metres below surface.

The calculations and assumptions used to move from Resource to Reserve tonnages are shown on A1pgs 7 to 12. A1pg 12 is a tabulation of the Basic Grade equation for the two reefs.

A1pg 13 and 14 show borehole positions and Resource categories for both reefs. The Merensky reef 60 metre reef contours are shown on A1pg 15.

The mining blocks for both reefs with grade reef contours and structure are shown on A1pgs 16 to 19.

A1pg 20 shows the mining areas by decline.

The three decline systems including chairlifts and horizontal levels are shown separately on A1pgs 21, 22 and 23.

A comparison between m^2 stoped and m^2 created by development is shown on A1pg 24. There are two distinct phases as the Merensky builds up and depletes, with the UG2 repeating the pattern. The initial UG2 development may need to be accelerated to maintain at least 2 years of developed Reserve at any time.

The Life of Mine stoping plan for each decline separately on the Merensky reef is shown on A1pg 25 and A1pg 26 for the UG2.

The Life of Mine Merensky plan by time period is on A1pg 27.

A1pg 28 shows the utilization of Merensky infrastructure in mining the UG2.

The Life of Mine stoping plan for the UG2 is shown on A1pg 28 and shows the existing Merensky infrastructure and the additional infrastructure developed for the UG2 extraction.

Using the development rates given on A1pgs 1 to 3 of Appendix 1, and the ledging, equipping and stoping rates defined in the previous part of this chapter, the key mining timelines are tabulated on A1pg 29.

Life of Mine production schedules are generated by reef type, by decline, by level and by development product and plant feed.

These schedules are summarized on A1pgs 30 to 33 and form inputs into the Financial Model.

Key elements from these schedules are graphed and tabulated on A1pgs 34 to 43. A1pgs 44 to 48 are schematics of development and equipping layouts for the current mine plan A1pg 49 is a plan of the proposed surface layout.

Item 21(i): Decline and Footwall Development

Development is dealt with in three broad categories in this report; Decline Development, Footwall Development, and Stope Development. This section deals with the first two categories, while Stope Development is dealt with in Item 21(c) and (f).

Decline Development

The orebody will be accessed by three sets of declines, which will connect surface to underground production levels.

An economic comparison of decline access versus vertical shaft access showed that decline access will provide the best return on investment and will reach ore in the shortest time. The use of three declines allows three shallow, high-grade areas of the orebody to be opened up simultaneously with consequent positive impact on project economics.

Accessing shallow or outcropping platinum and gold orebodies by decline has for a long time been a conventional method in South Africa. Recent improvements in Haul Truck engine power have increased the speed at which haul trucks climb declines, making decline access technically feasible and economically attractive in this application despite the ore sub-cropping at 150 m below surface.

Decline sets consist of a pair of declines - a Haulage Decline for rubber-tired vehicles (trackless equipment) for moving ore and waste, supplies and equipment, and supervisory and technical personnel, and a Chairlift decline for moving men. Haulage Declines will be mined at no steeper than 8.5 degrees grade, the maximum allowable for effective haul truck operation. Chairlift Declines will be mined to follow the dip of the orebody, which will generate grades of up to 22 degrees.

Haulage Decline cross-sections will be sized to accommodate 30 tonne diesel haul trucks, which form part of the chosen equipment, set for this study. Opening cross-sections and turning radii will be sized to permit the future use of 50 tonne haul trucks should it be decided to use these instead of 30 tonne haul trucks.

Decline development will commence from faces of competent rock, which will be created by excavating box-cuts through up to 30 m of weathered material.

Haulage Declines will by mined by mechanized methods, comprising trackless drill rigs, roof-bolters, LHD's and 30 tonne haul trucks. Chairlift Declines will be mined by the same mechanized methods wherever possible, but where mined at greater than 12 degrees, mechanized mining is not economic and handheld methods using jackleg rock drills and scraper winches will be employed.

Broken rock will be loaded by LHD from the face to 15 m long remuck bays installed at 75 m intervals. Waste rock will be hauled by 30 tonne trucks from remuck bays either directly to the surface waste dump or if coming from 45 Level or lower, to the underground waste rock bin on 45 Level. From the underground waste rock bin, it will be conveyed to the surface waste rock bin from where it will be hauled to the waste rock dump by surface haul trucks.

Once not required for development, remuck bays will be used as traffic passing bays, as temporary sumps and as spaces for electrical substations, material storage, vehicle parking, and the construction of refuge stations.

Where developed in parallel, Haulage Declines and Chairlift Declines will have a minimum 10 m intervening solid pillar to facilitate their use as independent means of emergency egress. Haulage and Chairlift Declines will be advanced at the same rate of advance, and will be connected by cross-connectors at 75 m intervals.

After reaching the orebody, Haulage Declines and Chairlift Declines will follow the dip of the Merensky Reef at an average depth of 30 m in the footwall. Where the ore dip is steeper than 8.5 degrees, Haulage Declines will follow the apparent dip of the orebody in a series of switchbacks to maintain an effective roadway grade of 8.5 degrees. Chairlift Declines are less limited in the grade they may be installed at, and will follow the dip of the ore at grades ranging from almost horizontal to greater than 20 degrees.

Chairlift declines will be connected to footwall travelling ways by short inclined walkways installed at no greater than 34 degrees and which will have a concrete stairway installed to facilitate foot traffic.

The back and walls of headings will be scaled and ground support will be installed as headings advance. Rock bolting to the face will be practiced. Resin-grouted rebars at 2.4 m long in a 1.5 x 1.5 m pattern will be used as the standard for access development, based on geotechnical analysis reported on in Item 21(b) 4 for the purposes of development productivities and cost estimation.
Permanent pipelines, ventilation ducts, and power cables will be installed as declines are advanced.

Footwall Development

Footwall development will consist of Strike Haulages developed at approximately 20 metres below reef, cross cuts, inclined travelling ways and a single ore pass per raise line.

Footwall developments will be mined by the same trackless equipment and methodology used in developing Access Declines.

Footwall Development openings will be sized to accommodate the 30 tonne diesel haul trucks, which form part of the chosen equipment, set for this study. Opening cross-sections and turning radii will be sized to permit the future use of 50 tonne haul trucks should it be decided to use these instead of 30 tonne haul trucks.

Pipelines, ventilation ducts, and power cables will be installed as headings are advanced.

Underground Development Rates

Decline and footwall development rates significantly impact project economics and project schedule, especially in the early years of mine development prior to commencing ore production, and after that as stopes are being developed as quickly as possible. Choosing achievable and realistic advance rates for underground development has been the subject of much research in this study.

Recognizing the importance of this, three South African underground contractors and one Canadian underground contractor were canvassed for opinions on advance rates for this project. Their views have been combined with Wardrop engineers' knowledge base of advance rates derived from similar projects and have led to the adoption of the following advance rates.

Table 92 - Haulage Decline Development Initially - Haulage Decline Development Only .

Activity	Conditions	Equipment	Advance Rate
Haulage Decline Development	Multiblast conditions - no production	One dedicated suite of mechanized equipment	100 m/mo

Table 93 - Haulage Decline Development Intermediate - Haulage Decline and level development to assist rapid build up, concurrent with limited production.

Activity	Conditions	Equipment	Advance Rate
Haulage Decline Development	Blasting at two fixed times per day. Decline has priority over other development	One dedicated suite of mechanized equipment	85 m/mo
Haulage Level and Cross -cut Development	Blasting at two fixed times per day	One dedicated suite of mechanized equipment	100 m/mo if one face available. 160 m/mo if two or more face available

Table 94 - Haulage Decline Development Steady State -Decline and level development concurrent with steady state production.

Activity	Conditions	Equipment	Advance Rate
Haulage Decline Development	Blasting at two fixed times per day. Decline has priority over other development	One dedicated suite of mechanized equipment	85 m/mo
Haulage Level and Cross -cut Development	Blasting at two fixed times per day	One dedicated suite of mechanized equipment	30 m/mo per face to a maximum of 120 m/mo

Table 95 - Chairlift Decline Development

Activity	Conditions	Equipment	Advance Rate
Chairlift decline development parallel to Haulage Decline	Multi-blast conditions	One dedicated suite of mechanized equipment	100 m/mo
Chairlift decline Development independent	Blasting at two fixed times per day	Conventional hand held mining	30 m/mo

Table 96 - Inclined Development

Activity	Conditions	Equipment	Advance Rate
Reef Dip Development	Blasting at two fixed time per day. Drop-raised from centre gulley before stoping commences	Conventional hand-held mining	30 m/mo
Orepass Development	Blasting at two fixed times per day	Conventional hand held mining	30 m/mo
Orepass Development	Blasting at two fixed time per day. Drop-raised from centre gulley before stoping commences	Raise Bore	30 m/mo

Table 97 - Reef Horizontal Development

Activity	Conditions	Equipment	Advance Rate
Reef Strike Development	Blasting at two fixed times per day	Conventional hand-held mining	30 m/mo

Underground Development Schedule

Early mine development has been scheduled to achieve full production of 160,000 tpm of ore in the shortest time possible. Development will commence simultaneously on all three declines, and will proceed at the fastest rate possible. Contractors will be used to excavate the decline box-cuts and to mine the declines, as it is expected that they will be able to both mobilize and to reach maximum productivity faster than owner's forces.

The underground development schedule forms part of the overall Project Schedule described in Item 21(n).

Initial mine development will focus on:

- providing access for men, materials, and equipment to high-grade, shallow areas of the orebody
- providing an early means of ventilating stopes
- providing two routes for emergency egress at all times

- installing mine services (electric power distribution, communications, compressed air, water supply, mine dewatering)

The early development schedule does not include for time to obtain environmental and mining permits, surface access roads, power lines, and site infrastructure construction.

Underground Mobile Equipment List

Sizing and selection of mining equipment is state-of-the-art technology and is chosen based on operating experience in mines with similar conditions. Mobile underground equipment types will be:

Table 98 - Underground Mobile Equipment list

MOBILE UNIT	MAKE & TYPE
Load-Haul-Dump	
4.6m3 (10 tonne capacity)	Toro 007
Haul Trucks	
30 tonne	EJC 533
Drilling Fleet	
2 Boom Jumbo	Tamrock Axera 7
Bolter	Robolt 5-126
Jackleg	Generic
Stoper	Generic
Compressor	Generic
Service Vehicles	
Flatbed Truck	Fermel
Explosives Truck	Fermel
Scissor Lift	Cassette
Fuel & Lube	Cassette
Light Vehicle	Toyota
Surface Truck	Volvo 20t

Underground haul trucks and LHD’s will be equipped with engines using DDEC technology to optimize efficiency of the engines and to minimize emissions.

The new design allows direct loading of haul trucks from stope orepasses.

Underground Development and Transport Personnel

Underground Development and Transport hourly labour start at 100 and vary between 250 and 340 at full production.

Item 21(j): Underground Services

Underground services will include:

- Mine Dewatering
- Drilling and fire fighting water
- Compressed air
- Electric power
- Communications

Mine services will be installed progressively as headings are advanced.

Compressed Air

Compressed air will be used underground for:

- secondary mine water pumping using pneumatic pumps
- explosive emulsion loading
- blasthole cleaning
- miscellaneous pneumatic small tools

Water Supply

Industrial-quality water will be distributed by steel pipelines throughout the underground workings for drilling equipment, dust suppression, and fire fighting. Flexible hoses will be used to connect water pipelines to drilling equipment at working faces.

A water tank located on surface will provide an uninterrupted supply of fresh water and fire-fighting water.

Mine Dewatering

Water inflows to the underground mine derive from groundwater inflows, drilling water, and dust suppression water.

Mine water on Strike Haulages and Cross-cuts will be directed by gravity to pump niches located at 600 m intervals, from where it will be pumped horizontally to a dirty water pumping station. Dirty water pumping stations will be located on every second Strike Haulage, providing a 120 m vertical interval from one dirty water pump station to the next. Mine water will be pumped progressively from one pump station to the next until arrival at surface.

Ultimately, a clean water pumping station will be installed on the 45 Level, allowing more efficient and cost-effective clean-water pumping to be performed from 45 Level to surface.

Underground Communication System

A leaky feeder communication system will be installed as the communication system for the mine and surface operations. Telephones will be located at key infrastructure locations such as chairlift stations, electrical sub-stations, conveyor drives, refuge areas, lunchrooms, and pumping stations.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) will be supplied with underground radios for contact with the leaky feeder network.

'Stench' Gas Warning System

A mine-wide 'stench' gas warning system will be used to alert underground workers of a fire.

Item 21(k): Underground Construction and Mine Maintenance

Supervised crews will perform general mine maintenance and construction work, ground support control and scaling, road checking and maintenance, construction of ventilation doors, bulkheads, and concrete work, mine dewatering, services movement, and safety work. Underground graders and scissor lifts will be used to maintain haulages travelled by mobile equipment.

Underground Mobile Equipment Maintenance

Initially, mobile underground equipment will be maintained in a shop located on surface close to decline portals. Ultimately, it will become economical to save vehicle travel time by establishing underground workshops and service bays. Surface and underground workshops are described fully in Item 21(m).

A maintenance supervisor will provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision of maintenance crews. He will also manage training of the maintenance workforce.

A maintenance planner will schedule maintenance and repair work and provide statistics on equipment availability, utilization and life cycle, mine efficiency, and personnel utilization. A computerized system using commercially available software will facilitate planning.

Equipment operators will provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required throughout the shift.

Mechanics using a mechanic's truck will perform minor repairs and emergency repairs on equipment and vehicles where these cannot make their way to a workshop.

Major rebuild work will be conducted off site.

Item 21(l): Underground Logistics

The efficient movement of rock to surface, and delivery of consumable supplies and equipment to where these are needed for development and production underground, together with the recovery of equipment for refurbishment, are key to achieving the planned production rate within the planned cost structure.

To this end, state-of-the-art diesel trackless vehicles comprising 30 tonne haul trucks, cassette carriers, and flat-bed trucks will be used to physically move rock, materials and equipment underground.

To complement this equipment, appropriate levels of logistics management, supervision, and materials control hardware and software are allowed for.

Ore and Waste Rock Movement

Waste Rock

Waste rock from Decline, Footwall, and Stope Development will be hauled to waste dumps on surface by 30 tonne haul trucks via three Haulage Declines.

In Year 4, the Central Decline conveyor system will be commissioned and waste rock produced on or below the 45 Level will be hauled to an underground waste rock bin on the 45 Level, which will feed onto the Central Decline conveyor system. Waste rock originating from above the 45 Level Waste will continue to be hauled directly to surface because of the safety-based decision made not to perform loaded down-hill hauling.

Ore

Ore will be loaded from the ore passes in each raise line directly into 30 tonne haul trucks .

Early in the life of mine, ore from stopes and from reef development will be hauled in 30 tonne haul trucks directly to the process plant ore bin on surface.

Once the Central Decline conveyor system is commissioned in Year 4, ore produced on or below the 45 Level will continue to be hauled by truck to ore bins on the 45 Level, which will feed onto the Central Decline conveyor, which will convey it the surface ore bin. Ore produced above the 45 Level will continue to be hauled directly to surface by haul trucks as a result of the safety-based decision made not to perform loaded down-hill hauling.

Switch-over from Truck Hauling to Conveying Ore and Waste

While the Central Decline conveyor system is being installed and commissioned, it will not be possible for haul trucks to use the Central Decline as a travel route to surface. During this period, all underground vehicles will have to operate in the South and North Declines. This switchover period is conservatively estimated at 6 months.

During the switch-over to conveying, the mine will be operating at a production rate of 160,000 tpm ore and 60,000 tpm waste rock, coincidentally the highest rock production planned during the life of mine. Traffic simulations have been performed which show that this traffic density can be accommodated by the South and North declines without restricting the flow of broken rock to surface or requiring additional haul trucks. Consequently, the mine production schedule and mine equipment list have not been modified during this switchover period.

Transportation of Underground Workers

The majority of underground workers will be moved between surface and their workplaces by chairlifts installed in declines parallel to the Haulage Declines. Workers will walk from where they get off the chairlifts to the stopes in the strike haulages. No truck haulage will take place during major shift movements.

Chairlifts convey at up to 900 persons per hour, and will allow a complete shift-change to be accomplished in less than 2 hours at the greatest distance of stopes from surface.

Maximum walking distance will be generally 1,000 m, although this may be greater in a small number of locations in the mine.

Some parts of the deposit are small and isolated and at greater walking distances than 1,000 m. It will not be economic to install Chairlifts to access these, and trackless personnel vehicles will be used in these cases, typically of the cassette personnel carrier type.

Transportation of Supervisory and Technical Personnel Underground

Supervisors and technical personnel will have convenient and rapid access to underground operations by light diesel vehicles throughout the life of mine.

Transportation of Materials and Supplies Underground

Consumable supplies such as ground support materials will be delivered underground by diesel supply vehicles. Supply vehicle type will predominantly be 10 tonne cassette carrier diesel trucks having the capability of offloading cassettes containing supplies and equipment at the destination point within the mine, and self-loading empty cassettes for removal to surface.

A lesser amount of supply vehicles will be the flat-bed type, with or without a crane for self-loading.

To the greatest extent possible, supply materials will be bundled, palletized on standard wooden pallets, or supplied in standard bulk bags, for ease of movement on and off flat-bed vehicles and in cassettes. Stoping ground support supplies will be assembled on surface and delivered in unit loads. Unit loads, which will contain the correct mix of the various materials and supplies required by an operating raise-line.

Ground support materials cassettes will be off-loaded in cross-cut storage bays (timber bays) from where materials will be taken from the cassette as required and moved to the raise-lines and stopes by mono-line winches.

Transportation and Storage of Explosives Underground

Explosives for Access Development and Footwall Development will be of the emulsion type with electric initiation.

Emulsion explosives will be delivered by the manufacturer to bulk silos on surface. From there it will be loaded into special explosives vehicles for delivery on a daily basis to underground development headings. Blast holes will be charged pneumatically.

Packaged explosive will be transported as palletized unit loads and offloaded direct from the delivery truck at an explosives receiving area on surface from where it is loaded into cassettes and delivered to underground explosive stores.

Fuel Storage and Distribution

Underground mobile equipment will operate on low-sulphur No. 2 diesel in order to meet emission standards for underground vehicles.

At full production, underground mobile equipment will consume 17,000 litres per day.

Haul trucks and auxiliary vehicles will report to surface at the end of each shift when they will be refuelled and inspected. At a certain point, it will become more economical to refuel vehicles underground. Diesel will be piped from surface to a fuel tank in the underground fuel/lube station on 45 Level. Diesel will be transferred underground in batches in an un-pressurized steel pipeline installed in the Central Decline, which will remain empty at all times other than when diesel is being batched underground.

Lubricants will be delivered in drums for storage at the fuel-lube bay on surface, and in cassettes for transport and storage in the fuel-lube bay underground.

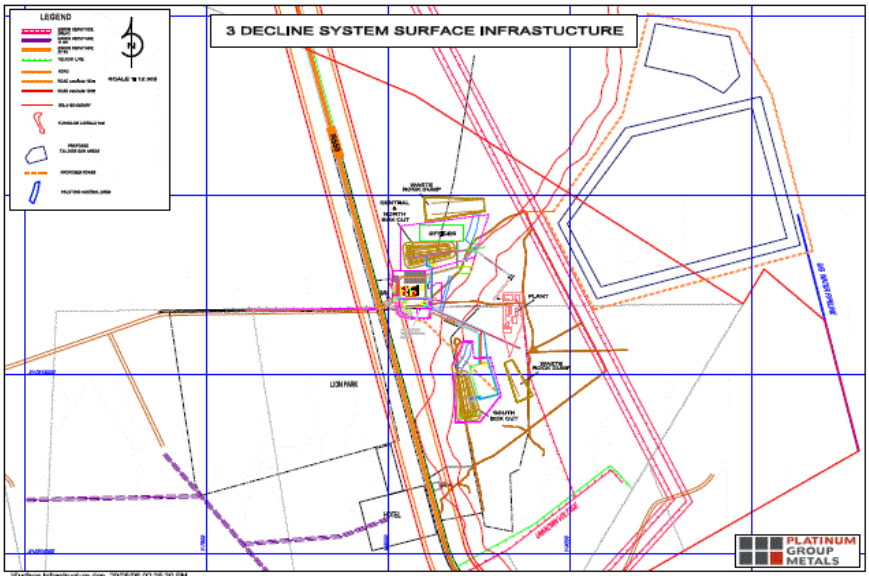
Fuel/lube vehicles will refuel and lubricate daily those vehicles, which will not leave their working places at the end of each shift, such as drill jumbos and rockbolters.

Item 21(m): General Mine Infrastructure

Site Layout

The site layout is as depicted below with the three decline clusters as indicated with the plant and tailings dam as shown.

Figure 36 - Surface Layout



Power Supply

Grid power will be required at the WBJV Project 1 mining site. It is estimated that the decline system envisaged will require about 24MVA maintaining full production. It is expected that 16MVA will be needed for the concentrator and the remainder for the mining and other surface infrastructure.

The only authorized supplier of bulk power is the parastatal organisation, ESKOM. South Africa has recently been subjected to severe shortages of bulk power, but this situation has been somewhat alleviated and the supply commitments from ESKOM have apparently normalised. The WBJV project team in this Updated Feasibility Study has assumed that bulk power will be available from ESKOM but some 25% of installed capacity will need to be generated on site as per documentation received from ESKOM - the team estimates that this will be at the end of December 2012 when additional ESKOM generating capacity is planned to be commissioned. Temporary power is to be made available for the early sinking of the declines and if more power is required, self generation by the contractors if required.

As a result of this situation, self generation of electric power has been included into the overall mine model, although this has been reduced from the Feasibility Study. The following power supplies are costed into the overall model:

- 1.5MVA ESKOM Temporary Power from early 2009
- 10MVA diesel generated power from late 2008 to early 2009
- ESKOM grid supply from end 2012

This situation has reduced the cost of self generated power supply from R501m to approximately R80m.

The status of the ESKOM application process is that the WBJV has received the formal proposal and payment negotiations are underway with the full ESKOM feasibility quote payment to be finalised by early 2010. The WBJV has confirmation that the 1.5MVA will be available, although the timing has not been stated.

The ESKOM tariff is currently approximately R0.32 per kWhr for bulk power supply, whereas the diesel generated power will cost R2.50 and HFO generated will be R1.20 per kWhr. The economics of operating the mine with self generated power are not attractive. The cost of ESKOM grid power is expected to increase by up to 45% per annum for the next 3 years, which means that when the WBJV comes into production, the power price could have increased to R0.97/kWhr - the effect of this increase has been identified in a sensitivity analysis in Item 21(s).

Water Supply

The Bulk Water supply will be sourced from Magalies Water Board, a government institution supplying bulk water to the regional communities. The source of bulk water is the Vaalkop dam, some 60km by road from the WBJV site. Magalies Water has stated in numerous meetings that they have enough water from the dam to supply the Mine requirements, but not the infrastructure to deliver the water.

At present all available delivery capacity for water supply is fully utilized. Thus a new pipeline system will be necessary to supply water to the WBJV site. To deliver water from Magalies Water at Vaalkop Dam, additional power will be required from ESKOM. Whilst Magalies Water is of the opinion that the power will be supplied, the country power risk cannot be ignored, as detailed in the Power Report.

There are a number of Memorandum of Understandings in place dealing with the water supply. The two water schemes that WBJV are required to participate in to have water delivered to the mine are:

- Pilansberg Water Scheme - North (PWSN): cost estimate R44m
- Pilansberg Water Scheme - South (PWSS): cost estimate R74m

At the time of this Technical Report, no funding has been paid to either of the water schemes, although adequate provision has been made in the capital forecast.

The water consumption can be calculated at 6Ml/day based upon a thumb rule of approximately 1kL water per tonne of rock mined. This is confirmed by the attached water balance. The study financial model has included for 0.77kL per tonne milled for the concentrator with 0.33kL/tonne mined.

The indicated current bulk water tariff is R2.42 per kilolitre delivered to the water meters for the mine. This is a consumption tariff only and excludes any capital, rates and surcharge tariff. It is expected that this cost will escalate by up to 17.5% per annum as a result of the power increase estimates - it is expected that in three years time, the cost of water supplied to the mine will be about R2.84 per kL. This has also been included in the sensitivity analysis detailed in Item 21(s).

Surface Infrastructure

The Surface Infrastructure required to operate the WBJV Project 1 platinum mine will consist of the following aspects:

- Power Supply and Reticulation
- Water Supply and Reticulation
- South Decline and associated infrastructure
- Central & North Declines and associated infrastructure
- Surface Ventilation Fans and associated infrastructure
- Waste Rock Dump sites
- Surface tip and waste rock disposal
- Concentrator Plant
- Tailings Dam site
- Surface Roads
- Office complex and other buildings

As part of the decision process and as agreed within the environmental studies that have progressed, the drainage channel on the mine from the south to the north will not be disturbed except for two crossings. These crossings will be for access to the South Decline portal from the main infrastructure and a conveyor route from the Central Decline portal to the Concentrator plant.

Land Ownership

The project infrastructure is based on being developed on surface area, some of which is not currently owned or under the direct control of WBJV. There are four parcels of land that will be necessary for the decline option to proceed as designed. These are the following:

- The Sundown Ranch Hotel land to the east of the R565 for infrastructure
- The Sundown Ranch Hotel land to the west of the R565
- The Van Vuuren ‘airstrip’ to the east of the power lines for tailings dam - purchased
- The land owned by JC Grobler to the north west of the decline portal site to the west of the R565 - purchased

Negotiations are underway with the remaining land owner for the purchase of the properties.
Provision has been made in the capital expenditure forecast for the land purchase as below:

- Hotel - R110 million
- Van Vuuren - purchased
- JC Grobler ground - purchased

Access Roads

A number of access roads will be required around the property including:

- Main Road access from R565
- Plant access road
- Roadway to tailings dam
- Upgrade of R565

Provision has been made in the capital for these access roads plus those directly associated with access to the decline portals and the reef tip. A total provision of R40.7m has been allocated in the capital forecast, a reduction from the Feasibility Study by the removal of the contribution to the regional road upgrade.

Other

Additional soils geotechnical investigation will be required prior to construction and provision of R0.75m has been made in the capital forecast.

Provision of the upgrade of the 'hotel' has been made for the construction camp and other requirements - a total of R3m has been allocated.

Storm water control measures have been catered for with a provision of R11m in the capital estimate.

Logistics

The WBJV Project 1 mine is located in an industrialised area with neighbouring mines and thus logistics are not a concern to the project.

Item 21(n): Project Schedule

The life of the mine will be 14 years at peak production with a two year ramp-up and a 4 year tail, resulting in maximum production life of 20 years. The Merensky Reef profile is at the peak for seven years, followed by a three year period of Merensky mixed with UG2 and then four years of UG2 only.

If the decision to proceed is given in early 2010, the mine will start production reef in 2011, with milling commencing in late 2012. The mine will cease production in 2031 with an exhausted Resource.

Schedule - Decline Option

Considering the decline option, under the ideal conditions (referring to permitting and approvals) the important dates are proposed as follows:

- Completion of a Updated Feasibility Study (UFS) - October 2009
- Decision to proceed with the project start of year (minus) 2 - January 2010
- Completion of project financing - March 2010
- Approval of Prospecting License - August 2010
- Completion of necessary permitting - October 2010
- Order first 2MVA diesel generation plant - December 2010
- Appoint Mining Contractor - January 2010
- Commencement of surface preparation work - March 2010
- Commencement of first decline - June 2010
- Commencement of mining operation from decline - July 2011
- Commencement of toll milling from decline - not applicable
- Appointment of plant construction contractor - June 2010
- First shipment of concentrate - December 2012
- First revenue from concentrate - March 2013
- Achieve 140,000 tonne per month from underground - January 2014
- Achieve 160,000 tonne per month from underground - September 2014
- Commence additional metallurgical drilling - March 2010 at the latest
- Commence pilot plant campaign on mind ore - April 2011

The decline schedule involves four months of preparatory work with regard to design and site establishment, during which time the box-cut access for the two decline systems can be developed. WBJV has an approved permit to sink the decline for bulk sampling.

Value Engineering

The project description and design work that has been associated with the Updated Feasibility Study requires to be reviewed and 'value engineered' where necessary. This process should continue following the submission of the Updated Feasibility Study prior to the project approval and continue after project implementation to ensure optimised mine layout and design is achieved.

Contract Negotiation

The contract negotiations for the mining contractor need to commence as soon as possible following the submission of the Updated Feasibility Study, even before the project has been approved. This will ensure that the site work can commence as soon as possible following the project approval in January 2010.
The negotiations with the process plant contractor need to commence in June 2010 at the latest to ensure that they are available to commence the detailed design work from August 2010, thus allowing the milling plant to be ordered as soon as possible thereafter for commissioning in December 2012.

Tailings dam contractor negotiations will commence during early 2011, thus ensuring that the tailings dam will be available from November 2012, well in time for the first tailings to be discarded during commissioning.

Long Delivery items

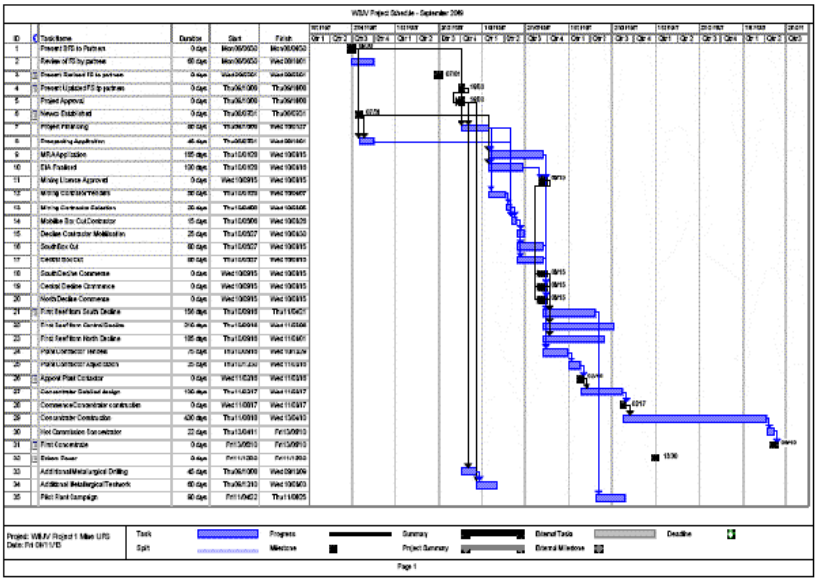
Following the detailed design work, the concentrator will take between 18 and 24 months to construct. The milling plant will have the longest lead-time, with deliveries currently at approximately 65 weeks. The plant needs to be operational by December 2012 or earlier, a there will be stockpiled reef on surface waiting to be processed.

The only remaining long-delivery item is the supply of electric power from the ESKOM grid and the Diesel units for self generation of power. The ESKOM process requires 24 to 30 months from date of acceptance of the budget quotation, which has been formally presented to the WBJV, with commencement in early 2010. Diesel power generation machines have a delivery time of 6 to 8 months and the first should be ordered as soon as the project has been approved.

Gantt Chart

The attached high level Gantt chart reflects the overall project timetable, including the risk elements associated with the Mining Rights Application not having been submitted as at October 2009. It also reflects the risk of not owning the surface rights upon which the mine should be developed.

Figure 37 - Project Schedule



Item 21(o): Metal Marketing & Metal Prices

Concentrates produced by the Western Bushveld Joint Venture (WBJV) may be sold to partner in the JV, Anglo Platinum Limited. As such the WBJV will not be sourcing off takers to purchase end products. The emphasis applied in this marketing report is therefore on future commodity prices.

The major products contributing to revenue are platinum, palladium and rhodium. Minor revenue contributors are gold, copper, nickel, ruthenium and iridium. All of these commodities are covered in this report.

Commodity prices are influenced by the levels of supply and demand which are considered in detail for each commodity. Major economic parameters including the rand dollar exchange rate are also considered.

The WBJV study is complicated by the long time frames involved. Generally short term price forecasts cover the ensuing 12 months with long term forecasts being for 2 to 3 years into the future. The WBJV Updated Feasibility Study requires forecasts some 10 to 15 years into the future. Changes in supply and demand can and will be significant in such a time frame making accurate forecasting impossible. One has only to consider the affect of technological changes and the influence that they will have on demand. On the other hand supply could be affected by successful geological exploration in new areas.

The only logical approach is to use available knowledge on mine developments in projecting supply figures and available knowledge on technological research and economic development in different countries in projecting demand.

The situation is further complicated by the large increases in the demand for a wide range of commodities from 2005 and the recent downturn in the demand cycle. This change has largely been driven by Chinese demand which is set to continue and increase in the long term. Because of this many commentators believe there has been a structural change in commodity markets.

Our price forecasts have been based on the assumption that this structural change has indeed occurred. The forecasts are therefore based less on historical figures and more on views of continued growth in developing countries and particularly in China.

We are aware that this may concern project funders and have therefore made another forecast which we have termed the “fall back” price. This forecast places a greater emphasis on historical figures and is consequently more conservative. It is suggested that if the project is viable at these “fall back” prices then its robustness will be well demonstrated.

In the Updated Feasibility Study, the forecast prices and the “fall back” prices have not been altered since the Feasibility Study was completed in mid-2008. These forecasts are of a long term nature and the happenings of the last 24 months should not be used to make short term changes.

The project is also likely to be listed on the American Stock Exchange and we understand that prospectus reports for this exchange require forecast prices to be based on monthly average prices over a 3 year period. These prices have also been calculated. These prices have been updated.

In summary the metal prices to be used in the Updated Feasibility Study are summarised in the following table.

Table 99 -Summary of Commodity Price Forecasts

as at 24 September 2009						Long Term (BFS)	
		3 year Price	2 year Price	1 year Price	Sep-09	High Road	Low Road
		Case A3	Case A2	Case A1	Case B	Case C	Case D
Pt	$US/oz	$ 1,343	$ 1,402	$ 1,078	$ 1,295	$ 1,450	$ 1,100
Pd	$US/oz	$ 322	$ 311	$ 226	$ 296	$ 600	$ 400
Rh	$US/oz	$ 4,951	$ 4,588	$ 1,469	$ 1,650	$ 2,500	$ 2,000
Au	$US/oz	$ 807	$ 883	$ 896	$ 997	$ 843	$ 500
Ir	$US/oz	$ 439	$ 441	$ 431	$ 425	$ 300	$ 250
Ru	$US/oz	$ 349	$ 257	$ 118	$ 90	$ 300	$ 250
Cu	$US/tonne	$ 6,428	$ 6,113	$ 4,469	$ 6,250	$ 5,000	$ 4,500
Ni	$US/tonne	$ 25,529	$ 19,262	$ 12,966	$ 18,000	$ 20,000	$ 11,000

The project exchange rate has been set at R8.00: US$1.00 for all scenarios.

Item 21(p): Capital Costing

The Life of Mine capital cost of the WBJV project excluding working capital and capitalised operating cost is R4 761 million, with the capitalised operating cost being R745 million. This is detailed in the following table.

There is no working capital provision within the estimate.

The capital cost to get the mine into first concentrate production will be R3 421 million. This is based on first production in December 2012. This is summarised in the table below.

Peak funding requirement for the project will be R3 545 million. The difference between the peak funding and the capital cost to first concentrate production, is the delayed revenue to cater for working capital.

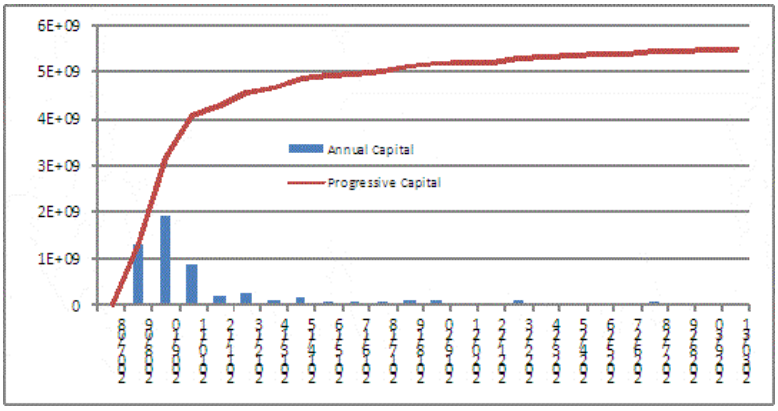
Table 100 - Summary Capital Cost Estimate

WBJV Project 1 - CAPITAL COST ESTIMATE - Life of Mine				
SURFACE INFRASTRUCTURE			R	367,329,367
Electric Power Infrastructure				
ESKOM	R	115,253,117		
Power Generation	R	69,594,000		
Substations	R	29,103,250		
Water Supply	R	127,469,000		
Other Surface	R	25,910,000		
MINE RELATED SURFACE INFRASTRUCTURE			R	216,414,340
Access Roads	R	42,550,455		
Box Cuts	R	67,960,900		
Buildings	R	82,836,730		
Surface Vehicles	R	7,500,000		
Other Mining Related	R	15,566,255		
UNDERGROUND EQUIPMENT			R	1,218,612,278
Mobile Equipment	R	260,052,040		
Chairlift	R	61,597,491		
Ventilation	R	175,700,000		
Other Underground	R	721,262,747		
PROCESSING PLANT			R	540,581,432
Concentrator	R	476,253,432		
Other Process Related	R	1,200,000		
Tailings Dam	R	63,128,000		
PROJECT INDIRECTS			R	388,662,852
EPCM	R	182,254,679		
Construction	R	51,370,213		
Commissioning Spares	R	27,559,361		
Capital Spares	R	38,294,688		
First Fills & Inventory	R	9,367,391		
Freight & Logistics	R	61,753,617		
Commissioning & Start-up	R	18,062,902		
OWNER's COSTS			R	198,881,567
Project Management	R	37,500,000		
Project Engineering	R	18,881,567		
Other Owner Related	R	21,500,000		
Surface Rights	R	110,000,000		
Drilling & Metallurgy	R	11,000,000		
CONTINGENCY			R	372,164,883
SUSTAINING CAPITAL			R	714,167,695
Mining Related	R	580,912,360		
Concentrator Related	R	133,255,335		
CUMULATIVE CAPITAL COST			R	4,016,814,413

COSTS TO FIRST PRODUCTION (CTFP)		
Total Funding to Production	100%	3,421,127,479
Project Directs	51%	1,745,436,834
Project Indirects	11%	374,974,300
Owner's Costs	6%	198,881,567
Contingency	11%	372,164,883
Capitalized Operating Cost	22%	736,773,771
0	0	0
Portion of CTFP based on Allowances and Estimates	8%	281,126,544

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

Figure 38 - Capital Schedule & Profile



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

Table 101 - Early Capital Schedule Requirements

Year	2009/10	2010/11	2011/12	2012/13	2013/14	2014/15	2015/16
Capital	392,307	1,388,640	903,407	272,690	69,798	45,885	232,336
Capital OPEX	54,118	252,815	438,103	0	0	0	0
Total	446,425	1,641,455	1,341,510	272,690	69,798	45,885	232,336
Cumulative	446,425	2,087,879	3,429,389	3,702,079	3,771,877	3,817,762	4,050,098
% of Total	9%	44%	72%	78%	79%	80%	85%

The capital cost changes from the Feasibility Study to the Updated Feasibility Study are summarised below and this has resulted in a total capital reduction of 19.6%. The capital changes were in the following areas:

- **ESKOM** reduction from R138m to R115m, mainly as a result of changing the supply from premium to standard supply - **saving** R13m
- **Power Generation** reduction from R506m to R70m as a result of the improved power availability from ESKOM - **saving** R436m

- **Substation** reduction from R31m to R29m as a result of the reduced power generation requirement - **saving** R2m
- **Water Supply** increase from R125m to R127m as a result of improved scope definition - **increase** R2m
- **Other Surface** reduced from R55m to R26m as a result of removal of the provincial road upgrade from the estimate - **saving** R29m
- **Mine related Surface Infrastructure** reduced from R224m to R216m by updating some of the mining costs, especially for the box-cuts - **saving** R8m
- **Underground Mobile Equipment** reduced from R1,386m to R1,219m by transferring the equipment rebuilds to working costs - **saving** R167m
- **Processing plant and Tailings** increased from R527m to R541m by updating capital costs for plant and tailings - **increase** R14m
- **Project Indirect Costs** have decreased from R491m to R389m due to reduced scopes as defined above - **saving** R102m
- **Owners costs** increased from R198m to R199m with savings from land purchase but increases from owners management costs - **increase** R1m
- **Contingency** reduced from R467m to R372m as a result of reduced scope as defined above - **saving** R95m
- **Sustaining Capital** reduced from R847m to R714m as a result of reduced scope of supply - **saving** R133m

There were some cost increases as a result of the increased throughput, but these costs were negated by the cost savings from other areas in the project.

Item 21(q): Operating Costs

The cost centres are: Mining (including Stoping, Development and Underground Transport), Processing, Stockpile Re-handling, Tailings Dam, Analytical and Assay, Concentrate Transport, Concentrate Treatment Charges (including Sampling Charges), Services (power and water for concentrator), Administration Costs and Overheads with provision for a Rehabilitation Funds.

The mine will be operated on a contractor basis with a Mining Contractor, Processing Contractor and Tailings Dam Contractor. Other services will also be out-sourced to competent local companies as appropriate.

There will be a small management team consisting of up to 10 individuals to manage the contractors with a discipline head covering Management, Engineering, Mining, Process and Administration with accounting staff for Financial Control and Human Resources.

The Life of Mine operating cost is projected to be R539.59 per tonne milled. The costs comprise the following aspects, based on the parameters detailed above.

Table 102 - Operating Cost estimate

TOTAL	R('000)	17,021,849,053
Mining	R('000)	13,502,852,235
Processing	R('000)	1,756,541,980
Stockpile Rehandle	R('000)	-
Tailings Disposal	R('000)	30,504,857
Analytical & Assay	R('000)	157,693,532
Conc. Transport	R('000)	44,796,238
Conc. Treatment	R('000)	406,126,142
Services	R('000)	583,707,035
Admin.& Overheads	R('000)	413,443,297
Rehabilitation Fund	R('000)	126,183,736

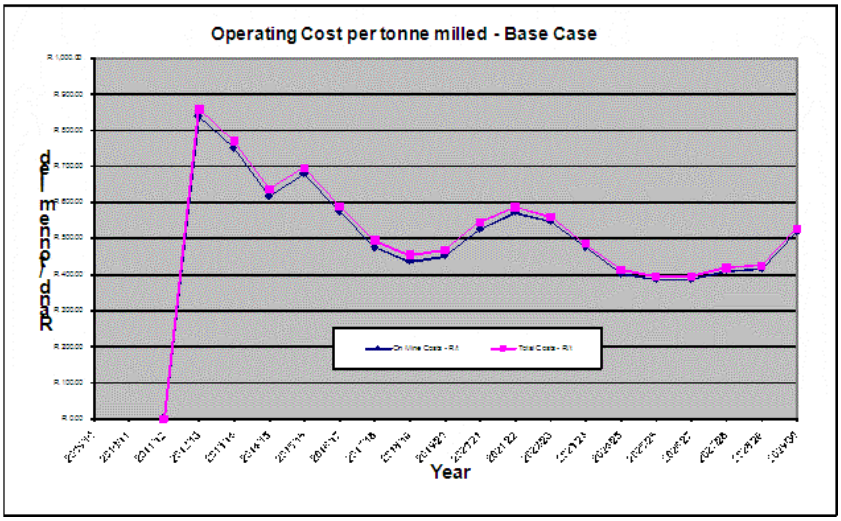
		Cost to Final metal
TOTAL	R/tonne milled	539.59
Mining	R/tonne milled	428.04
Processing	R/tonne milled	55.68
Stockpile Rehandle	R/tonne milled	0.00
Tailings Disposal	R/tonne milled	0.97
Analytical & Assay	R/tonne milled	5.00
Conc. Transport	R/tonne milled	1.42
Conc. Treatment	R/tonne milled	12.87
Services	R/tonne milled	18.50
Admin.& Overheads	R/tonne milled	13.11
Rehabilitation Fund	R/tonne milled	4.00

		Cost to concentrate
TOTAL	R/tonne milled	525.30
Mining	R/tonne milled	428.04
Processing	R/tonne milled	55.68
Stockpile Rehandle	R/tonne milled	0.00
Tailings Disposal	R/tonne milled	0.97
Analytical & Assay	R/tonne milled	5.00
Conc. Transport	R/tonne milled	
Conc. Treatment	R/tonne milled	
Services	R/tonne milled	18.50
Admin.& Overheads	R/tonne milled	13.11
Rehabilitation Fund	R/tonne milled	4.00

The off-mine costs - those associated with the concentrate sales and transport - account for R14.29 per tonne milled. The expected on-mine cost, up to and including concentrate production is 525.30 per tonne milled.

The Life of Mine working cost has the trend as depicted in the following graph.

Figure 39 - Operating Costs - R/tonne

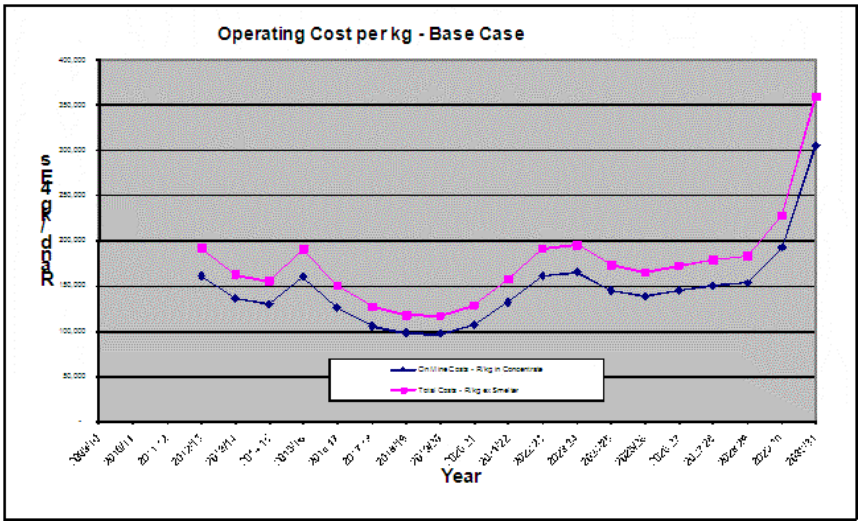


The higher costs in the first years of production are as a result of the reduced scale of production, increased development rates but now do not include the effect of the higher power costs for self generation, as a result of the anticipated improvement in the ability of ESKOM to supply grid power as required for the project timing.

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 R161 015 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 R135 335 per kg 4E in concentrate.

Figure 40 - Operating Costs - R/kg



The estimated operating cost is expected to be accurate to within 10%, but no contingency allowance has been included, apart from for the concentrator.

The Life of Mine working costs are summarised in the following Table in both Rand and $US terms at an exchange rate of R8.00: $US1.00.

Table 103 - Summary of Operating Costs

Operating Costs		
R/tonne		539.59
R/kg in Concentrate		135,335
R/kg Sold		161,648
$US/ounce in Conc.	$	526.17
$US/ounce sold	$	628.48

Early Working Costs

The working costs during the period to December 2013 are considerably higher than for the rest of the Life of Mine as a result of the reduced production rate and the high level of mine development. This is indicated in the following table.

Table 104 - Operating Costs - first 3 years and Life of Mine

Operating Costs to Concentrate			
	R/tonne	R/kg	$US/ounce
Life of Mine	525.30	135,335	$ 526.17
First 3 years	717.08	139,839	$ 543.69
After First 3 years	491.23	134,208	$ 521.79

Cost Increase from Feasibility Study

The working cost increase of 15.9% can be identified as being attributable to the increased power cost and the changed mining costs as a result of the inclusion of the equipment rebuilds in working costs rather than in capital. The comparison between the Feasibility Study and the Updated Feasibility Study is indicated in Table 105.

Table 105 - Operating Cost comparison

		Feasibility	Updated Feasibility	% Change
TOTAL	R	13,680,569,966	17,021,849,053	24%
Mining	R	10,739,038,484	12,778,164,011	19%
Processing	R	1,380,692,316	1,756,541,980	27%
Stockpile Rehandle	R	4,976,756	-	
Tailings Disposal	R	28,657,628	30,504,857	6%
Analytical & Assay	R	117,441,402	157,693,532	34%
Conc. Transport	R	43,290,912	44,796,238	3%
Conc. Treatment	R	392,478,738	406,126,142	3%
Services	R	482,779,589	1,308,395,259	171%
Admin.& Overheads	R	373,674,890	413,443,297	11%
Rehabilitation Fund	R	117,539,252	126,183,736	7%
		Cost to Final metal	Cost to Final metal	% Change
TOTAL	R/tonne milled	465.57	539.59	16%
Mining	R/tonne milled	365.46	405.07	11%
Processing	R/tonne milled	46.99	55.68	19%
Stockpile Rehandle	R/tonne milled	0.17	0.00	-100%
Tailings Disposal	R/tonne milled	0.98	0.97	-1%
Analytical & Assay	R/tonne milled	4.00	5.00	25%
Conc. Transport	R/tonne milled	1.47	1.42	-4%
Conc. Treatment	R/tonne milled	13.36	12.87	-4%
Services	R/tonne milled	16.43	41.48	152%
Admin.& Overheads	R/tonne milled	12.72	13.11	3%
Rehabilitation Fund	R/tonne milled	4.00	4.00	0%

The differential increase in the total costs versus the cost per tonne milled is a result of the 8% increase in volume being processed.

Item 21(r): Economic Evaluation

The economic valuation model used for the WBJV Project 1 mine is used to determine the Nett Present Value (NPV) at various discount rates and Internal Rate of Return (IRR) for the project.

This financial model requires the following inputs to be determined by the project team:

- Production Schedule - as derived by the Project Team
- Project Capital Expenditure - as derived and modified by the Project Team and detailed in the Capital Estimate Schedules
- Working Costs per tonne Milled - as derived and modified by the Project Team and detailed in the Working Cost Schedule
- Project Schedule - as derived by the Project Team
- Metal Price for PGM Basket
  - Case A - Base Case - 3 year trailing average metal prices
  - Case B - September 2009 spot prices
  - Case C - 'highroad' long term price
  - Case D - 'low road' long term metal price
- Mining Royalty - as interpreted from recent draft legislation

Neither escalation nor inflation factors have been applied within the financial model. The estimated capital cost and operating cost is base dated September 2009.

Metal Price

In summary the metal prices to be used in the Updated Feasibility Study are summarised in the following table.

Table 106 - Summary of Commodity Price Used and Forecast

as at 24 September 2009						Long Term (BFS)	
		3 year Price	2 year Price	1 year Price	Sep-09	High Road	Low Road
		Case A3	Case A2	Case A1	Case B	Case C	Case D
Pt	$US/oz	$ 1,343	$ 1,402	$ 1,078	$ 1,295	$ 1,450	$ 1,100
Pd	$US/oz	$ 322	$ 311	$ 226	$ 296	$ 600	$ 400
Rh	$US/oz	$ 4,951	$ 4,588	$ 1,469	$ 1,650	$ 2,500	$ 2,000
Au	$US/oz	$ 807	$ 883	$ 896	$ 997	$ 843	$ 500
Ir	$US/oz	$ 439	$ 441	$ 431	$ 425	$ 300	$ 250
Ru	$US/oz	$ 349	$ 257	$ 118	$ 90	$ 300	$ 250
Cu	$US/tonne	$ 6,428	$ 6,113	$ 4,469	$ 6,250	$ 5,000	$ 4,500
Ni	$US/tonne	$ 25,529	$ 19,262	$ 12,966	$ 18,000	$ 20,000	$ 11,000

The 'basket price' for all cases is detailed in the following table for reference.

Table 107 - Basket Price

as at 24 September 2009							Long Term	
			3 year Price	2 year Price	1 year Price	Sep-09	High Road	Low Road
			Case A3	Case A2	Case A1	Case B	Case C	Case D
Estim. Exchange Rate			R 8.000	R 8.000	R 8.000	R 8.000	R 8.000	R 8.000
MR	4E's	$US/oz	$ 1,185	$ 1,209	$ 855	$ 1,024	$ 1,232	$ 917
		R/kg	R 304,707	R 310,841	R 219,796	R 263,488	R 316,916	R 235,858
UG2	4E's	$US/oz	$ 1,433	$ 1,431	$ 894	$ 1,068	$ 1,328	$ 1,002
		R/kg	R 368,533	R 368,158	R 229,894	R 274,606	R 341,552	R 257,720

Off-take Agreement

A draft concentrate off-take agreement is in place between PTM, Wesizwe and Anglo Platinum and this forms part of the Joint Venture agreement. This draft agreement defines the outlines of the treatment contract that will be applied to the WBJV Project 1 mine.

The basic conditions that have been incorporated into the financial model are as indicated below:

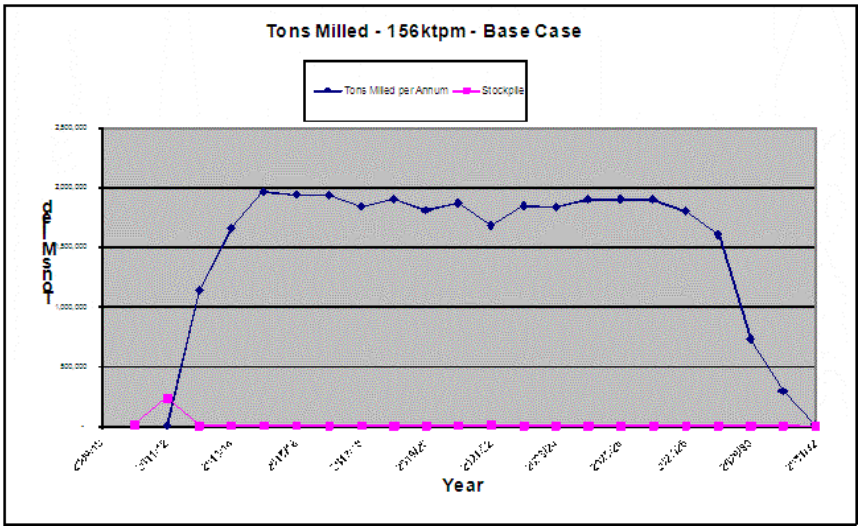
- Payable metals in concentrate shall be
  - Platinum - Pt
  - Palladium - Pd
  - Rhodium - Rh
  - Gold - Au
  - Ruthenium - Ru
  - Iridium - Ir
  - Copper - Cu
  - Nickel - Ni
- Penalties shall be applied for the following:
  - Chromitite content above a minimum limit - stated as 1% with a sliding scale
  - Moisture content above a set limit - estimated to be 15%
  - Minimum concentrate 4E’s grade - estimated to be 150 g/t
- Charges shall be applied on the following basis
  - Treatment charge per dry tonne received
  - Sampling charges per batch received
- Payment terms shall be
  - Three months after delivery of concentrate

The final terms and condition have not been negotiated between the parties involved but indicative terms have been presented by PTM to AP for comment.

Production

The production profile for the concentrator is depicted in the following graph, with the stockpile also indicated.

Figure 41 - Combined Merensky & UG2 Production Profile - tonnes per annum



The mine will produce a total of 31.5 million tonnes of Merensky Reef and UG2 Reef during its life and the mill feed will contain 145 tonnes of 4E's. The concentrator will deliver to the smelter some 122 tonnes of 4E contained in 792,441 tonnes of concentrate.

The metallurgical test work that has been conducted at Mintek is complete and the expected recovery and grade performances have been determined from this metallurgical test work and are expected to be as detailed below, for the concentrator operating at 'Name Plate' capacity.

Merensky:

Concentrate Grade

150 g/t 4E

3.15% Cu

4.22% Ni

<1.0% Chromite - assume 0.99%

Recovery

87.5% 4E

68.9% Cu

44.6% Ni

UG2:

Concentrate Grade

165 g/t 4E

2.77% Chromite

Recovery

82.5% 4E

40% Cu

30% Ni

The data generated from the test work indicates that this performance can be expected from the concentrator once operations have stabilized.

As the plant will be processing up to 160,000 tonnes per month, the recovery has been discounted as detailed in the Table 108 .

Table 108 - Predicted Plant Recovery in excess of 'name plate' capacity

Tonnage		4E Recovery		Cu Recovery		Ni Recovery	
per Month	per Annum	MR	UG2	MR	UG2	MR	UG2
130,000	1,560,000	87.50%	82.50%	68.90%	40.00%	44.60%	30.00%
135,000	1,620,000	87.50%	82.50%	68.90%	40.00%	44.60%	30.00%
140,000	1,680,000	87.50%	82.50%	68.90%	40.00%	44.60%	30.00%
145,000	1,740,000	87.00%	82.00%	68.50%	39.60%	44.25%	29.65%
150,000	1,800,000	86.50%	81.50%	68.10%	39.20%	43.90%	29.30%
155,000	1,860,000	86.00%	81.00%	67.70%	38.80%	43.55%	28.95%
160,000	1,920,000	85.50%	80.50%	67.30%	38.40%	43.20%	28.60%
165,000	1,980,000	85.00%	80.00%	66.90%	38.00%	42.85%	28.25%

The 4E metal production for the WBJV Project 1 mine is shown in the following graphs for both annual kilogram and annual ounce in concentrate and sold.

Figure 42 - Production Profile - kg per annum

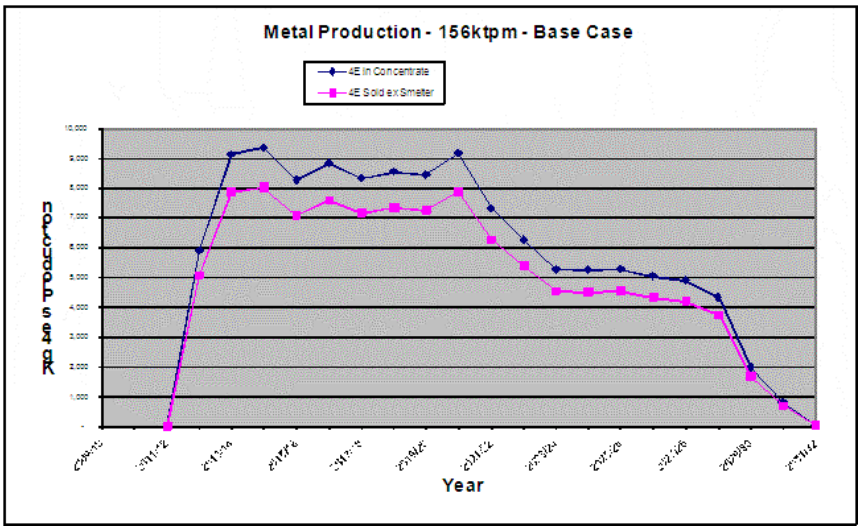
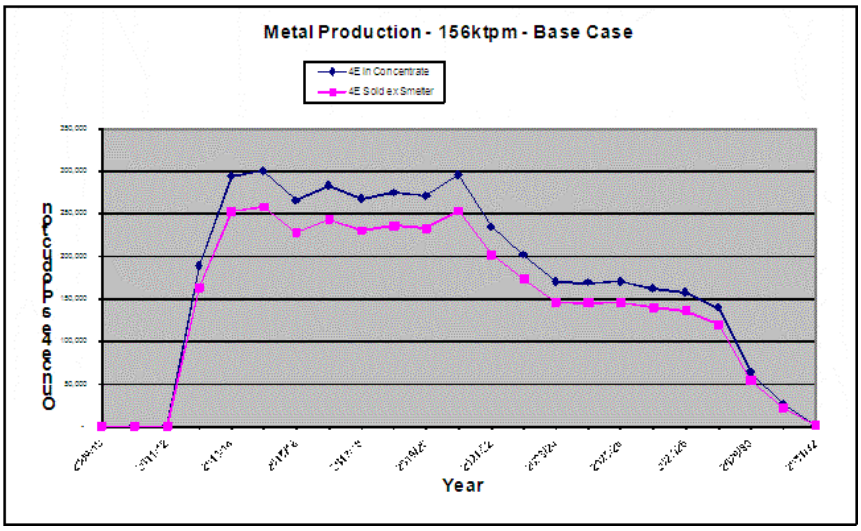


Figure 43 - Production Profile - ounces per annum



The metal in concentrate will be nominally 275,000 ounces per annum for the first nine years of steady state production, with metal sold being at 236,000 ounces per annum for the same period.

Penalties

The total penalties that are expected during the Life of Mine are primarily related to chromite inclusion from UG2 mining and amount to less than 0.3% of life of mine revenue. Total penalties during the life of the mine are expected to exceed R117 million.

This has been included in the financial model.

Royalties

The South African government has released a number of draft Royalty Bill revisions for comment commencing in March 2003, October 2006, November 2007 and more recently in mid-2009. The modified Royalty Bill notice that was released in mid-2009 has been incorporated into the Technical Report.

The Mid-2009 revision of the Royalty Bill proposes that a royalty would be payable on the following basis:

- Royalty Payable = (aggregate gross sales - allowable deductions) x royalty rate
- The royalty rate is to be based on the following formula:

$$0.5 + \frac{\text{Earnings before interest, taxes x 100}}{\text{Aggregate Gross Sales x 12.5}}$$

This revision is different to previous versions that proposed a fixed royalty rate would be applied to the aggregate gross sales - the royalty rate would be different for each mineral. In the mid-2009 revision, the maximum royalty rate is 7% for unrefined metal and 5% for refined metal.

Due to the uncertainty of the progression of the Royalty Bill revisions, the Updated Feasibility Study is based on a royalty rate of 3% of gross revenue from PGE's and 2% for base metals with a financial sensitivity at various royalty rates.

Economic Valuation

The economic evaluation for each of the six metal price scenarios as listed in Table 106 is summarised in the following sections.

Case A3 - 3 year trailing prices

The internal rate of return (IRR) and net present value (NPV) at different discount rates, as at September 2009, are calculated in a pre-tax and post-tax scenario and are detailed as follows:

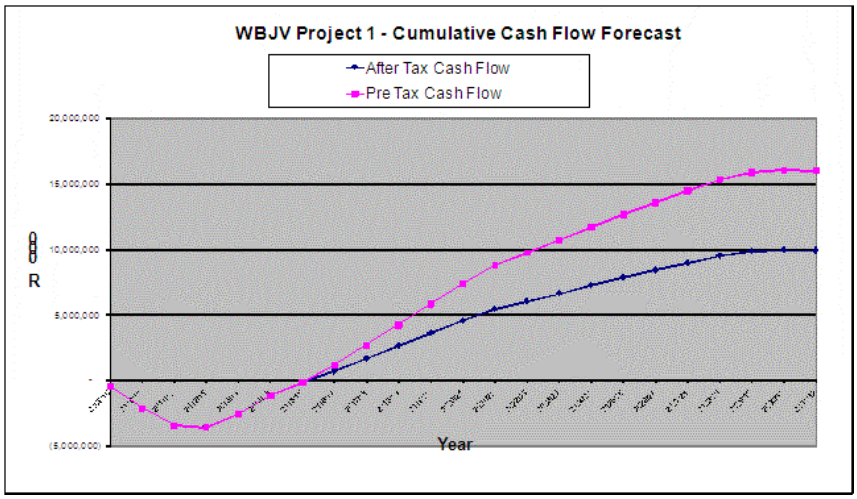
Table 109 - 'Case A3' Financial Parameters

PRE-TAX Net Present Value @ Sept. 2009		
Discount Rate	000's Rand	000's $US
5.0%	7,847,681	$ 980,960
8.0%	5,108,274	$ 638,534
10.0%	**3,803,526**	**$ 475,441**
12.5%	2,577,642	$ 322,205
Pre-Tax IRR	23.54%	
Payback	7.1	

POST-TAX Net Present Value @ Sept. 2009		
Discount Rate	000's Rand	000's $US
5.0%	4,660,043	$ 582,505
8.0%	2,881,462	$ 360,183
10.0%	**2,030,769**	**$ 253,846**
12.5%	1,229,227	$ 153,653
Pre-Tax IRR	18.94%	
Payback	7.2	

The project cash flow graph is shown below.

Figure 44 - 'Case A3' Cash Flow Profile



The peak funding requirement will be R3 545.2 million.

The other metal price cases all form similar trends to that detailed above as 'Case A3' and the results of these evaluations are detailed as Table 112 .

Item 21(s): Sensitivity Analysis

Sensitivities to the base case financial model are to be determined for the following factors for a ± 20% variation:

- Capital cost
- Working cost
- Basket metal prices
- Head grade

In addition, the following metal price scenarios will be evaluated:

- Case A3 - 3 year trailing price
- Case A2 - 2 year trailing price
- Case A1 - 1 year trailing price
- Case B - June 2008 ‘spot’ price
- Case C - ‘high road’ long term metal price
- Case D - ‘low road’ long term metal price

Due the uncertainty of the following factors, variations in these inputs will also be evaluated:

- Changes in Royalty Bill factors
- Increase in power and water costs

Case A3 - 3 year trailing prices

The sensitivity analysis has been developed for the base case (Case A3) financial model with ±20% variations in Basket Price, Capital Cost, Working Cost and head grade. The following table summarises this sensitivity analysis.

Pre-Tax Review

Table 110 - 'Case A3' - Sensitivity Data - Pre-tax

WBJV PROJECT 1
SENSITIVITY ANALYSIS
PRE-TAX

Parameter	Change in Parameter	Change in Parameter	Change in Parameter
PGM Basket Price	-20%	0%	20%
IRR (pre-tax)	14.5%	23.5%	31.2%
NPV (5% Discount) (R'000)	R 3,494,034	R 7,847,681	R 12,202,493
NPV (10% Discount) (R'000)	R 1,110,415	R 3,803,526	R 6,497,650
NPV (12.5% Discount) (R'000)	R 409,388	R 2,577,642	R 4,746,843
Opex	-20%	0%	20%
IRR (after tax)	27.5%	23.5%	19.4%
NPV (5% Discount) (R'000)	R 9,835,360	R 7,847,681	R 5,861,167
NPV (10% Discount) (R'000)	R 5,058,937	R 3,803,526	R 2,549,128
NPV (12.5% Discount) (R'000)	R 3,599,981	R 2,577,642	R 1,556,249
Capex	-20%	0%	20%
IRR (after tax)	28.4%	23.5%	19.9%
NPV (5% Discount) (R'000)	R 8,635,275	R 7,847,681	R 7,061,252
NPV (10% Discount) (R'000)	R 4,476,996	R 3,803,526	R 3,131,069
NPV (12.5% Discount) (R'000)	R 3,205,896	R 2,577,642	R 1,950,334
Grade	-20%	0%	20%
IRR (after tax)	14.5%	23.5%	31.2%
NPV (5% Discount) (R'000)	R 3,494,034	R 7,847,681	R 12,202,493
NPV (10% Discount) (R'000)	R 1,110,415	R 3,803,526	R 6,497,650
NPV (12.5% Discount) (R'000)	R 409,388	R 2,577,642	R 4,746,843

The above data is shown visually in the following graph for NPV at the 5% discount rate on a pre-tax basis.

Figure 45 - Case A3 - NPV Sensitivity Graph - Pre-tax @ 5%

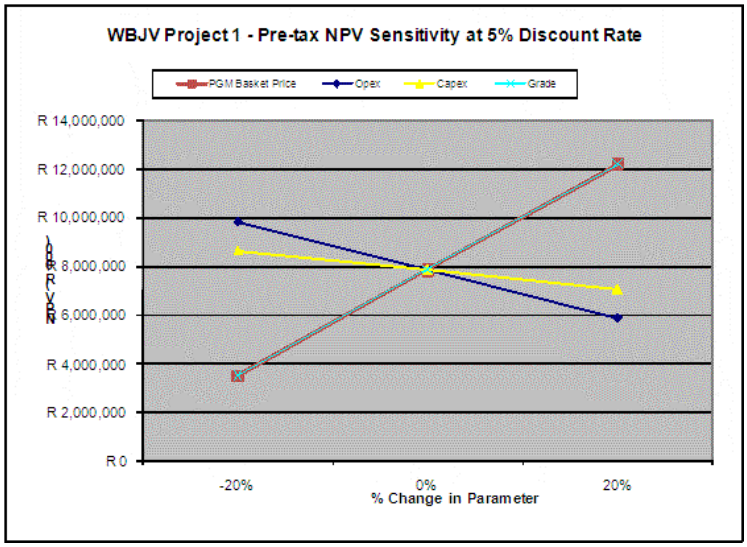
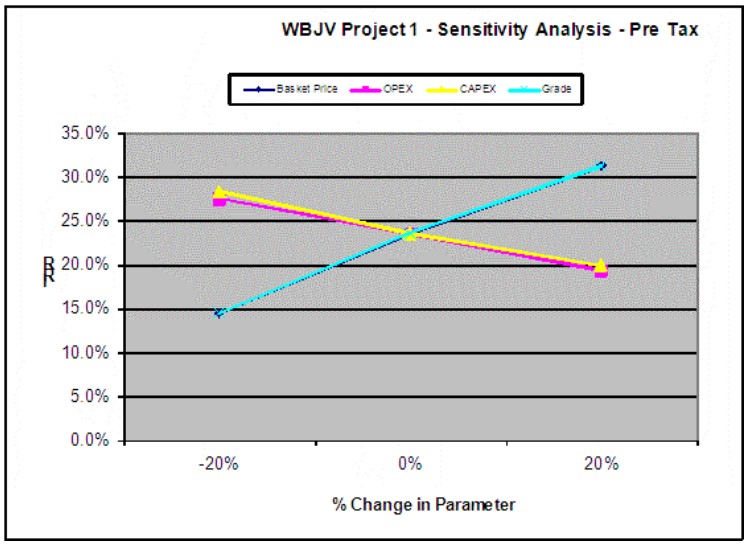


Figure 46 - Case A3 - IRR Sensitivity Graph - Pre-tax @ 5%



Post-Tax Review

On a post-tax basis, the following table and graph indicate the sensitivity to variations in the base parameters.

Table 111 - 'Case A3' - Sensitivity Data - Post-tax

WBJV PROJECT 1
SENSITIVITY ANALYSIS
POST-TAX

Parameter	Change in Parameter	Change in Parameter	Change in Parameter
PGM Basket Price	-20%	0%	20%
IRR (post-tax)	11.3%	18.9%	25.3%
NPV (5% Discount) (R'000)	R 1,891,545	R 4,660,043	R 7,393,426
NPV (10% Discount) (R'000)	R 266,241	R 2,030,769	R 3,747,375
NPV (12.5% Discount) (R'000)	-R 214,893	R 1,229,227	R 2,623,644
Opex	-20%	0%	20%
IRR (post-tax)	22.3%	18.9%	15.4%
NPV (5% Discount) (R'000)	5,916,608	R 4,660,043	R 3,393,336
NPV (10% Discount) (R'000)	2,841,134	R 2,030,769	R 1,206,106
NPV (12.5% Discount) (R'000)	1,897,123	R 1,229,227	R 546,355
Capex	-20%	0%	20%
IRR (post-tax)	23.0%	18.9%	15.9%
NPV (5% Discount) (R'000)	R 5,201,712	R 4,660,043	R 4,110,855
NPV (10% Discount) (R'000)	R 2,532,360	R 2,030,769	R 1,518,460
NPV (12.5% Discount) (R'000)	R 1,712,552	R 1,229,227	R 734,647
Grade	-20%	0%	20%
IRR (post-tax)	11.3%	18.9%	25.3%
NPV (5% Discount) (R'000)	R 1,891,545	R 4,660,043	R 7,393,426
NPV (10% Discount) (R'000)	R 266,241	R 2,030,769	R 3,747,375
NPV (12.5% Discount) (R'000)	-R 214,893	R 1,229,227	R 2,623,644

Figure 47 - Case A2 - NPV - Sensitivity Graph - Post tax @ 5%

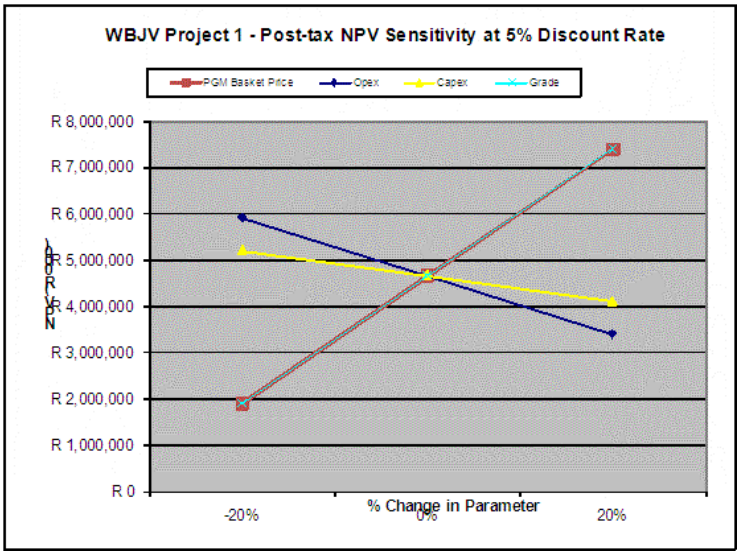
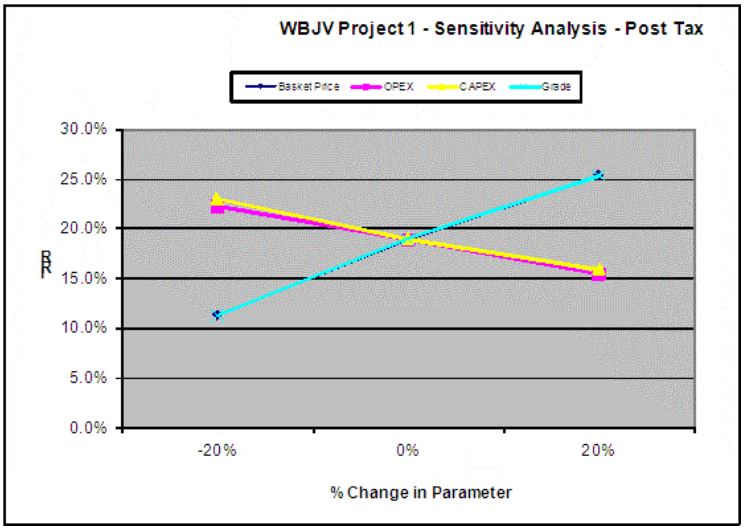


Figure 48 - Case A3 - IRR Sensitivity Graph - Post-tax @ 5%



Analysing the above tables and graphs, it is apparent that the project is most sensitive to the head grade and the metal price and least sensitive to capital costs and operating costs.

The above trends are evident for all the metal price cases evaluated.

Metal Prices

The six metal price cases, namely:

- Case A3 - 3 year trailing price
- Case A2 - 2 year trailing price
- Case A1 - 1 year trailing price
- Case B - June 2008 ‘spot’ price
- Case C - ‘high road’ long term metal price
- Case D - ‘low road’ long term metal price

The different economic parameters are evaluated in the following table for comparison purposes.

Table 112 - Metal Price Sensitivity Analysis

			5% Discount	10% Discount	12.5% Discount
Case A3	Pre-Tax	IRR%	23.54%	23.54%	23.54%
		R'000,000	R 7,847.68	R 3,803.53	R 2,577.64
		US$'000,000	$ 980.96	$ 475.44	$ 322.21
	Post-Tax	IRR%	18.94%	18.94%	18.94%
		R'000,000	R 4,660.04	R 2,030.77	R 1,229.23
		US$'000,000	$ 582.51	$ 253.85	$ 153.65
Case A2	Pre-Tax	IRR%	23.04%	23.04%	23.04%
		R'000,000	R 7,532.47	R 3,619.91	R 2,433.65
		US$'000,000	$ 941.56	$ 452.49	$ 304.21
	Post-Tax	IRR%	18.50%	18.50%	18.50%
		R'000,000	R 4,461.60	R 1,912.95	R 1,135.77
		US$'000,000	$ 557.70	$ 239.12	$ 141.97
Case A1	Pre-Tax	IRR%	7.41%	7.41%	7.41%
		R'000,000	R 710.76	R -531.52	R -886.28
		US$'000,000	$ 88.84	$ -66.44	$ -110.79
	Post-Tax	IRR%	5.37%	5.37%	5.37%
		R'000,000	R 95.83	R -842.81	R -1,111.66
		US$'000,000	$ 11.98	$ -105.35	$ -138.96
Case B	Pre-Tax	IRR%	15.63%	15.63%	15.63%
		R'000,000	R 3,670.86	R 1,329.81	R 622.21
		US$'000,000	$ 458.86	$ 166.23	$ 77.78
	Post-Tax	IRR%	12.15%	12.15%	12.15%
		R'000,000	R 2,016.38	R 424.50	R -59.43
		US$'000,000	$ 252.05	$ 53.06	$ -7.43
Case C	Pre-Tax	IRR%	23.49%	23.49%	23.49%
		R'000,000	R 7,449.99	R 3,644.73	R 2,478.57
		US$'000,000	$ 931.25	$ 455.59	$ 309.82
	Post-Tax	IRR%	18.84%	18.84%	18.84%
		R'000,000	R 4,414.19	R 1,934.62	R 1,170.45
		US$'000,000	$ 551.77	$ 241.83	$ 146.31
Case D	Pre-Tax	IRR%	10.69%	10.69%	10.69%
		R'000,000	R 1,871.07	R 154.63	R -344.18
		US$'000,000	$ 233.88	$ 19.33	$ -43.02
	Post-Tax	IRR%	8.07%	8.07%	8.07%
		R'000,000	R 849.41	R -373.38	R -730.63
		US$'000,000	$ 106.18	$ -46.67	$ -91.33

The three year trailing prices as shown in Case A3 has a very similar economic performance to the ‘High Road’ long term metal prices.

The September 2009 metal prices are very similar to the ‘Low Road’ long term metal prices which would result is a difficult project which would need to be revaluated by the owners.

Recovery

The concentrator recovery has been discounted as a result of the throughput being in excess of the 140,000 tonnes per month ‘name plate’ capacity. If the concentrator recovery can be maintained at the ‘name plate’ recovery of 87.5% for Merensky and 82.5% for UG2, then the sensitivity analysis is indicated in the following table.

Table 113 - Sensitivity to Concentrator Recovery

			5% Discount	10% Discount	12.5% Discount
Case A3 - Variable Recovery with increased tonnes milled	Pre-Tax	IRR%	23.54%	23.54%	23.54%
		R'000,000	R 7,847.68	R 3,803.53	R 2,577.64
		US$'000,000	$ 980.96	$ 475.44	$ 322.21
	Post-Tax	IRR%	18.94%	18.94%	18.94%
		R'000,000	R 4,660.04	R 2,030.77	R 1,229.23
		US$'000,000	$ 582.51	$ 253.85	$ 153.65
Case A3 - 87.5% for MR and 82.5% for UG2	Pre-Tax	IRR%	24.22%	24.22%	24.22%
		R'000,000	R 8,238.24	R 4,043.20	R 2,768.99
		US$'000,000	$ 1,029.78	$ 505.40	$ 346.12
	Post-Tax	IRR%	19.51%	19.51%	19.51%
		R'000,000	R 4,904.60	R 2,182.74	R 1,351.41
		US$'000,000	$ 613.08	$ 272.84	$ 168.93

There is a significant improvement in economic performance if the concentrator recovery can be maintained at the ‘name plate’ capacity levels. It may be necessary to consider increasing the ‘name plate’ capacity by the inclusion of additional capital, but this has not been included in this sensitivity analysis.

Mining Royalty

The uncertainty in the royalty that may be promulgated for platinum mines producing concentrate requires a sensitivity to be conducted with up side and down side potential. The base case has a 3% royalty for PGM’s and a 2% royalty for base metals.

The sensitivity analysis has been conducted at 1.5% for PGM’s with 1% for base metals and 6% for PGM’s with 4% for base metals.

Table 114 - Sensitivity to Mining Royalty

			5% Discount	10% Discount	12.5% Discount
Case A3 - 3% for PGM's and 2% for Base Metals	Pre-Tax	IRR%	23.54%	23.54%	23.54%
		R'000,000	R 7,847.68	R 3,803.53	R 2,577.64
		US$'000,000	$ 980.96	$ 475.44	$ 322.21
	Post-Tax	IRR%	18.94%	18.94%	18.94%
		R'000,000	R 4,660.04	R 2,030.77	R 1,229.23
		US$'000,000	$ 582.51	$ 253.85	$ 153.65
Case A3 - 1.5% for PGM's and 1% for Base Metals	Pre-Tax	IRR%	24.20%	24.20%	24.20%
		R'000,000	R 8,179.71	R 4,011.74	R 2,746.38
		US$'000,000	$ 1,022.46	$ 501.47	$ 343.30
	Post-Tax	IRR%	19.50%	19.50%	19.50%
		R'000,000	R 4,870.51	R 2,166.49	R 1,340.93
		US$'000,000	$ 608.81	$ 270.81	$ 167.62
Case A3 - 6% for PGM's and 4% for Base Metals	Pre-Tax	IRR%	22.21%	22.21%	22.21%
		R'000,000	R 7,183.63	R 3,387.11	R 2,240.17
		US$'000,000	$ 897.95	$ 423.39	$ 280.02
	Post-Tax	IRR%	17.81%	17.81%	17.81%
		R'000,000	R 4,239.10	R 1,759.33	R 1,005.83
		US$'000,000	$ 529.89	$ 219.92	$ 125.73

The project is sensitive to the royalty percentage that will be promulgated in the near future as indicated above.

Power and Water Costs

The reported increases in ESKOM power costs of 45% per annum for the next 3 years will impact directly on the power costs, but there will also be an indirect impact upon the cost of water supplied to the WBJV mine. The base case has a power cost of R0.32/kWhr and a water cost of R2.42/kL.

The sensitivity analysis has been conducted at R0.65/kWhr and R0.97/kWhr for power and R2.63/kL and R2.84/kL respectively for water.

Table 115 - Sensitivity to Power and Water supply costs

			5% Discount	10% Discount	12.5% Discount
Case A3 - Power @ R0.32/kWhr & Water @ R2.42/kL	Pre-Tax	IRR%	23.54%	23.54%	23.54%
		R'000,000	R 7,847.68	R 3,803.53	R 2,577.64
		US$'000,000	$ 980.96	$ 475.44	$ 322.21
	Post-Tax	IRR%	18.94%	18.94%	18.94%
		R'000,000	R 4,660.04	R 2,030.77	R 1,229.23
		US$'000,000	$ 582.51	$ 253.85	$ 153.65
Case A3 - Power @ R0.65/kWhr & Water @ R2.63/kL	Pre-Tax	IRR%	22.02%	22.02%	22.02%
		R'000,000	R 7,124.76	R 3,351.41	R 2,209.40
		US$'000,000	$ 890.59	$ 418.93	$ 276.18
	Post-Tax	IRR%	17.64%	17.64%	17.64%
		R'000,000	R 4,195.70	R 1,730.06	R 979.34
		US$'000,000	$ 524.46	$ 216.26	$ 122.42
Case A3 - Power @ R0.97/kWhr & Water @ R2.84/kL	Pre-Tax	IRR%	20.54%	20.54%	20.54%
		R'000,000	R 6,423.61	R 2,912.92	R 1,852.26
		US$'000,000	$ 802.95	$ 364.11	$ 231.53
	Post-Tax	IRR%	16.37%	16.37%	16.37%
		R'000,000	R 3,745.34	R 1,438.42	R 736.99
		US$'000,000	$ 468.17	$ 179.80	$ 92.12

The project is very sensitive to the power and water costs that are reported to be increasing significantly during the coming years. The operating costs will increase from R539.59 per tonne milled at current pricing to R579.83per tonne milled for the interim increase and to R618.85 per tonne milled for the anticipated final increase.

Item 21(t): Human Resources and Housing

This section of the report outlines the labour requirements, labour availability, labour sourcing areas, as well as Human Resource Operational Readiness related matters for the Western Bushveld JV Mine Project, considered during the Updated Feasibility Study Phase. Based on the mine and technical design parameters obtained from the technical design team, this hybrid mining operation will produce 156,000 reef tons per month at steady state production. The planned labour strategy for this project is based on the model where all mining and plant operations will be outsourced to third party contractors, hence all labour requirements and cost estimates presented in this report are based on information obtained from potential contracting companies.

The planned WBJV Mine will be a Green Field operation. It is assumed that, in addition to a small owner's team who will be responsible for day to day project management during the development stage and management of the facility once operational, contractors will be responsible for the day to day operation and production targets. The bulk of the labour requirements associated with this project will therefore be the sourced via the selected mining contractors.

From a labour sourcing and people management perspective, it is important that cognisance is taken of the fact that local communities within the surrounding areas of the mine site will have job expectations. This will require pro-active engagement of local tribal and provincial authorities to ensure that sound labour and stakeholder relations are established from the start. The WBJV project partners are committed to recruit the required labour to operate the planned mine facility, from the surrounding communities as far as possible. Mining Contractors and other third party service providers will therefore be bound to source their labour from the local communities within the surrounding area of the planned mine site and align HR practices with the guiding principles stipulated in the Social and Labour Plan.

The Human Resource Strategy is focused on creating a culture in which competent, committed employees work together to achieve the facility's operational and safety targets. This will be achieved through:

- The employment of competent and suitably qualified employees at all levels, targeting the HDSA population.
- The development of these employees through well defined and communicated career development plans.
- Upfront and continuous training of resources to ensure a competent work force.
- Open, honest and continuous communication with all stakeholders.
- Long term, stable relationships with all stakeholders, especially the local communities, employees, Trade Unions & Associations, and Contractors.
- Strengthened efforts related to HIV/AIDS education and training on all levels.
- The achievement of Mining Charter requirements.

The steady state labour complement for this planned mining operation as presented by the Mining Contractor is calculated at 2,387 which exclude unavailables, concentrator, and outsourced labour such as mine security and cleaning services. It is however important to note that, although these labour complements have not been included the costs associated with each one have been included in the financial model. The un-availables number is calculated at 298 and is based on 15% of lower level production related occupations, to make additional provision for training, leave relief, absentees, sick leave etc. Hence the labour plan reflects a total "On strength" figure of 2,685, which is the sum of an "At work" figure and un-availables.

The proposed operating cycle is based on the traditional 11 shift fortnight, with one production shifts per day.

The steady state labour unit cost (including un-availables) for the project is estimated at R 199 per tonne milled. The steady state monthly labour cost associated with the proposed labour complement is estimated R28m per month, with an average rate of R 10 499 per employee.

Although the provision of accommodation and a social responsible environment for all employees is not regarded as core-business, the WBJV through industry specialists engaged in some extensive exploration of different possibilities to address the housing requirements. The WBJV will oversee the process of provision of bulk infrastructure to facilitate access to accommodation for the WBJV Mine employees; regardless of the fact that the planned labour strategy is based on contract mining. Failure to do so will affect the attraction and retention of prospective employees for the project.

The implementation of the mine is to be based on a small WBJV management team supported by a number of specialist contractors as depicted in the following organograms.

Figure 49 - Proposed Organogram for Implenation and Construction

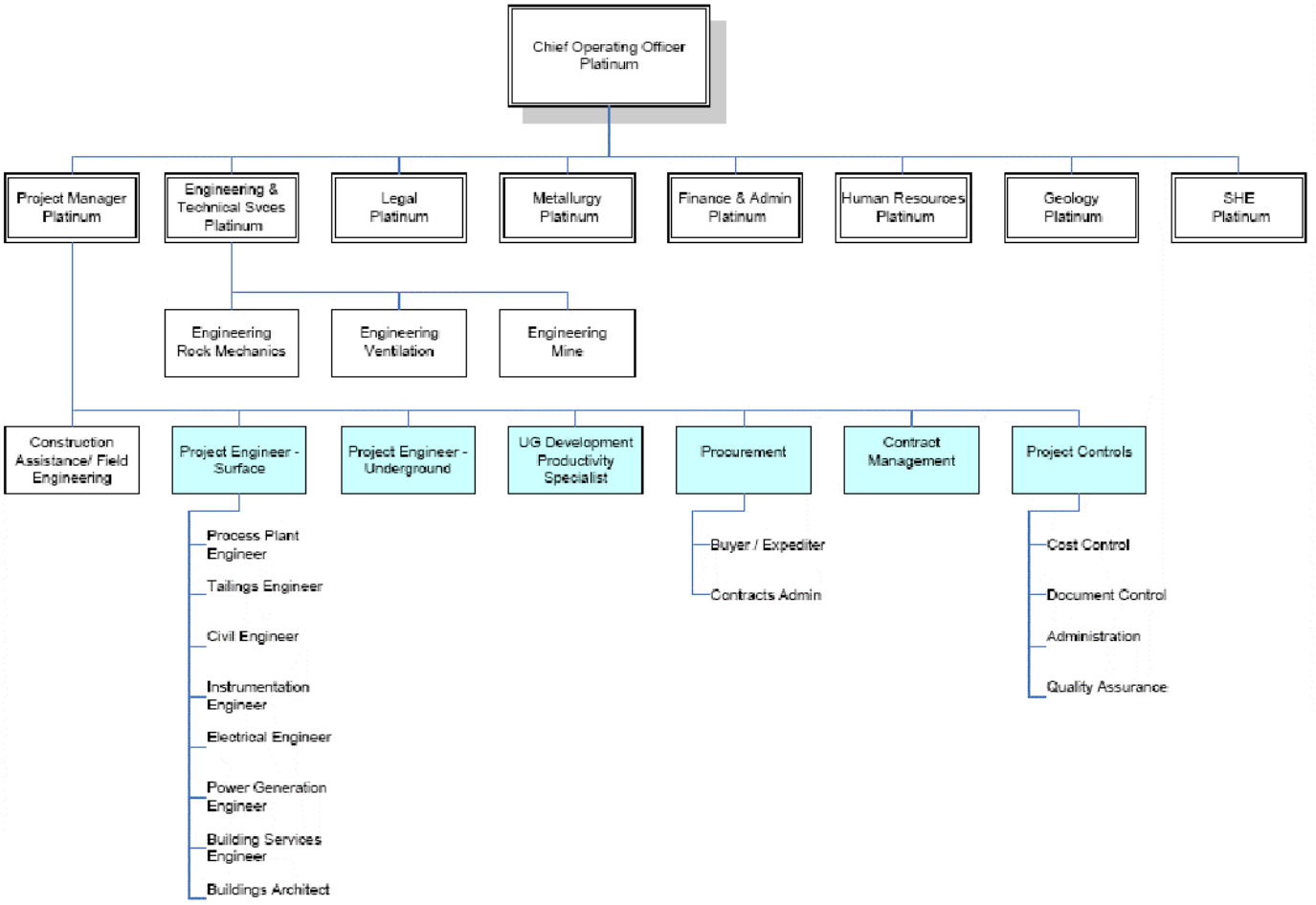
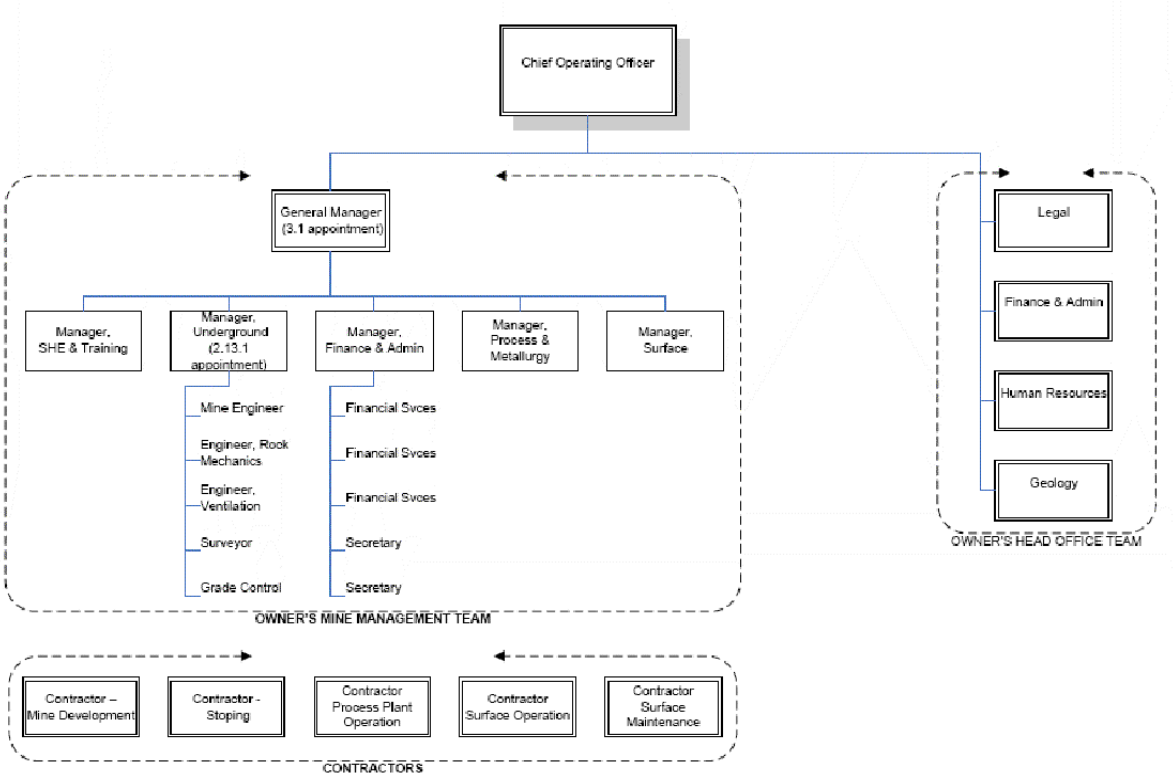


Figure 50 - Proposed Organogram for Production



Item 21(u): Project Risks and Opportunities

The following table summarises a number of the risks identified for the project and this should be read in conjunction with the formal risk assessment:

Table 116 - Risk Analysis

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.
	Decline Development Rates	Not achieving the 100m decline development advance rates will delay first ore production and this will impact severely on the overall project schedule and financials
	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.
Decline 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 - ESKOM	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.
	Power Supply - Self Generation	As a result of the power crisis with ESKOM, the WBJV is to purchase diesel and HFO power generation equipment for a number of years thus incurring the capital and operating costs of such units.
	Land Ownership	Without the hotel being part of the mining areas, the project will need to be redesigned and additional infrastructure will be required. In addition, the hotel was planned to provide the construction camp.
	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
	Decline Portal 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	Formation of Newco	Formation of Newco is to be finalised by the partners
	BEE	BEE partner in place with Wesizwe
	Local community	Local community involvement
	MRA Licences	Application not submitted to the DME.
	Prospecting License amendment	Application not submitted to the DME.

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.
Capital Cost	Equipment	Major equipment price increases.
	Products	Escalation of all products.
	Steel	Steel price increases.
Working Cost	Wages	Wage increases.
	Consumables	Consumables price increases - escalation.
	Consumption	Increased consumptions of items - increased costs.
Revenue	Metal price	Real 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.

As the WBJV Project 1 mine is expected to be a low cost producer of platinum group metals, many risks will be ameliorated. These risks could also become future opportunities if proved incorrect.

A formal risk assessment was conducted and the report is available. This highlighted all of the risks that would be expected for such a project.

Opportunities

There are a significant number of opportunities that can be considered for the WBJV Project 1 including:

- metal price improvements and exchange rate weakening
- improvement in the structural complexity with reduced mining costs
- purchase of second hand equipment such as power generators
- sale of equipment once project has used this equipment with added revenue
- residual value of equipment at end of mine life to assist with rehabilitation cost
- combination of infrastructure with neighbouring mine to develop the synergies with these operations

ITEM 22: 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. Part of this Resource has been converted into a Mineral Reserve with appropriate financial factors having been applied.

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.

Mining Project

Reviewing the above details of the project, it is concluded that the project is viable, although there are risks associated with the timing of the project.

The economic analysis is positive, but the risks include the following items:

- Surface ownership of land, especially the hotel
- Delays in the formation of ‘NEWCO’
- Delays in obtaining the Mining Right Approval
- Delays in the EIA
- Delays in the SLP
- ESKOM power availability
- The cost of self generation of power

- Availability of water from Magalies Water Board
- Metallurgical recovery

Project Economics

The WBJV Mine Project is financially robust at the 3 year historical metal prices.

ITEM 23: RECOMMENDATIONS

It is recommended that the Updated Feasibility Study be submitted by Platinum Group Metals, the Operator of the WBJV Project 1 mine, to the partners of the WBJV for consideration and approval.

Further work required

All future planned work will be related to Project Implementation and Construction and to develop design details for surface geotechnical evaluation, decline geotechnical evaluation and metallurgical and mine design.

As a result of the release of the updated Mineral Resource statement, it is strongly recommended that the mine plan again be reviewed based on the changes to the mineable Resource with the updated model. It is apparent that some of the shallower reefs are now unpay and may not be targeted early in the life of the mine, which could change the decline layout and the resulting production profile.

ITEM 24: REFERENCES

Assibey-Bonsu W and Krige DG (1999). Use of Direct and Indirect Distributions of Selective Mining Units for estimation of Recoverable Resources/Reserves for new Mining Projects. Proc. APCOM 1999, Colorado, USA.

Bredenkamp G and Van Rooyen N (1996). Clay thorn bushveld. In: Low AB and Rebelo AG (1996) Vegetation of South Africa, Lesotho and Swaziland. Department of Environmental Affairs and Tourism, Pretoria.

Cawthorn (1996). Re-evaluation of magma composition and processes in the uppermost Critical Zone of the Bushveld Complex. Mineralog. Mag. 60, pp. 131-148.

Leeb-Du Toit A (1986). The Impala Platinum Mines. Mineral Deposits of South Africa, Volume 2, pp. 1091-1106. Edited by Anhaeusser, CR and Maske, S.

Matthey J (2005). Platinum Report 2005.

MINTEK Report No.5328 . External Report titled ‘Evaluation of the Elandsfontein / Frischgewaagd PGM Deposit for the Western Bushveld Joint Venture’ by S van Zyl and MAW Bryson

Rutherford MC and Westfall RH (1994). Biomes of southern Africa: an objective categorization. National Botanical Institute, Pretoria.

SAMREC (2005). South African code for reporting of Mineral Resources and Mineral Reserves.

Schürmann LW (1993) . The Geochemistry and Petrology of the upper Critical Zone of the boshoek Section of the Western Bushveld Complex, Bulletin 113 of the Geological Survey South Africa.

SGS Lakefield Research Africa (Pty) Ltd (2005/2006). Mineralogical Reports (MIN0306/015; MIN0805/64 and MIN0805/06).

Siepker EH and Muller CJ (2004). Elandsfontein 102 JQ. Geological assessment and Resource estimation. Prepared by Global Geo Services (Pty) Ltd for PTM RSA (Pty) Ltd.

Smit PJ and Maree BD (1966). Densities of South African Rocks for the Interpretation of Gravity Anomalies. Bull. of Geol.Surv. of S.Afr, 48, Pretoria.

Stallknecht H and Rupnarain J (2006) . Comminution and Flotation Testwork on PGM Inner Dog box Core samples from the Ngonyama Deposit. Prepared by SGS Lakefield Research Africa (Pty) Ltd.

Vermaak CF (1995). The Platinum-Group Metals - A Global Perspective. Mintek, Randburg, pp. 247.

Viljoen MJ and Hieber R (1986). The Rustenburg section of the Rustenburg Platinum Mines Limited, with reference to the Merensky Reef. Mineral Deposits of South Africa, Volume 2, pp. 1107-1134. Edited by Anhaeusser, CR and Maske, S.

Viljoen MJ (1999). The nature and origin of the Merensky Reef of the western Bushveld Complex, based on geological facies and geophysical data. S. Afr. J Geol. 102, pp. 221-239.

Wagner PA (1926). The preliminary report on the platinum deposits in the south-eastern portion of the Rustenburg district, Transvaal. Mem. Geol.Surv.S Afr., 24, pp. 37.

ITEM 25: DATE

The date of this Updated Feasibility Study report is 20 November 2009.

_____

GI Cunningham

B E (Chemical) . FSAIMM, Pr Eng

ITEM 26: APPENDICES & CERTIFICATES

Appendix 1
Mine Planning & Production Schedule

Certificates
GI Cunningham
TV Spindler
CJ Muller
B Stewart

Appendix 1

Certificates

CERTIFICATE OF AUTHOR

I, Charles Johannes Muller, BSc (Hons), Pr.Sc.Nat., do hereby certify that:

1. I am currently employed as a Director by:
   Minxcon (Pty) Ltd.
   Postnet Suite No 47
   Private Bag X5
   Strubens Valley 1735

2. I graduated from the Rand Afrikaanse University (BSc. (1988) and BSc. Hons (1992)).

3. I am a member in good standing of the South African Council for Natural Scientific Professionals (SACNASP), registration number 400201/04.

4. I have worked as a geoscientist for a total of 20 years since my graduation from university.

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 fulfill 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 Thys Botha of Platinum Group Metals (RSA) (Pty) Ltd., leading up to the compilation of the report referenced herein.

7. I am co-author of the technical report entitled "Updated TECHNICAL REPORT (Updated FEASIBILITY STUDY) Western Bushveld Joint Venture PROJECT 1 (ELANDSFONTEIN AND FRISCHGEWAAGD) ", dated October 08, 2009 (the "Report"). I have reviewed the entire Report and the work of other qualified persons who contributed to the Report.

8. The mineral resource section of the Report was completed using a dataset compiled from technical data collected during this assessment phase by Platinum Group Metals Ltd. Although the dataset is the responsibility of Platinum Group Metals Ltd., I have taken reasonable steps to provide comfort that the dataset is accurate and reliable.

9. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.

10. I am independent of the issuer, Platinum Group Metals Ltd., applying all of the tests in NI 43-101.

11. I am familiar with the type of deposit found in the area visited 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.

Dated the 20th day of November 2009.

_____
Charles Johannes Muller
BSc (Hons), Pr.Sc.Nat.

CONSENT OF QUALIFIED PERSON

Attention: **Alberta Securities Commission**
Autorité des marches financiers
British Columbia Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

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 "Updated Technical Report (Updated Feasibility Study) – Western Bushveld Joint Venture – Project 1 - Elandsfontein and Frischgewaagd", dated 08 October 2009 (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.

Dated this 20 th day of November 2009.

_____

Charles Johannes Muller
BSc (Hons), Pr.Sc.Nat.

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 Fellow 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 7 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 fulfill 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 of Platinum Group Metals RSA (Pty) Ltd. on numerous occasions over the period June 2005 to October 2009.

7. I accept overall responsibility for the entire report titled "Updated Technical Report (Updated Feasibility Study) Western Bushveld Joint Venture Project 1 – Elandsfontein and Frischgewaagd)", dated 08 November 2009, as revised (the "Updated Feasibility Study").

8. I have relied upon outside sources of information used in the completion of Items 18 and 21. 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 Updated Feasibility Study as of October 8, 2009 that is not reflected in the Updated Feasibility Study, the omission to disclose which makes the Technical Report 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.

Dated the 20 th day of November 2009

Gordon Ian Cunningham
B Eng. (Chemical), Pr. Eng., FSAIMM

No.8 6 th Avenue, Melville, Johannesburg, South Africa. Email: turnbery@iafrica.com
PO Box 2199, Rivonia, 2128, South Africa Tel: (011) 726 1590 Fax: (086) 607 5125 Cell: (083) 263 9438

Director: G.I.Cunningham

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

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 "Updated Technical Report (Updated Feasibility Study) – Western Bushveld Joint Venture – Project 1 - Elandsfontein and Frischgewaagd", dated 08 October 2009 (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.

Dated this 20 th day of November 2009

Gordon Ian Cunningham
B.Eng. (Chemical), Pr.Eng.

No.8 6 th Avenue, Melville, Johannesburg, South Africa. Email: turnbery@iafrica.com
PO Box 2199, Rivonia, 2128, South Africa Tel: (011) 726 1590 Fax: (086) 607 5125 Cell: (083) 263 9438

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 Fellow 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 13 years since my graduation from university. I have worked as a Consulting Mining Engineer for 10 years since graduation. I have been associated with Turnberry Projects for 7 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 "Updated Technical Report (Updated Feasibility Study) – Western Bushveld Joint Venture Project 1 – Elandsfontein and Frischgewaagd)", dated 08 October 2009.

7. I have visited the property, viewed the core and discussed the technical issues and the geology of the project with Willie Visser of Platinum Group Metals RSA (Pty) Ltd. on numerous occasions over the period June 2005 to October 2009. I am familiar with the specific type of deposit found in the area.

8. I am responsible for portions on Item 21 of this report dealing with stoping, mine design, mine planning, scheduling and the Mineral Reserve Statement. I have relied upon outside sources of information used in the completion of Item 21. 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 modeling. 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.

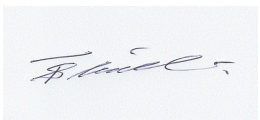
9. I am not aware of any material fact or material change with respect to the subject matter of the Updated Feasibility Study that is not reflected in the Report, the omission to disclose which makes the Technical Report misleading .

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 20 th day of November 2009.



Timothy Vyvyan Spindler
B Sc. (Mining), Pr.Eng

No.8 6 th Avenue, Melville, Johannesburg, South Africa. Email: turnbery@iafrica.com
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Director: G.I.Cunningham

Turnberry Projects

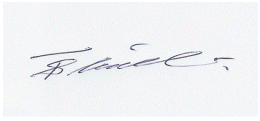
(Pty) Ltd
Reg No. 1993-/003160/07

CONSENT OF QUALIFIED PERSON

Attention: **Alberta Securities Commission**
Autorité des marches financiers
British Columbia Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

I, Timothy Vyvyan SPINDLER, B Sc. (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 "Updated Technical Report (Updated Feasibility Study) – Western Bushveld Joint Venture – Project 1 - Elandsfontein and Frischgewaagd", dated 08 October 2009 (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.

Dated this 20 th day of November 2009



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Timothy Spindler
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