PLATINUM GROUP METALS LTD

FORM 6-K
(Report of Foreign Issuer)

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

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

For: **Aug 1 - 19, 2005**

Platinum Group Metals Ltd.
(SEC File No. 0-30306)

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

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

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): [ ]

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101 (b)(7): [ ]

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

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

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

Date: **August 24, 2005**

"R. Michael Jones"
R. MICHAEL JONES
President, Director

PLATINUM GROUP METALS (RSA) (Pty) LIMITED REPUBLIC OF SOUTH AFRICA REGISTERD COMPANY

REGISTRATION NUMBER: 2000/025984/07

A WHOLLY OWNED SUBSIDIARY

INDEPENDENT PRELIMINARY ASSESSMENT

SCOPING STUDY REPORT AND RESOURCE UPDATE

Western Bushveld Joint Venture ELANDSFONTEIN PROJECT

(PROJECT 1)

GLOBAL GEO SERVICES (Pty) Ltd

TURNBERRY PROJECTS (Pty) Ltd

A REPORT ON A PORTION OF THE **WESTERN BUSHVELD JOINT VENTURE** WHICH FORMS PART OF

A NOTARIAL JOINT VENTURE AGREEMENT BETWEEN PLATINUM GROUP METALS (RSA) (Pty) LIMITED, PLATINUM GROUP

METALS LIMITED, RUSTENBURG PLATINUM MINES LIMITED AND AFRICA WIDE MINERAL PROSPECTING AND EXPLORATION

(PTY) LIMITED

8 AUGUST 2005

IMPORTANT NOTICE

This Preliminary Assessment Report includes Inferred Resources that have not been sufficiently drilled to enable them to be categorized as Reserves. Until there is additional drilling to upgrade the Inferred Resource to an Indicated Resources, there can be no certainty that the economics of this Preliminary Assessment will be realized.

The TSX Exchange has not reviewed and does not accept responsibility for the accuracy or adequacy of this news release, which has been prepared by management. There can be no assurance that any of the assumptions in the Preliminary Assessment will be supported by a Feasibility Study or will come to pass. Data is incomplete and considerable additional work will be required to complete further evaluation including but not limited to drilling, engineering and socioeconomic studies and investment. No firm quotes for costs have been received. The legal right to mine the project discussed has not been confirmed or applied for, and the process for such application is new in South Africa and untested. The potential capital cost of the project is beyond the current means of the Company and there can be no assurance that financing for further work will be available.

Note to U.S. Investors: - Investors are urged to consider closely the disclosure in the PTM Form 20F, File No. 0-30306, available at the PTM offices: Suite 328-550 Burrard Street, Vancouver BC, Canada, V6C 2B5 or from the SEC: 1(800) SEC-0330. The Company may access safe harbour rules.

The US Securities and Exchange Commission does not recognize the reporting of Inferred Resources. These resources are reported under Canadian National Instrument 43-101 and have a great amount of uncertainty and risk as to their existence and economic and legal feasibility. It cannot be assumed 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. US INVESTORS ARE CAUTIONED NOT TO ASSUME THAT PART OR ALL OF AN INFERRED RESOURCE EXISTS, OR ARE ECONOMICALLY OR LEGALLY MINEABLE.

ITEM 2 - CONTENTS

ITEM 3 - SUMMARY

Platinum Group Metals RSA (Pty) Limited ("PTM") announced a joint venture with Rustenburg Platinum Mines Limited (a subsidiary of Anglo Platinum Limited) ("RPM") and Africa Wide Mineral Prospecting and Exploration (Pty) Limited ("AW) in November 2004. This joint venture known as the Western Bushveld Joint Venture ("WBJV") includes the properties Elandsfontein 102 JQ, Onderstepoort 98 JQ, Frischgewaagd 96 JQ and Koedoesfontein 94 JQ covering a surface area some 67 km^2.

The project is situated in the western sector of the Bushveld Igneous Complex adjoining Anglo Platinum's Bafokeng Rasimone Platinum Mine and Styldrift Project which lies to the south of the project area and to the east of the project area respectively. The infrastructure in the immediate vicinity of the project area includes a paved road and railway line with platinum and chrome plants within a 10 km distance from the project.

Platinum Group Metals RSA (Pty) Ltd appointed Global Geo Services (Pty) Ltd as an independent geological consultant to provide a preliminary assessment over certain portions (Elandsfontein Project Area) of the property of the Western Bushveld Joint Venture. Turnberry Projects (Pty) Limited was appointed as the independent Metallurgical and Engineering consultants. The independent assessment covers the farms Frischgewaagd 96 JQ, portion 7 (a portion of portion 2) and portions 15 and 16 plus Elandsfontein 102 JQ, portion 12, mineral area 2 (portion of mineral area 1) situated towards the south-east of the larger joint venture entity.

This preliminary assessment was commissioned to update the resource covered in the "Western BIC Project" – Report dated March 3, 2005, within the project area and to evaluate the economic potential thereof.

The results of the resource update are for the Hartzburgite area, which contains the Merensky Reef, has an Inferred Resource of 13.87Mt at an average 3PGM+Au grade of 9.67 g/t (200cm g/t cut-off) and thus a metal content of 4.31 million ounces 3PGM+Au. The average adjusted channel width is 1.12m. Feldspathic Pegmatoidal Pyroxenite-type Merensky Reef has an Inferred Resource of 0.53 million tonnes at an average 3PGM+Au grade of 6.47g/t (400 cm g/t cut-off) and thus a metal content of 0.11 million ounces.

The mineral resource for the UG2 Reef is an Inferred Resource of 2.21 million tonnes at an average 3PGM+Au grade of 4.32g/t with a metal content of 0.31 million ounces 3PGM+Au.

The total Inferred Resource update is 4.73 million ounces compared to a previous reported 4.7 million ounces which was made up of 15.4 million tonnes grading at 7.92g/t 3PGM+Au of Merensky Reef and 10.5 million tonnes of UG2 Reef grading of 2.52g/t 3PGM+Au using a zero a cm g/t cut-off.

The results of the assessment are economically favourable and to the extent that further investment in the project should be undertaken. The further investment would involve further drilling, engineering and socio-economic and permitting investigations and applications. As the study relies on Inferred Resources and the design work is preliminary, a high level of confidence in the conclusions should not be drawn as to the economic viability of the project. The additional work will assist in determining the full potential and this would take the form of a Pre-Feasibility Study.

Regarding the geology of the project, the potentially economic horizons are the Merensky Reef and UG2 Reefs situated within the Critical Zone of the Rustenburg Suite of the Bushveld Igneous Complex. The Merensky Reef in the project area is the main target reef for possible exploitation whereas the UG2 is unlikely to have any immediate economic potential.

The TSX Exchange has not reviewed and does not accept responsibility for the accuracy or adequacy of this news release, which has been prepared by management. There can be no assurance that any of the assumptions in the Preliminary Assessment will be supported by a Feasibility Study or will come to pass. Data is incomplete and considerable additional work will be required to complete further evaluation including but not limited to drilling, engineering and socioeconomic studies and investment. No firm quotes for costs have been received. The legal right to mine the project discussed has not been confirmed or applied for and the process for such application is new in South Africa and untested. The potential capital cost of the project is beyond the current means of the Company and there can be no assurance that financing for further work will be available.

Note to U.S. Investors: - Investors are urged to consider closely the disclosure in our Form 20F, File No. 0-30306, available at our office: Suite 328-550 Burrard Street, Vancouver BC, Canada, V6C 2B5 or from the SEC: 1(800) SEC-0330. The Company may access safe harbour rules.

The Preliminary Assessment on the initial resource area, called the Elandsfontein Project (or Project 1) has returned a pre-tax Net Present Value ("NPV") of R1.909billion (USD294 million at 6.5 Rand/USD or C$347 million at 5.5 Rand/C$) at a 5% discount rate resulting in an Internal Rate of Return ("IRR") of 18.9%. The figures are at approximately current metal prices and exchange rates as detailed in this report. At a 10% discount rate the NPV of the project is R796 million (USD122 million or C$145 million).

The preliminary mine plan in this study involved the sinking of twin shafts, each to 665 meters deep to access the better grade parts of the resources on the Merensky Reef and will concentrate about 251,000 ounces per year over the life of the shaft. The ounces receive post smelting for revenue purposes are about 218,000 ounces per year. The ounces are derived from a production rate of about 1,357,000 tonnes per year over the life of the project. The Merensky Reef is between 20cm and 1.2 meter thick layer.

This area is the source of most of the world's platinum production and is being mined at the adjacent BRPM platinum mine by Anglo Platinum. For comparison the BRPM platinum mine has resource grading 6.39g/t and the Western Bushveld Joint Venture is targeting the section of the resource at an in situ grade of about 9.67g/t (200cm g/t cut-off).

The estimated initial capital cost (Phase 1) is R1,428 billion (USD 220 million or C$259 million) with a payback period of 4.5 years from completion of construction.

There is sufficient information to warrant additional work and expenditure on the project. This additional work may result in there being sufficient information to warrant a pre-feasibility study. In order for this to be done in a scientific and timeous manner, firstly the resource needs to be up upgraded to an Indicated Resource, which may be achievable with the recommended drilling program. The second important information required is a preliminary confirmation of the metallurgical nature of the Merensky Reef prior to entering into the Pre-Feasibility stage. Additional monies for drilling and engineering work are recommended so as to reach a point of decision so as to advance to a definitive feasibility study.

SALIENT FEATURES – PRELIMINARY ASSESSEMENT

Updated Inferred Resource Base:

	Cut-off (cm g/t)	Million Tonnes	3PGM+Au Grade (g/t)	Channel Width (metre)	Diluted Channel Width (metre)	Tonnes PGM (3PGM+Au)	Ounce (Millions)
MR Domain 1	200	13.87	9.67	1.11	1.12	134.11	**4.312**
MR Domain 2	400	0.53	6.47	0.42	1.00	3.46	**0.111**
UG2 Domain 1	400	2.21	4.32	1.35	1.35	9.57	**0.310**
TOTAL	218	16.61	8.85	1.11	1.13	147.14	**4.723**

The cut-offs have been established by the QP after a review of iterative potential mine plans - 3PGM+Au = 4E = platinum, palladium, rhodium, gold

Capital and Working Costs

	Capital Cost in South African Rands	Capital Costs in Canadian Dollars	Period in Years over which the Capital is Spent
PTM Costs	R23.40m	C$4.25m	Partially Spent – remainder over 1.5 years
Owners Costs	R31.00m	C$5.64m	1.5 Year
Phase 1 Capital	**R1,428.90m**	**C$259.80m**	**4 Years**
Phase 2 Capital	R429.68m	C$78.12m	Further 5 Years
Working Capital	R71.70m	C$13.03m	Further 2 Years
Onsite Working Cost	R323.59/tonne	C$58.83/tonne	Average Life of Mine
Payback			**4.5 Years from completion of Construction**
Life Of Mine – Merensky Domain 1 Only			**14 Years of Production**

Production Profile at Steady State

Tonnes Milled (metric tonnes)	Head Grade after Mining Dilution – 3PGM+Au	Recovered Grade (g/t) to concentrate	Kilograms 3PGM+Au in concentrate	**Ounces 3PGM+Au in Concentrate**
1,357,000 tonnes/year	6.10 g/t to 6.86 g/t	5.13g/t to 6.01 g/t	7800 kg/year to 7900 kg/year	**250,775 oz/year to 253,990 oz/year**

Project Evaluation (Pre-Tax)

	South African Rands	Canadian Dollars	USA Dollars	IRR
NPV at 5%	**R1,908.960m**	**C$347m**	**USD294m**	**18.9%**
NPV at 10%	R795.618m	C$145m	USD122m	18.9%

QUALIFIED PERSONS

Independent Geological Qualified Person (“First QP”):

Mr Charles J. Muller (BSc (Hons) Pr Sci Nat (Reg. No. 400201/04)

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Independent Qualified Mining Engineer (“Second QP”):

Mr T Spindler (BSc Mining Engineering) (Reg. No. 880491)

Turnberry Projects 272 Kent Avenue,

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Independent Qualified Engineer (“Third QP”):

Mr Gordon Cunningham (B Eng (Chemical)) Pr. Eng. (Reg. No. 920082)

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Internal and NOT Independent Qualified Person (“Fourth QP”)

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ITEM 4 - INTRODUCTION AND TERMS OF REFERENCE

Platinum Group Metals Limited (“PTML”) and Platinum Group Metals RSA (Pty) Ltd (“PTM”) have successfully entered into a joint venture with Rustenburg Platinum Mines Limited (“RPM”), a subsidiary of Anglo Platinum Ltd (“AP”) and Africa Wide Mineral Prospecting and Exploration (Pty) Limited (“AW”). This joint venture (“WBJV”) agreement includes the properties Elandsfontein 102 JQ, Onderstepoort 98 JQ, Frischgewaagd 96 JQ and Koedoesfontein 94 JQ covering some 67 km^2 in area.

The areas that are reported on in this report have been subdivided in the areas as indicated below. The reason for the subdivision is that each area has a standalone licence and Environmental Management Program.

1. Elandsfontein (PTM)

2. Elandsfontein (RPM)

3. Onderstepoort 4, 5 and 6

4. Onderstepoort 3 and 8

5. Onderstepoort 14 and 15

6. Onderstepoort (RPM)

7. Frischgewaagd

8. Koedoesfontein

Item 4(a): Terms of Reference: This report is compiled in terms of the 43-101 (NI), the 43-101 CP and 43-101 CP (Proposed Amendments), 43-101F1 and the information and status of the project is disclosed in the manner prescribe by the Securities Commission. Specific reference is made to the following:

1. In Part 4 (4.2.8) of the 43-101 (NI) the company is obliged to file a technical report should there be a "material change" in the status of the company. A "material change" is as defined in 43-101 CP Part 2.4.

2. The Securities Commission allows for a Preliminary Assessment and this is defined under 43-101CP (Proposed Amendment) Part 1.7. The definition refers to a "Preliminary Assessment" and defines this as "… *an economic evaluation at an early stage* ….". It also refers to ".. *the information has a high degree of uncertainty*

…". The definition also refers to 43-101 (NI), Part 2.3 (3) (b) in which the Preliminary Assessment may be determined at an Inferred Resource confidence level. This being the case, the appropriate disclosures, warnings and cautionary measures are hereby emphasised.

Item 4(b): Purpose of the Report : The intentions of the report are to: -

Inform investors and shareholders of the progress of the project To make public the updated geological model for the project To make public and update the resource calculations for the project

To make public a Preliminary Assessment in terms of the 43-101 Regulations

Item 4(c): Source of Information:

The independent authors (First QP, Second QP and the Third QP) of this report are totally reliant on the information provided by PTM's representative and internal qualified person (Fourth QP). This information is derived from historical records for the area as well as information currently compiled by the operating company, which is PTM. The PTM generated information is under the control and care of WJ Visser, the Fourth QP, who is an employee of PTM and is not independent. The AP information pertaining to the deposit and their earlier resource calculations have been under their control and custody of AP and have been certified by AP personnel and have subsequently been supplied to the independent Qualified Persons. The independent Qualified Persons have visited the property of the WBJV and have undertaken prudent due diligence with respect to the data. Snowden Mining Consultants have previously completed an independent audit and sampling of the Elandsfontein (PTM) property which covers

Portions 12 and 14 of the farm Elandsfontein 102JQ, a small part of the resource area.

Item 4(d): Involvement of the QP:

All of the listed (First QP, Second QP and Third QP) independent qualified persons have in no manner any financial or preferential business relationships with PTM. Those independent qualified persons have a purely business relationship with the operating company and provide technical and scientific assistance when required and

requested by the company. The independent Qualified Persons all have other significant client lists and have no financial interest in PTM.

ITEM 5 - DISCLAIMER

In preparing this report the author/s relied upon:

1. PTM land title information for Elandsfontein 102 JQ and Frischgewaagd 96 JQ as provided by PTM.
2. Geological and assay information supplied by PTM and made available by AP. It is understood that AP's competent person has accepted responsibility for the integrity of historical data provided by AP.
3. Drill hole analytical and survey data compiled by PTM.
4. Information made available at the time of preparation.
5. The data supplied or obtain from sources outside of the company.
6. Assumptions, conditions, and qualifications set forth in this report.
7. The technical input and assumptions of First QP, Second QP, Third QP and Fourth QP regarding the metallurgical, mine design and basic in-house information, parameters, rates and assumptions. The areas of expertise are listed below.

Other than as disclosed herein the outside sources of information were relied upon without extensive inquiry and review. The authors had access to all information available and had the opportunity to visit the property and review the core. The author/s make no particular representation to the degree of accuracy of that information and do not bear liability thereto. A dataset was compiled from all available data supplied by AP as well as data added received during this assessment phase by either the Fourth QP or PTM geological personnel.

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for PTML by the First QP CJ Muller. CJ Muller has a geological and geostatistical background and has been involved in the evaluation of the precious metal deposits for over 20 years. The First QP (CJ Muller) has reported and made conclusions within this report with the sole purpose of the report being used by PTM subject to the terms and conditions of PTM's contract with the qualified persons and contributing qualified persons. The contract permits PTM to file this report as a Technical Report with Canadian Securities Regulatory Authorities or other regulators pursuant to provincial securities legislation or other legislation. Except for the purposes legislated under provincial securities laws or any other security laws any other use of this report by any third party is at that party's

sole risk.

Specific Areas of Responsibility are as follows: -

The First QP, CJ Muller: specifically contributed to Items 1 to 17, 19 to 24 and 26 of this report but accepts overall responsibility for the whole report. The First QP is however reliant on the information provided by the Fourth QP, WJ Visser who is the internal and not independent qualified person. CJ Muller has also relied upon the input of the Second and Third Qualified Persons in compiling this filing.

The Second QP, T Spindler and the Third QP, G Cunningham are responsible for Items 18 and 25 of this report and are totally reliant on the information provided by the First QP, CJ Muller in terms of evaluation and are totally reliant on the information provided by the Fourth QP, WJ Visser in terms of the additional information.

ITEM 6 - PROPERTY DESCRIPTION AND LOCATION

Item 6(a) and Item 6(b): Area and Extent and Location of Project:

The WBJV project is located on the south-western limb of the Bushveld Igneous Complex which is located some 50 km north-west of the North West Province town of Rustenburg (Diagram 1). The property adjoins AP's Bafokeng Rasimone Platinum Mine (BRPM) and Styldrift Project to the southeast and west respectively. The area of interest consists of farms Frischgewaagd 96 JQ, Portion 7 (a Portion of Portion 2), Portions 15 and 16 and Elandsfontein 102 JQ, Portion 12 of Elandsfontein 102 JQ, Mineral Area 2 (a Portion of Mineral Area 1) (Diagram 2) situated in the south-eastern corner of the larger joint venture area. Rustenburg is situated about 120km north-west of Johannesburg within the Republic of South Africa.

The total joint venture area includes PTM's properties Elandsfontein 102 JQ and Onderstepoort 98 JQ, but also certain portions of Elandsfontein 102 JQ, Frischgewaagd 96 JQ and Koedoesfontein 94 JQ contributed by Rustenburg Platinum Mines Limited ("RPM"), a wholly owned subsidiary of AP These properties are centred on Longitude 27 ^{o} 00' 00'' (E) and Latitude 25 ^{o} 20' 00'' (S) and the mineral rights cover an extent of approximately 67 km ^{2} or 6,700.000 Ha in extent.

Item 6(c): Licences:

Within the Western Bushveld Joint Venture Properties, there are seven separate licences and are specifically listed in the manner below to cross reference to the licence specifications. The licences over the WBJV area and are as follows:

1. Elandsfontein (PTM)

2. Elandsfontein (RPM)

3. Onderstepoort 4, 5 and 6

4. Onderstepoort 3 and 8

5. Onderstepoort 14 and 15

6. Onderstepoort (RPM)

7. Frischgewaagd

8. Koedoesfontein

Prospecting on **Elandsfontein (PTM)** Elandsfontein 102 JQ Portions 12 (a portion of portion 3) (a total area of 213.4714 Ha), Portion 14 (a total area of 83.4968 Ha) and Remaining Extent of Portion 1 (a total area of 67.6675 Ha) was originally carried out under the now expired Prospecting Permit No.PP269/2002 reference RDNW (KL) 5/2/2/4477. A new Prospecting Permit Application was submitted by PTM on 12 October 2003. The application is still being processed.

The prospecting permit over **Elandsfontein (RPM)** (Elandsfontein 102 JQ, Portions 8 (a Portion of Portion 1) (a total are of 35.3705 Ha) and RE9 (a total area of 403.9876 Ha)) was issued on 23 March 2004 and expires on 24 March 2006. The second permit number is PP 73/2002, Reference RDNW (KL) 5/2/2/4361. A prospecting permit number is PP 50/1996 and was issued on 11 March 2004 and has the reference RDNW (KL) 5/2/2/2305 and is valid until 10 March 2006. This permit covers the area Mineral Area 2 (a Portion of Mineral Area 1) (total area of 343.5627Ha) of the Farm Elandsfontein 102JQ.

The prospecting permit application over **Onderstepoort Portions 4, 5 and 6** (Onderstepoort 98JQ, Portion 4, a Portion of Portion 2 (a total area of 79.8273 Ha), Portion 5 (a Portion of Portion 2) (a total area of 51.7124 Ha) and Portion 6 (a portion of Portion 2) (a total area of 63.6567 Ha) was awarded on 30 April 2004 (Ref. No RDNW (KL) 5/2/24716, PP No.48/2004) and is valid until 30 April 2007.

A prospecting permit application over **Onderstepoort 3 and 8** (Onderstepoort 98JQ, Remaining Extent of Portion 3 (a total area of 274.3291 Ha) and Portion 8 (a Portion of Portion 1) (a total area of 177.8467 Ha), was issued on 24 March 2004, Prospecting Permit Number PP 26/2004 (Reference RDNW (KL) 5/2/2/4717) and is valid until 23 April 2006.

A Notarial New Order Prospecting Right for **Onderstepoort 14 and 15** (Onderstepoort 98JQ, now consolidated under Mimosa 81JQ, Portions 14 (a Portion of Portion 4) (total area of 245.2880 Ha) and Portion 15 (a Portion of Portion 5) (a total area of 183.6175 Ha) was awarded to PTM on 25 April 2005. The agreement was signed before the Notary Jacques Hattingh in Klerksdorp. The agreement is also held by J. Hattingh, as protocol number 7. The agreement is in force for a period of three years and terminates on 24 April 2008.

A new order prospecting right for **Onderstepoort (RPM)** (Onderstepoort Previous Portion 9) (a Portion of Portion 3) (127.2794 Ha) has been applied for. A new order prospecting right has also been applied for over Mineral Area No.1 (total area of 29.0101 Ha) of Ruston 97JQ, which is now consolidated under Mimosa 81 JQ. A permit application has also been applied for over Mineral Area No. 2 (total area of 38.6147 Ha) of the farm Ruston 97JQ which is also consolidated under Mimosa 81JQ. Both applications are awaiting the approval of Government.

A prospecting permit was issue to RPM over **Frischgewaagd** (Frischgewaagd 96JQ). A permit was also issue to RPM, Permit Number PP 294/2002 (Reference RDNW (KL) 5/2/2/4414) over the following areas: the portions of Frischgewaagd covered by PP 294/2002 include the following areas:

Portion RE4 (286.8951 Ha), Portion 3 (made up of Portion RE and Portion 13) (466.7884 Ha), Portion 2 (made of up Portion RE2 and Portion 7 (a Portion of Portion 2)) (616.3842 + 300.7757 Ha), Portion 15 (78.7091 Ha), Portion 16 (22.2698 Ha) and Portion 18 (45.0343 Ha).

The permit was valid until 16 October 2004. A conversion to a new order right was timeously applied for but the approval is still outstanding. The outstanding approval has no bearing on the validity of the application and prospecting continues under the old permit.

A prospecting permit was issued to RPM over **Koedoesfontein** 94JQ (2795.1294 Ha). The permit was issued on 19 March 2004 under Prospecting Number PP 70/2002 (Reference 5/2/2/4311) and is valid until 18 March 2006.

Item 6(d): Rights to Surface, Minerals and Agreements:

Regarding **Elandsfontein (PTM),** the dispute that was declared over the property has been settled by way of an Agreement of Settlement, which was signed on 26 April 2005. Party to this agreement was a Sale Agreement. The Agreement of Settlement has entitled PTM to the rights to the minerals as well as the freehold. The payment schedule is R1m within 10 days for signature, R0.5m within 60 days of signature, R2.2m within 90 days of signature and R3m by the 15 December 2005. All necessary payments to date have been made timeously.

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

The detailed terms of the Western Bushveld Joint Venture (which include Onderstepoort (RPM), Frischgewaagd and Koedoesfontein) were announced on October 27, 2004. The WBJV will immediately provide for a 26% Black Economic Empowerment interest in satisfaction of the 10-year target set by the Mining Charter and newly enacted Minerals Resources and Petroleum Development Act (2002). PTM and RPM will each own an initial 37% working interest in the interest in the farms and mineral rights contributed to the joint venture, while AW will own an initial 26% working interest. AW will work with local community groups in order to facilitate their inclusion in the economic benefits of the joint venture, primarily in areas such as equity, but will also include training, job creation and procurement to Historically Disadvantaged South Africans (HDSA's).

The WBJV structure and business plan is in compliance with South Africa's recently enacted minerals legislation, and will pursue platinum exploration and development on the combined mineral rights covering 67 square kilometres on the WBJV.

PTM is the operator of the WBJV and PTM undertook a due diligence on the data provided by RPM. The due diligence included the verification of the RPM declared resource of 3.7 million ounces platinum, palladium, rhodium and gold.

PTM has undertaken to incur cost of exploration to the amount of R35 Million over a 5 year period starting with the first 3 years at R5 Million increasing to R10 million a year for the last two, with the option to review yearly. The expenditure to date is in excess of the PTM's obligations to the joint venture agreement.

Item 6(e): Survey:

Elandsfontein (PTM) is registered with the deeds office (RSA) under Elandsfontein 102JQ, North West Province and measures 364.6357Ha. The farm can be located on the 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 are 27 ^{o} 05' 00'' (E) and 25 ^{o} 26' 00'' (S).

Onderstepoort (PTM) and Onderstepoort (RPM) are registered with the Deeds Office (RSA) under Onderstepoort JQ, Northern Province and measures 1,085.2700Ha. The farm can be located on the 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 co-ordinates (WGS84) are 27 ^{o} 02' 00'' (E) and 25 ^{o} 07' 00'' (S).

Frischgewaagd and Koedoesfontein: Frischgewaagd is registered with the Deeds Office (RSA) under Frischgewaagd 96, registration district JQ, Northern Province and measures 1,836.8574 Ha and Koedoesfontein that is registered with the Deeds Office (RSA) under Koedoesfontein 94, registration district JQ, Northern Province and measures 2,795.1294Ha. Both the farms can be located on the 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 co-ordinates (WGS84) are 27 ^{o} 02' 00'' (E) and 25 ^{o} 07' 00'' (S).

Item 6(f): Mineralised Zones:

The Bushveld Igneous Complex is well known for its large proportion of the world's platinum and palladium resources. There are two very different ore bodies within the Complex. The Merensky Reef ("MR"), the Upper Group 2 ("UG2") chromitite, which together can be traced on surface for 300 km in two separate arcs. The Northern Limb (Platreef) extends for over 120 km. Their global importance has justified several resource calculations in the past. Such historical data are compared with the information in recent mining company annual reports. Resource calculations tend to be larger by a considerable factor, because mining company reports include only proven and probable reserves, where sufficient information is available rigorously to justify such a classification. However, the remarkable continuity of layers within the Bushveld Igneous Complex certainly justifies qualitative extrapolation to adjacent areas, although current mines are probably exploiting the most favorable sections of reefs. The major platinum mining companies hold most of the mineral rights to these areas.

Historical estimates for all of the Bushveld bearing platinum and palladium reefs have been estimated at about 770 and 480 million ounces, respectively (down to a depth of 2 000 metre below surface). These estimates do not distinguish between the different categories of Proven and Probable Reserves and Inferred Resource. The present calculations indicate 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. Already mining is already taking place at 2 km in the Bushveld Igneous Complex, and so Inferred Resources, and ultimately mineable ore, could almost certainly be considered far greater than even these calculations suggest. These figures represent about 75 and 50 % of the world's platinum and palladium resources,

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respectively. These figures for proven and probable reserves in the Bushveld Igneous Complex alone are sufficient for the next 40

years at the current rate of production. However, estimated world resources are such as to permit extraction at an annually increasing rate of 6 % per annum for over 50 years. Expected sufficiency is less for palladium. Thereafter, down-dip extensions of existing Bushveld mines, 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.

Item 6(g): Liabilities and Payments:

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

Item 6(h): Environmental Liabilities:

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

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

1. Environmental management shall conform to the Environmental Management Programme ("EMP") as approved by the Department of Minerals and Energy (DME).
2. 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 Mineral Development.
3. Rehabilitation of the disturbed surface caused by prospecting activities will be rehabilitated to the standard as laid down in the EMP.
4. Financial provision in the form of a Rehabilitation Trust and/or Financial Guarantee.
5. A performance assessment, monitoring and evaluation report must be submitted annually.

Prospecting Permits are issued subject to the approval of the EMP which in turn is subject tot having provided a financial guarantee.

On **Elandsfontein (PTM)** the operator conducted exploration on Elandsfontein under an Environmental Management Program ("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 have been lodged with the Department of Minerals and Energy ("DME") in the name of PTM and are currently being processed. A follow up EMP was requested by the DME and was compiled by an independent consultant (Geovicon CC, Mike Bate) and was compiled on 23 August 2004. The updated EMP was accepted by the DME on 20 October 2004. The EMP financial guarantee submitted to cover this application is held by the Standard Bank of South Africa, Guarantee Number M410986 for the amount of R10,000.00. The Notarial Prospecting Agreement (Clause 10) requires that the Minister or authorised person have the right to inspect the performance of the company with respect to environmental matters.

With regards to the Onderstepoort area that was contributed by PTM, all the EMPs's were lodged with the DME and were approved on 30/04/2004 for **Onderstpoort 4,5 and 6** and on 24/04/2004 for **Onderstepoort 3 and 8** . Financial provision of R10,000.00 each for both optioned areas have been lodged with Standard Bank (Guarantee No. TRN M421363 for Onderstepoort 3 and 8 and No. TRN M421362 for Onderstepoort 4, 5 and 6 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 was compiled on 23 August 2004. The updated EMP was accepted by the DME on 20 October 2004. The EMP financial guarantee submitted to cover this application is held by the Standard Bank of South Africa, Guarantee Number M410986 for the amount of R10,000.00. The Notarial Prospecting Agreement (Clause 10) requires that the Minister of authorised person have the right to inspect the performance of the company with respect to environmental matters.

In the areas of the WBJV that were originally owned by RPM, PTM will take responsibility for the EMP's that originated from RPM over Elandsfontein, Onderstepoort, Frischgewaagd and Koedoesfontein. PTM, as operator of the joint venture, will be the custodian and will be responsible as operator for all aspects of the Environmental Programs and over all specifics as set out in the

different allocated and approved EMP's on all 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 over portions Mineral Area 2 (a Portion of Mineral Area 1) of the farm Elandsfontein 102JQ which has been dated 11 March 2004.

Regarding **Frischgewaagd** (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.

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

The EMP over **Koedoesfontein** is dated as having been received by the Department of Minerals and Energy on 22 September 2002.

ITEM 7 - ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Item 7(a): Topography, Elevation and Vegetation:

The WBJV properties are located on a central plateau characterized by extensive savannah, with vegetation consisting of grasses and shrubs with few trees.

For the **Elandsfontein (PTM)** and **Elandsfontein (RPM)** property the total elevation relief is greater since prominent hills occur in this portion of the property. Variations in topographical relief are minor and limited to low gently sloped hills. Elevations range from 1080 to 1156m with an average of 1100m on the Elandsfontein and neighbouring properties. The **Elandsfontein (PTM) and Elandsfontein (RPM)** project area is located on the southwestern limb of the Bushveld Complex, some 26 km west of the North West Province town of Rustenburg. The WBJV adjoins the Bafokeng Rasimone Platinum Mine ("BRPM"), which lies to the south-east. The town of Boshoek is situated 10 km to the south of the project area along the tar road linking the town of Rustenburg with Sun City and crosses the project area (Diagram 3a and 3b). A railway line linking BRPM to the national network passes the project area immediately to the east with a railway siding at Boshoek.

The climatic conditions (information provided by the South African Weather Bureau) for the area in which the project is situated, is typical of the northern part of the North West Province. Summer day temperatures are warm to hot, and the winter months are moderate to cool with temperatures rarely dropping below 0℃. The area is considered semi arid, with a n annual rainfall of 520 mm. The rainy season falls over the summer months of October through April and the highest rainfall occurring during December and January. The highest rainfall ever recorded in any 24-hour was 65mm. Wind conditions are relatively calm. The prevailing wind direction is north-north-west and wind speeds average 2.5 m/s.

This area is classified as Mixed Vegetation and is typically composed of grass between low trees and shrubs. 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 open tree savanna. 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 the herbaceous layer is dominated by grasses such as Fingergrass Digitaria eriantha, Kalahari Sand Quick Schmidtia pappophoroides, Wool Grass Anthephora pubescens, Stipagrostis uniplumis, and various Aristida and Eragrostis species. On deeper and more sandy soils, Silver Clusterleaf Terminalia sericea becomes dominant, with Peeling Plane Ochna pulchra, Wild Raisin Grewia flava, Peltophorum africanum and Burkea africana often prominent woody species, while Broom Grass Eragrostis pallens and Purple Spike Cats' tail Perotis patens are

characteristically present in the scanty grass sward.

On the **Onderstepoort (PTM)** and **Onderstepoort (RPM)** properties the site elevation is approximately 1050m. The highest point is 1105 m. No major roads or township developments exist on the property. Only one minor water dam occurs on the property. The northern boundary of the property is formed by the Elands River which is a perennial steam draining to the northeast. Minor drainage into the Elands River is from south to north on the area of concern. The main soils are moderate to deep, black and red clay soils, with thin sandy loam soils to the east. The Limpopo Province is generally characterised by limited high potential agricultural soil. The erodibility index is 5 (high). The average sub-catchment sediment yield is 83 x 10^{3} tonnes per annum.

Drainage of the streams is towards the north-east and joins into the Elands River, which forms the northern boundary of the area under concern. The farm lies in Quaternary sub-catchments A22F, the Elands River sub-catchments of the Limpopo drainage region.

This area is classified as Mixed Bushveld vegetation and is typically dominated by with grass between low trees and shrubs. 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 rather open tree savanna. 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 the herbaceous layer is dominated by grasses such as Fingergrass Digitaria eriantha, Kalahari Sand Quick Schmidtia pappophoroides, Wool Grass Anthephora pubescens, Stipagrostis uniplumis, and various Aristida and Eragrostis species. On deeper and more sandy soils, Silver Clusterleaf Terminalia sericea becomes dominant, with Peeling Plane Ochna pulchra, Wild Raisin Grewia flava, Peltophorum africanum and Burkea africana often prominent woody species, while Broom Grass Eragrostis pallens and Purple Spike Cats’ tail Perotis patens are characteristically present in the scanty grass sward.

The typical animal life of the Bushveld has largely disappeared from the area due to farming and hunting. Efforts are 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. The Southern Greater Kudu found here are amongst the biggest in the country. On the area in question it is expected that larger buck 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. Monitor lizards, snakes and geckos are present, and the most characteristic birds include lilac breasted rollers, African hoopoes and owls.

Frischgewaagd and Koedoesfontein areas are classified as mixed bushveld vegetation and are typically made up of grass between low trees and shrubs. 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 rather open tree savanna. 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. In these areas the grazing is sweet, and the herbaceous layer is dominated by grasses such as Fingergrass Digitaria eriantha, Kalahari Sand Quick Schmidtia pappophoroides, Wool Grass Anthephora pubescens, Stipagrostis uniplumis, and various Aristida and Eragrostis species. On deeper and more sandy soils, Silver Clusterleaf Terminalia sericea becomes dominant, with Peeling Plane Ochna pulchra, Wild Raisin Grewia flava, Peltophorum africanum and Burkea africana often prominent woody species, while Broom Grass Eragrostis pallens and Purple Spike Cats'tail Perotis patens are characteristically present in the scanty grass sward.

The climate is temperate with low rainfall and high summer temperatures, resulting in a semi-arid environment.

Item 7 (b): Means of Access to the Property:

South Africa has a very large well-established mining industry in which the project is located. As a result of the mining activity (amongst others) the infrastructure is well established with abundant well-maintained highways and roads as well as electricity distribution networks and telephone systems. The Elandsfontein, Onderstepoort, Frischgewaagd and Koedoesfontein Properties are easily accessible from Johannesburg by travelling 120 kilometres northwest on the Regional Road 24 to the town of

Rustenburg and then a further 35 kilometres to the properties. Numerous gravel roads cross both properties, which provides for easy access. The resort of Sun City is located approximately 10 km north of the Elandsfontein Property (Refer to Diagram 2). The Elandsfontein property borders the AP's managed Bafokeng-Rasimone Platinum Mine which lies to the south east of the property as well as the Styldrift Joint Venture (joint venture between the Royal Bafokeng Nation and AP) which lies directly to the east of the property which is also serviced by modern access roads and services.

Item 7(c): Population Centres and Nature of the Transport:

The major population centre is the town of Rustenburg, which lies about 35km directly to the south of the project. Pretoria is approximately 100km to the East and Johannesburg lies about 120km to the south-east. A popular and unusually large hotel and entertainment centre (Sun City) lies about 10km to the north of the project. A paved highway Rustenburg crosses the property. Access across most of the property can be achieved by truck without significant road building.

Item 7(d): Climate:

The climate is mild throughout the year and can be classified as semi-arid. South Africa has summer from November to April. South African winter season is from May to October. In summer the days are hot and generally sunny in the morning, with afternoon showers or thunderstorms. Daytime temperatures can rise to 38 ^{o} C (100 ^{o} F) and night temperatures drop to around 15 ^{o} C (68-77 ^{o} F). The afternoons can be humid. In winter, days are dry, sunny and cool to warm, while evening temperatures drop sharply. Daytime temperatures generally reach 20 ^{o} C (68 ^{o} F) and can drop to as low as 5 ^{o} C (41 ^{o} F) at night.

Tabulated below is a guide to monthly averages for temperature, sunshine and rainfall for the region. (Reported within the submitted Environmental Management Program which was submitted in conjunction with the Prospecting Permit application: Investigation conducted by DWA, a contractor trading under the name of Digby Wells and Associates, Environmental Solutions Provider).

Monthly averages	J	F	M	A	M	J	J	A	S	O	N	D
TEMP (0 ^{o} C)	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

Item 7(e): Infrastructure with respect to Mining:

This report details the exploration programs and a preliminary assessment. At this stage it is sufficient to note that all areas are close to major towns with paved roads being the norm. Power lines actually cross both project areas and water resources are generally derived from wells, which are close to the local towns and villages. As several platinum mines are located within 50km of the property there is excellent access to materials and skilled labour. One of the smelter complexes of AP is located within 60 kilometres of the property.

ITEM 8 - HISTORY

Item 8(a): Prior Ownership :

Elandsfontein (PTM), Onderstepoort 4, 5 and 6, Onderstepoort 3 and 8, Onderstepoort 14 and 15 were all in the hands of private owners. All previous work done on these properties has not been researched and is generally unpublished. There has been a limited amount of academic type work done over these properties by the Council of Geosciences (Government Agency) but is generally not of an economic nature.

Onderstepoort (RPM), Elandsfontein (RPM), Frischgewaagd and Koedoesfontein have generally been in the hands of the major mining groups resident in the Republic of South Africa. Portions of Frischgewaagd were held by Impala Platinum Mines Limited but were subsequently acquired by Johannesburg Consolidated Investment Company Limited, who subsequently was acquired by AP through RPM.

Item 8(b): Work done by Previous Owners:

Previous geological exploration and resource estimation assessments were done by AP who is the owner of mineral rights to the area of interest. AP managed the exploration-drilling program for the ELN and FG borehole series in the area of interest (23 boreholes in total). Geological and sampling logs and an assay database are available for this work.

Regional gravity and ground magnetic surveys were available to interpret the regional and local geological setting of the reefs. A distinct increase of gravity values occurs from south-west to northwest, most probably reflecting the thickening of the Bushveld sequence in that direction. The low gravity trends south-east north-west. The magnetic survey reflects the magnetite rich Main Zone and some large displacements and intrusives in the area.

Item 8(c): Historical Reserves and Resources:

Previous reserves and resources quoted for the area, and derived from are those published in the AP 2004 Annual Report including 7.8Mt grading 5.88 g/t 3PGM+Au on the Merensky Reef and 4.8Mt grading 4.42 g/t 3PGM+Au on the UG2 Reef. This is reported for their 37% interest (equal to PTM's as the WBJV was completed at that time). As to a 100% interest in the property this would result in an estimate of 21.1 Mt grading 5.88 g/t 3PGM+Au on the Merensky and 13.0 Mt grading 4.42 g/t

3PGM+Au on the UG2 reef. The resources of AP are reported as subject to a satisfactory independent audit.

Item 8(d): Production from Property:

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

ITEM 9 - GEOLOGICAL SETTING

The WBJV adjoins the Anglo Platinum's Bafokeng Rasimone Platinum Mine (BRPM), which lies to the south and the Styldrift Project, which lies to the east. All these projects lie within the southwestern limb of the Bushveld Complex and comprises the stratigraphic units of the Rustenburg Layered Suite. This sequence comprises mostly gabbros, norites, anorthosites and pyroxenites. There are two potentially economically viable platinum-bearing horizons in this area, namely the UG2 Reef that is a chromite seam and the Merensky Reef, occurring as a feldspathic pegmatoidal pyroxenite, or a hartburgite or a coarse grained pyroxenite.

The Merensky Reef and UG2 Reefs sub-outcrops beneath a relatively thick (+/- 2 m) layer of black turf overburden. The entire sequence strikes north-northwest to south-southeast and dips 6 ^{o} north-easterly (in this area specifically) towards the centre of the Bushveld Igneous Complex.

The Bushveld Igneous Complex sequence, specifically the lower portion of the Main Zone and the Critical Zone (HW1 -5 and Bastard reef to FW 6), thins dramatically towards the west with the result that the lithological units/marker horizons and the potentially economic reefs pinch out. A further complication would be the increased presence of iron-replaced pegmatoidal bodies towards the south of the area of interest.

Stratigraphy: The general stratigraphy of the western Bushveld is depicted in Diagram 4a. The detailed stratigraphy as encountered at BRPM is depicted in Diagram 4b and Impala Platinum in Diagram 4c. The identifiable and correlatable units within the WBJV area are the base of the noritic rich Main Zone, the anorthositic hanging wall sequence (HW 1 -5), the Bastard Reef pyroxenite -MID 1 to 3 (noritic at base to anortositic at the top) -Merensky Reef pyroxenite, the anorthositic footwall FW 6/Lone Chrome unit and the FW 12 anorthosite unit overlying the UG2, the UG2 or a shear zone and the Alteration Zone, represented by an even grained medium crystalline norite. The basal alteration zone is not normally representative of the Bushveld sequence at BRPM and would seem to be a chill zone in contact with the Transvaal Supergroup sediments. This lower part of the stratigraphy has been positively identified on the Elandsfontein Project, which adjoins the property immediately to the south and forms an extension of the reefs. A similar setting is envisaged for the Merensky Reef.

The Main and Critical Zones of the Bushveld Igneous Complex sequence as intersected in the WBJV boreholes on Western Limb (Refer to Tables 1a and 1b and Diagram 4a) consist of norites and gabbro-norites within the Main Zone (< 60 m) at the top of the sequence. Spotted and mottled anorthositic hanging wall units (HW 1 -5) (<40 m) overlying the Bastard pyroxenite (<2 m), which are followed by norite to mottled anorthosite. The MID 1 - 3 units (<6 m) overlie the Merensky Reef pyroxenite (<2 m). The Merensky Reef can either be a thin (<10 cm) feldspathic pegmatoidal pyroxenite and/or a millimetre thick chromitite layer and/or a contact only and/or a thicker (>100cm) type reef consisting of harzburgite and/or pegmatoidal pyroxenite units. Some of the noritic footwall units (FW 1 -5) in the immediate foot of the reef are not always developed and is in total much thinner (<13 m) than at BRPM setting to the south-east. The mottled anorthosite footwall unit, FW 6 (<2 m) with a thin (millimetres thick) chromitite layer, the so-called Lone Chrome layer, although thinner (within the Feldspathic Pegmatoidal Pyroxenite reef type area) is generally developed in this area and constitute a critical marker horizon. Footwall units, FW 7 to 11 (mostly noritic) are also not always developed and much thinner (<25 m) than at BRPM. The mottled anorthosite footwall unit, FW 12 is generally developed (<2 m) overlying a very thin UG2 chromitite/pyroxenite towards the northeastern corner of the property. Shearing may have occurred on the UG2 plane with the result that the UG2 reef is not properly developed. The chromitite layer is either very thin or the unit is pyroxenitic. The lower portion of the sequence has been attenuated with a sheared unit (incorporating the lower portion of the Critical Zone) followed by a medium crystalline noritic sequence. The alteration zone or chill zone is not part of the normal Bushveld Igneous Complex sequence and has developed in contact with the Transvaal Supergroup sediments.

Further to the east the Bushveld Igneous Complex stratigraphic sequence is more "normal" with the complete stratigraphy developed and the stratigraphic sequences thicker and better developed. A dramatic thickening of the sequence (HW 1 -5 to the Lone Chrome marker (FW 6)) occurs to the east of boreholes FG 30 (FG 07) and ELN 12. This thickening of the stratigraphic units trends more or less north-west south-east and may be the consequence of a general thickening of the entire Bushveld Igneous Complex as the complex is developed further away from the edge (and in contact with) the Transvaal Supergroup.

Correlation and Lateral Continuity of the Reefs: The upper noritic portion of the Main Zone could be identified and correlated with confidence. The contact with the anorthositic Hanging Wall sequence (HW 1 to 5) has been taken as a marker horizon. The Hanging Wall sequence (HW 1 to 5) thins significantly from east to west within the project area. Due to the thinning of the Critical Zone only FW 6 (mottled anorthosite with thin chromitite stringer at base (the so-called Lone Chrome)) and FW 12 (mottled anorthosite unit immediately overlying the UG2 horizon) as well as the Bastard Reef pyroxenite to Merensky Reef (separated by

the noritic, leuco-noritic to anorthositic MID 1 to 3 sequence (or part of)) could be identified with confidence. The sequence has been affected by iron-replacement, especially the pyroxenites, towards the west of the property.

The Merensky Reef was positively identified (and used in the resource estimation) in 39 intersections and the intersection depths are summarized in Table 1a. Only the reef intersections that had no faulting or disturbances were used in the resource estimate.

Resource estimation is not possible based on diamond drilling information within 50m from surface due to excessive core loss encountered, reef identification/correlation problems and thinning of the reefs towards the west.

Merensky Reef: Four types of Merensky reef have been identified in the area of interest viz.

1. Hartzburgite-type Reef ("Htz")

2. Feldpathic Pegmatoidal Pyroxenite-type Reef ("FPP")

3. Pyroxenite-type Reef ("Pxnt")

4. Contact-type Reef ("CR")

Further to the south-west no reef is developed since the reef has either outcropped or abutted against the shear zone or Transvaal Supergroup.

The Htz-type reef is developed to the north-east of the area of interest with the FPP-type reef towards the south-west (Diagram 8a). The Htz-type reef consists of interlayered harzburgite and pegmatoidal pyroxenite units and is in general thicker (47 to 224 cm) and of higher grade (6.86 to 16.99 ppm (3PGM+Au)) in relation to the FPP- type reef (60 to 91 cm, 4.35 to 7.50 ppm (3PGM+Au), as well as grade occurring in hanging wall pyroxenite). Reef development and grades are highly variable in the Feldspathic Pegmatoidal Pyroxenite-type reef area.

Structural Discontinuities: Potholes are not clearly discernable from the borehole data. To determine the existence of potholes on the property the possibility exist that pothole edges could be associated with the Contact Reef. Duplicated reef intersections could also represent pothole edge effects ("goose-necking"). Pseudo-reefs along the pothole edges and associated with goose-necking may be interpreted within the project area.

Faulting: Significant faulting has only been observed in borehole WBJV04. From the magnetic surveys some faulting can be

inferred. Fault losses have not been taken into account in the resource estimation and an expected loss of 30% has been used to accommodate geological, rock engineering and other unexpected losses of mineable ground.

Dykes: Only thin dolerite intrusives were intersected in some of the boreholes and are generally between 0.5m and 2m thick. An east-west trending intrusive is evident on the magnetic image. (Refer to Diagrams 5a and 5b) and is a reliable source of information for the determination of dykes within the area of interest.

Shear Zones: A shear zone along the Alteration Zone eliminating stratigraphy progressively from the UG 2 horizon to the Main Zone from east to west has serious consequences for the economic units. The elimination effect of the shear zone is restricted to about 200m from the outcrop and sub-outcrop lineation.

Replacement Pegmatites: Reef packages to the south in the Elandsfontein (PTM) area are marginally affected (Siepker and Muller, 2004) and this should be taken into consideration in the resource estimation and geological loss figures within the Feldspathic Pegmatoidal Pyroxenite reef type area.

Depth of Oxidation and Overburden: Weathering affects the reef horizons to a depth of 50m of surface since the pyroxenites are the most affected. The outcrop trends north-west to south-east.

Geological and Rock Engineering Related Losses: Industry standards for geological and rock engineering related losses are in the order of 30% for platinum mines and projects in the Bushveld Igneous Complex. Losses in this area within the Feldsphatic pegmatoidal pyroxenite reef area though could be as high as 40% due to the influence of replacement bodies, faulting, presence of contact reef type (highly variable grade) and the possibility of potholing. The industry average of 30% losses has been applied to the resource estimates.

Structural Model: A structural model was constructed from the geophysical information and the borehole intersections (Also refer to Diagram 5a, 5b, 6 and 7). In general three phases of faulting have been recognised in the area of interest. The older of the structural features are the NNW to SSE trending faulting, which appear to have a consistence down throw to the north-east. The second phase of structural deformation appears to be N-S trending faulting, which may have a wrench component. A possible final phase of deformation is possibly related to the E-W, dyke intruded structural weaknesses.

ITEM 10 - DEPOSIT TYPE

The project area forms part of the Western Limb of the Bushveld Igneous Complex. PGM mineralisation is hosted within the UG2 Reef and the Merensky Reef located within the Upper Critical Zone of the Rustenburg Layered Suite of the Bushveld Complex (Refer to Tables 1a, 1b and 2). The property is situated immediately north-west of and adjoining the Bafokeng Rasimone Platinum Mine and west of Anglo Platinum's Styldrift project. The geology of the BRPM mine is relatively well understood and is expected, in certain aspects, to be representative of the WBJV area.

The Merensky Reef in this area consists of four distinct reef types viz. Harzburgitic-type reef (interlayered harzburgite and feldspathic pegmatoidal pyroxenite units, tens of centimetres thick) developed towards the north-east and feldspathic pegmatoidal pyroxenite type reef occurring to the south-west with reef development deteriorating towards the west, abutting against a shear zone or in contact with the Transvaal Supergroup. Contact Reef can be found within any of the facies mentioned above. The UG2 Reef is well developed in the north-eat of the property but deteriorates towards the south-west of the property. In this area the UG2 develops into a thin chromitite layer and/or pyroxenitic unit only. The UG2 Reef in this area may also be assumed into the

shear zone along the alteration zone. The Merensky Reef outcrop (predominantly sub-outcrops a few metres below the black turf) has an approximate 800 m strike length, which runs roughly north-west south-east on the property. The Merensky Reef and the UG 2 (or shear zone) are separated by approximately 10 to 60 m (from south-west to the north-east) and dips approximately 6 ^{o} to the north-east.

ITEM 11 - MINERALISATION

Mineralisation Styles and Distribution: PGM mineralisation in the western Bushveld Igneous Complex is hosted within the Merensky Reef and is generally a 10cm to 120cm thick pegmatoidal pyroxenite unit and may be associated with thin chromitite layers. The UG2 chromitite layer is on the average a 60cm and up to 200cm thick unit of economic interest.

The Merensky Reef at BRPM consists of different reef types ("facies") such as contact, pegmatoidal pyroxenite, harzburgitic (Refer to Diagrams 8a and 8b). In general contact type reef represent waste on footwall contact, pegmatoidal pyroxenite reef is on average 10 cm thick with thin chromitite layers at the base and sometimes at the top with the harzburgitic type reef in general thicker, in the order of 40 cm. PGM mineralisation differs in association with these reef types. In general PGM mineralisation is low where pyroxenite is in direct contact with the footwall, high but variable grades are associated with pegmatoidal pyroxenite type reef and generally high and more uniform in association with Harzburgitic type reef.

ITEM 12 - EXPLORATION

Item 12(a): Survey (Field Observations) Results, Procedures and Parameters:

Fieldwork done to date was firstly completed on the Farm Onderstepoort where various aspects of the Lower Critical Zone, intrusive ultramafic bodies and structural features were identified. This information has contributed indirectly to the economic feasibility of the overall project, but the main focus of attention has been the Elandsfontein Project.

Geophysical information was obtained from AP and is shown in Diagram 5a and 5b. This information has been particularly useful in the estimation of major structural features as well has the typical Bushveld Igneous Complex layering.

Item 12(b): Interpretation of Survey (Field Observations) Results:

The structural features identified in the geomagnetic data have been interpreted in terms of a regional structural model and are shown in Diagram 7. Particularly noticeable is the evidence of the major dyke features. The first feature that can be seen is a major east-west feature running through the northern portion of the Elandsfontein Project. The second dyke is a north-north-west to south- southeast feature which runs through the east of the Elandsfontein Project.

Other major structural features include a structural disturbance, which has a north-south orientation and runs through the Elandsfontein Project. The major disturbance has been intersected in borehole WBJV04. In this borehole the structural feature is evident as an ultramafic altered sheared zone.

Other less prominent features shown up in the geophysical information include a step down faulted area within the centre of the Elandsfontein Project. A triplet of down step faults has a north-west to south-east orientation and the throw of the faults is consistently down to the north-east. These features have also been determined in the drilling of the project.

Other important geophysical information also made available by AP is satellite-enhanced imagery. The major feature evident in

this imagery is the presence of the Main Zone (weathered to a black clay-rich soil horizon which is indicated in a purple colour on the satellite image). Taking this information and read in conjunction with the geomagnetic interpretation, the presence of the Main Zone from a soil profile point of view is clearly evident.

Item 12(c): Persons responsible for Survey (Field Observations) Data Collection and Compilation:

The person responsible for the interpretation of the geophysical and satellite imagery has been supplied by AP, and assistance with the interpretation has been given by AP. Willie Visser (Fourth internal QP) has been responsible for the interpretation and modelling of the information. All other field data has been collected, collated and compiled by PTM personnel under the guidance and supervision of the Fourth QP.

Item 12 (d): Reliability of the Survey (Field Observations) Data:

PTM's qualified geologist, A Valigy, conducted the fieldwork done by PTM on the Onderstepoort properties. This work was done under the supervision and control of the Fourth QP, W Visser.

ITEM 13 - DRILLING

Type and Extent of Drilling:

The type of drilling that is being conducted on the WBJV is a diamond drilling, core recovery technique. The drilling involves a BQ size of solid core extraction. The drilling is being placed on an unbiased 500m by 500m grid. The grid has been extended to include the whole of the project area. Depending on the overall results of the project, further drilling may be necessary, and in which case the drill pattern will be altered to improve on the quality of the resource/reserve.

Procedures, Summary and Interpretation of Results:

The results of the drilling and the general geological interpretation are digitally captured in a GIS software package trading under the name of 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 manually drawn through the deposit. This information assists in the interpretation of the sequence of the stratigraphy intersected as well as verifying the information gathered as well as assisting in the placing of additional and boreholes.

Comment on True and Apparent Widths of the Mineralised Zones:

The overall geometry of the deposit has been clearly defined in the sections drawn through the property. On the average the dip of the reef does not exceed six degrees. All the diamond drill holes that have been drilled on the property are vertical holes and the drill holes surveys are virtually vertical. Positions from the holes are included in Diagram 9. Given the dip of the reef at no more than six degrees, and given that the holes do not deviate from the vertical, the variance between the apparent width and true width does not exceed 2%.

Comment on the Orientation of the Mineralised Zones:

The mineralised zones of the Elandsfontein Project include the Merensky Reef and the UG2 Reef. Both these reefs are planar tabular ultramafic precipitants of a differentiated magma and therefore form a continuous sheet-like accumulate. The stratigrahpic markers both above and below the economic horizons are unquestionably recognizable and emphasise the recognition of the Merensky Reef and the UG2 Reef.

There are a few exceptions to the quality of recognition of the stratigraphic sequences. These disturbances are generally of a structural nature and are expected within this type of deposit. The holes in which there is no clear and decisive stratigraphic recognition include WBJV 04.

ITEM 14 - SAMPLING METHOD AND APPROACH

Item 14(a): Description of the Sampling Method, Details of the Location, Number and Type of Sampling, Size of the Sampling and the Size of the Area Covered in the Sampling Exercise:

The sampling described relates to sampling of diamond drill core. Firstly the core is marked for distance below collar and for major stratigraphic units. Once the stratigraphic units are identified then the economic units are marked. The economic units in this project include the Merensky Reef and the UG2. The top and bottom contacts of the reef are clearly marked on the core. The name of the economic units is clearly marked on the core. Thereafter the core is rotated in a manner that all lineations pertaining to stratification are aligned to produce a representative split down the core. A centre cut line is then drawn for cutting and thereafter replacing in the core trays. The sample intervals are then marked as a line and a distance from collar. The sample intervals are typically 20cm to 25cm in length. The sample intervals are then allocated a sampling number. The number is also written on the core for reference purposes. The half core is then removed and place into standard high quality plastic bags together with the sampling tag. The responsible geologist then seals the bag. The sampling information is recorded on a specially designed sampling sheet enabling easy and accurate digital capture. The sampling extends for about a meter into the hanging wall and footwall of the economic reefs.

Item 14(b): Description of the Drilling Recovery Performance and the Effect on Sampling Bias:

All reef intersections that are sampled require a 100% core recovery. This is required by the drilling company, and if 100% is not recovered the drilling company will re-drill using a wedge to achieve the desired recovery.

Item 14(c): Description of the Sampling Quality, Suitability of the Sampling and the Sampling Bias:

The quality of the sampling is monitored and supervised by a qualified geologist. The methodology is in accordance with the company standards. The sampling is done in a manner that includes the entire economic unit together with hangingwall and footwall sampling. By rotating the core in a manner that the stratification is vertical and by inserting a cut line down the centre of the core, and by only removing one side of the core, the sampling bias is reduced.

ITEM 15 - SAMPLE PREPARATION, ANALYSIS, SECURITY AND DATA VERIFICATION

Item 15(a): Description of the Sampling Methodology, QA/QC, Chain of Custody, Sampling Processing, Sampling Reduction and Security:

Samples are subject to a chain of custody, which is tracked at all times. Samples are not removed from their secured storage location without a chain of custody documentation being completed to track the movement of the samples and persons responsible for the security of the samples during the movement. Ultimate responsibility for the safe 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 his written permission.

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

1. Samples are sequenced within their secure storage area and the sample sequences examined to determine if any samples are out of order or missing.
2. The sample sequences and numbers shipped are recorded both on the chain of custody form and on the analytical request form.
3. The samples are then placed, in sequential order, into securable shipping containers (the numbers of the samples enclosed on the outside of the container with, the shipment, waybill or order number and the number of containers included in the shipment (e.g. J88899 -J88999, OR04-2, Box 1 of 12).
4. 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. A copy of the analytical request form and Chain of Custody kept on site by the Project Geologist.
5. Once the above is completed and the sample shipping containers sealed the samples may be removed from the secured area. The method by which the sample shipment containers 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 transportation process 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 safe transportation of the samples to the analytical facility and to insure that the samples are, if necessary, outside the custody of PTM contractors or personnel for as little time as possible. Under ideal conditions personnel employed by PTM transport the samples to the analytical facility. In all cases the original chain of custody letter accompanies the samples to their final destination.

The Supervising Geologist ensures that the analytical facility is aware of the PTM standards and requirements. The analytical facility accepts the responsibility for inspecting for any evidence or possible contamination or tampering of the shipment that it has received from PTM. A photocopy of the chain of custody letter, signed and dated by an official of the analytical facility, is be faxed to PTM's offices in Johannesburg upon receipt of the samples by the analytical facility and the original signed letter is be returned to PTM along with the signed analytical certificate/s.

If the analytical facility suspects the sample shipment has been tampered with they have instructions to contact the Supervising Geologist immediately who will make arrangements to have someone in the employ of PTM examine the sample shipment and confirm it's integrity prior to the initiation 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 responsible geologist immediately notifies PTM and PTM Management of any concerns in writing and decides 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 is not suitable as the basis for a outside release until it's validity is proven via additional sampling, Quality control checks and examination.

Should evidence or suspicions of tampering or contamination of the sampling be uncovered, PTM will immediately commence with a complete security review of the operating procedure. An independent third party with the report to be delivered directly and solely to the directors of PTM for their consideration and drafting of an action plan would conduct the investigation. All in-country exploration activities will be immediately suspended until this review is complete and has been reviewed by the directors of the company and acted upon.

Item 15(b): Laboratory Particulars and Procedures, Laboratory Standards and Certification:

Three laboratories have been used to date: Anglo American Analytical Laboratories - AARL (South Africa), Genalysis (Australia) and currently Setpoint Laboratories (South Africa). Sample preparation is done by the Setpoint Laboratories. Samples are received, verified, checked for moisture and dried (if necessary). The samples are then weighed and results reported. A Jaw Crusher then crushes the samples, after which either Roller splitting or Riffler splitting splits them. Then the samples are milled to 90% < 75 um, per 2 kg unit, utilising an LM5 pulverisor. The excess sampling material is packaged dispatched back to the PTM.

Samples were analyzed for Au (ppb), Pt (ppb), Pd (ppb) and Rh (ppb) by standard 25g Lead fire assay with an ICP-MS (Inductively Coupled Plasma Mass Spectrometry) finish and for base metal elements by multi (four) acid digestion in Teflon test tubes and AAS (Flame Atomic Absorption Spectrometry) for Cu (ppm), Ni (ppm), Co (ppm) and Cr (ppm). The samples were assayed at Genalysis Laboratories Services in Perth Australia or AARL in Johannesburg (RSA) or Setpoint Laboratories, Johannesburg, RSA.

Blanks - The insertion of blanks provides an important check on the laboratory practices and the baseline calibration of laboratory instrumentation. Blanks consist of one half or one quarter drill core collected from a known interval devoid of Pt, Pd, Cu and Ni mineralisation. Typically this will be a basement or cover lithology previously tested. The blank being used is always noted to track its behaviour and trace metal content. Typically the first blank is sample number five in a given sampling sequence.

Duplicates - The insertion of duplicates track the reproducibility of sample results. Typically quartered core is submitted for both samples. The two samples receive sequential numbers. Notation is made in the log as to which sample is being duplicated. Typically the first sample duplicated is sample number ten in a sampling sequence.

Standards - Certified reference standards are inserted into the sampling sequence to check the accuracy of the analytical results. Generally the standards are inserted in place of the fifteenth sample in the sample sequence. The standard used is recorded in the drill log but there is never any obvious indication to the lab of which standard has been inserted. Standards are supplied by the company and as they are the sole method of tracking the accuracy of the analytical data they are be stored in sealed containers and considerable care is be taken to ensure they are not contaminated in any manner (i.e. stored in dusty environment, placed in less than pristine sample bag or sprayed/dusted by core saw contamination).

Monitoring the quality control of the analytical data is the responsibility of the Supervising Geologist.

Item 15(c): QA/QC Results and Comments:

The sampling procedure, diligence and accuracy are acceptable for the type of deposit being sampled. The insertion of standards and blanks and the monitoring of the standards and blanks against the certified standards is being carried out. The failure rate and monitoring is in accordance of the company's procedures and found to be acceptable.

The results of the QA/QC can be found on Chart 1 (CDN PGMS-5) and Chart 2 (CDN PGMS-6) of which there are failures within sampling but unrelated to the reef horizon. In particular there are NO failures within the reef horizons.

Item 15(d): Comments on the Sampling Adequacy, Sample Preparation, Security and Analytical Procedures:

The sampling preparation, security and analytical procedures are of a high standard and are of an acceptable industrial, commercial and scientific nature.

ITEM 16 - DATA VERIFICATION

Item 16(a): Description of the 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. This information is transferred into a spreadsheet. After been captured in the spreadsheet the data is electronically transferred to a digital logging program (SABLE). In undertaking the exercise, the program is very specific in the requirements and standards it requires. 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 either into a modelling package (DATAMINE or GEMCOM). Both modelling packages are unforgiving in their acceptance of conflicting data. This is to say that if there are any overlaps in distances, inconsistencies in stratigraphic or economic horizon nomenclature, and then the input is aborted. This is the second stage of verification.

Having gone through the two stages of digital data verification a third stage of section construction and continuity is generated either through DATAMINE or GEMCOM. The lateral continuity and the packages of hangingwall and footwall stratigraphic units then have to align or be in a format consistent with the general geometry. Should this not be the case then the information is again aborted and thus the third stage of verification is reached.

The fourth and final stage of verification of the data is the geostatistical nature and distribution of the information. Anomalies either in grade, thickness, isopach and isocon trends are noted and interrogated. Should inconsistencies and varying trends be un-explainable then the base data is again interrogated until the suitable explanation is obtained.

Item 16(b): Comment on the Authors Verification or Comment on the Responsible Persons Verification Process:

The geological and economic base data has been verified by the First QP, and has been found to be acceptable.

Item 16(c): Nature of the Limitations of the Data Verification Process:

As in the case of all information, inherent bias and inaccuracies can and may be present. However with the verification process that has been carried out, should there be a bias or inconsistency in the data, the error would have 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 that it is entered into the deposit modelling software. In addition to ongoing data checks by project staff, the senior management and directors of PTM have completed spot audits of the data and processing procedures. Audits have also been done on the recording

of the drill hole, the assay interpretation and final compilation of the information. The individuals in PTM's senior management and board of directors who completed the tests and designed the processes are non-independent mining or geological Qualified Persons.

Item 16(d): Possible Reasons for not having completed a Data Verification Process:

All data has been verified before being processed.

ITEM 17 - ADJACENT PROPERTIES

Item 17(a): Comment of Public Domain Information of the Adjacent Properties:

The adjacent property to the WBJV is the Bafokeng Rasimone Platinum Mine, which operates under a joint venture between AP and the Royal Bafokeng Nation. The operation lies directly to the south of the Elandsfontein Project and operating stopes are within 1500m of the WBJV current drilling area. This is an operational mine and the additional information is published in the 2004 AP Annual Report which can be found on www.angloplats.com website. The reference to the BRPM operations is found on page 48 of the Operations Review and on page 80 where the official reserves and resource are quoted.

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

Salient features derived from the sources mentioned above include the following (Investment Analysts Report March 11, 2005, Anglo Platinum Website):

1. An original design of 200,000 tonnes per month Merensky Reef operation from twin declines with a dip mining method. A team approach. The mine also completed an open cast Merensky Reef and UG2 Reef operation and mechanised mine was started in the south part of the mine.
2. The planned steady state is to increase to 220,000 tonnes per month, 80% from traditional breast mining. As a result of returning to traditional breast mining development requirements reduced.
3. The plan also reverted to single skilled operators.
4. The mine mills about 2,400,000 per year with a built up head grade of 4.47 g/t 3PGE+Au in 2004.
5. Mill recovery in 2004 was 85.83%.
6. 200,000 refined platinum ounces are planned to be produced in 2005.
7. Operating costs per tonne milled in 2002, 2003 and 2004 were R284/t, R329/t and R372/t respectively.

Item 17(b): Source of Adjacent Property Information:

The BRPM operations information is found on website www.angloplats.com and the BRPM Royal Bafokeng Nation's information is found on website www.rbr.co.za.

Item 17(c): Applicability of the Adjacent Property Information:

Due to the WBJV being a continuous and adjacent ore deposit to the WBJV, the information obtained from the BRPM operations

is vital and appropriate in making decisions about the WBJV.

Item 17(d): Comment on the Application of the Adjacent Property Information:

The BRPM technical and operational information can be useful to the WBJV in so far as planning statistics are concerned. It must be remembered that the overall design and modus operandi of the WBJV is different to that of the BRPM operations and only certain aspects of the BRPM design can be used. The overall design recommendations for the WBJV have relied upon a more "industrial norm" approach by choosing the best practice approached across the industry.

ITEM 18 - MINERAL PROCESSING AND METALLURGICAL TESTING

The reader is referred to Section 25 of this Preliminary Assessment for discussions on the

Metallurgical aspects of this project.

ITEM 19- MINERAL RESOURCE ESTIMATES

Item 19(a): Standard Reserve and Resource Reporting System:

The author has complied with the SAMREC code of reporting of mineral resources and mineral reserves. The code allows for a resource or reserve to be upgraded (or down graded) if, amongst others, economic, legal, environmental, permitting circumstances change. The author has allowed for a geological and geostatistical set or rules for the classification of either the resource or reserve. The methodology also relies on the structural and facies aspects of the geology to define the resource classification. The principals of the reserve and resource classification are consistent with the Inferred, Indicated and Measured resource classification and the Probable and Proved reserve classification.

Item 19(b): Comment on Reserves and Resources Subsets:

This particular report deals primarily with the Inferred Resources. The specific data distribution and geographic layout does not allow the inferred resource to qualify for any upgrade to higher confidence resource categories. The total resource is therefore within the Inferred Resource category and therefore has NO further subdivision or sub classifications.

Item 19(c): Comment on Indicated Resource Subset:

The definition of the resource is as defined in the SAMREC code and is in no manner or form duplicated and double accounted.

Item 19(d): Relationship of the QP/s to the Issuer:

The Qualified Persons responsible for this report have no commercial or any other relationship with PTM other than to compile and comment on the contents of this report.

Item 19(e): Detailed Mineral Resource Tabulation:

This preliminary assessment was commissioned to update the resource covered in the "Western BIC Project" -Report dated March 3, 2005, within the project area and to evaluate the economic potential thereof.

From the interpolated block model a mineral resource was calculated for the domains of the Merensky Reef and one domain for the UG2 reef. Domain 1 covers the hartzburgite facies of the Merensky Reef and Domain 2 covers the feldspathic pegmatoidal pyroxenite facies of the Merensky Reef. Table 3 shows the tonnage and grade

for each domain at specific cut-off grade (3PGM+Au (cm g/t)). The block model cells with a channel width of less than 1 metre were diluted to a minimum channel width of 1m. Channel width values of greater than 1m were kept as is. The cut-off grade categories are on content (3PGM+Au (cm g/t)) because the interpolation was done on content, as was the mechanism for the change of support or post processing. Diagram 10 shows the grade tonnage curve for the different reefs and respective domains.

Table 3: Inferred Mineral Resource (Diluted to 1m minimum mining width)

Cut-Off Grade	Tonnage	Avg. 3PGM+Au Grade	Avg. Channel Width	Avg. Mining Width (1m minimum)	Metal Content 3PGM+Au	Metal 3PGM+Au
cm g/t	Tonnes	g/t	m	m	g	Moz
Merensky Reef - Domain 1 - Hartzburgite - type Reef						
0	13,870,586	9.67	1.11	1.12	134,112,425	4.312
200	13,869,781	9.67	1.11	1.12	134,111,228	4.312
400	13,671,466	9.77	1.11	1.12	133,509,878	4.292
500	13,203,917	9.97	1.11	1.12	131,634,208	4.232
600	12,363,873	10.31	1.11	1.12	127,522,342	4.100
700	11,195,722	10.79	1.11	1.12	120,763,773	3.883
1000	6,978,111	12.73	1.11	1.12	88,808,675	2.855
Merensky Reef - Domain 2 - Pyroxenite - type Reef						
0	15,474,713	1.06	0.42	1.00	16,383,388	0.527
200	1,991,262	3.73	0.42	1.00	7,423,431	0.239
400	534,406	6.47	0.42	1.00	3,454,966	0.111
500	321,585	7.80	0.42	1.00	2,508,726	0.081
600	206,025	9.12	0.42	1.00	1,878,574	0.060
700	138,019	10.43	0.42	1.00	1,439,376	0.046

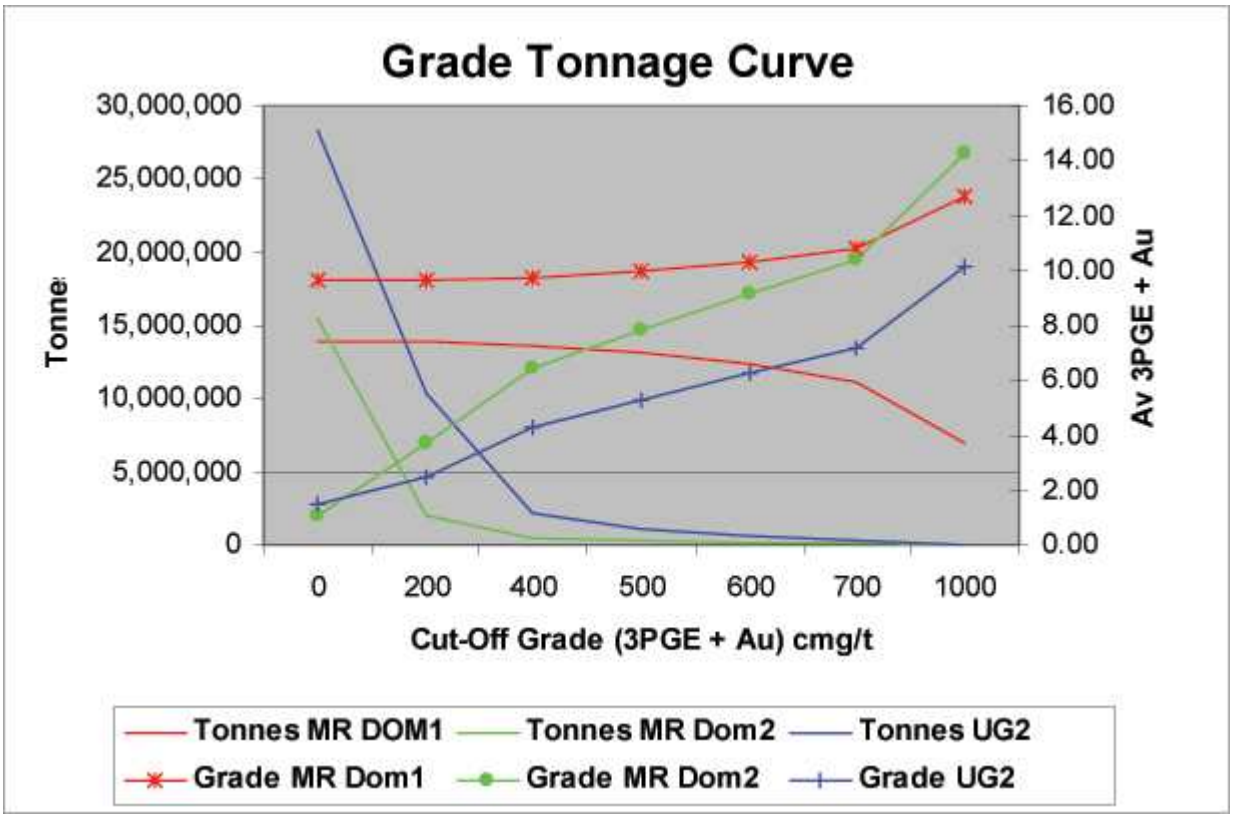
1000	50,502	14.30	0.42	1.00	722,368	0.023
			UG2 Reef Domain 1			
0	28,227,481	1.48	1.35	1.35	41,749,715	1.342
200	10,353,612	2.51	1.35	1.35	26,023,949	0.837
400	2,212,977	4.32	1.35	1.35	9,568,189	0.308
500	1,113,588	5.27	1.35	1.35	5,869,863	0.189
600	591,167	6.23	1.35	1.35	3,683,004	0.118
700	328,570	7.20	1.35	1.35	2,364,131	0.076
1000	69,796	10.11	1.35	1.35	705,429	0.023

(Footnote: If the Merensky Reef is less than 1metre then the value is corrected to 1m. Selected cut-off grades are broadly based on current economic considerations)

Domain 1 of the Merensky Reef has exceptionally high values. If the whole of the project area is considered as one domain (Merensky Reef -Domains 1 and 2) then the grade is 2.9g/t (no cut-off applied). Domain 1 represents of the Merensky Reef made up of the Harzburgitic reef type and is associated with high grades. When different

estimation methods are compared the simple kriging estimate is the more conservative estimate (at no cut-off grade). The composited sample data is on average 10.35g/t (1159cm g/t, 112cm), ordinary kriging 10.16g/t (1138cm g/t, 112cm), Sichel T is 9.99g/t (1119cm g/t, 112cm) and simple kriging is 9.67g/t (1086cm g/t, 112cm).

Diagram 10: Grade Tonnage Curve



Item 19(f): Key Assumptions, Parameters and Methods of Resource Calculation:

A total of 28 boreholes were drilled in the area of interest (Refer to Table 2 and Diagram 11) of which only 24 boreholes could be used for Merensky Reef mineral resource estimation and 22 boreholes for UG2 mineral resource estimation. A number of historical boreholes were originally found to not meet with the quality assurance criteria and were not used in the evaluation of the project area.

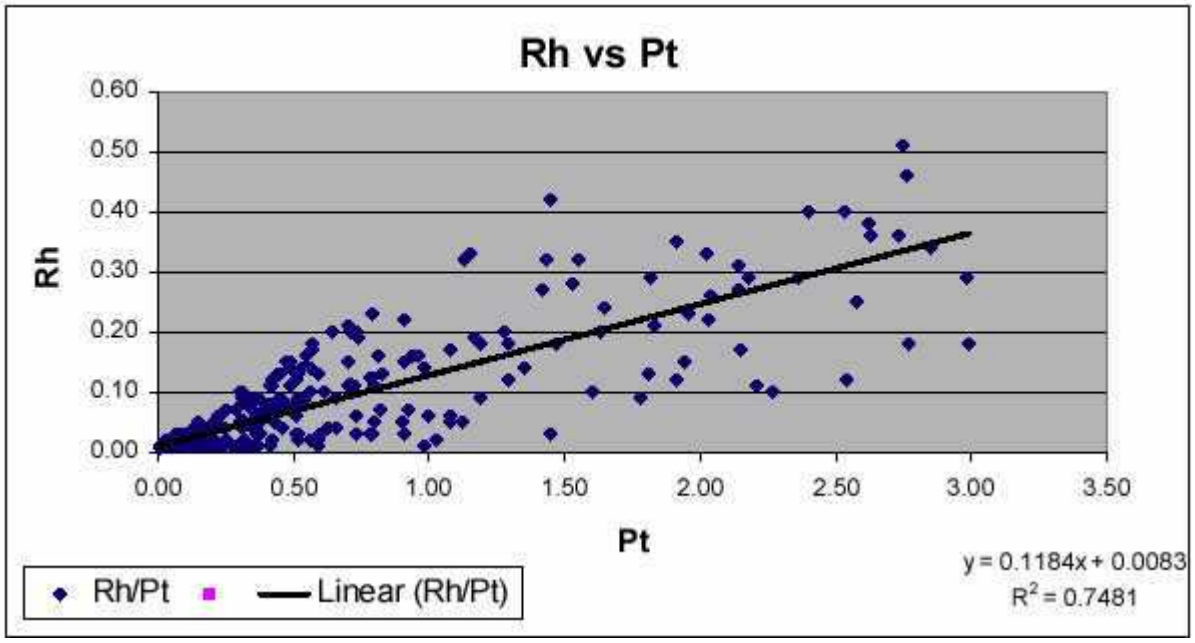
Mineral resources were estimated for the Merensky Reef based on 24 boreholes with 2 to 3 deflections per borehole and the UG2 reef based on 22 holes and deflections. A total of 10 boreholes intersected the Harzburgitic type reef and 14 boreholes the Feldspathic Pegmatoidal Pyroxenite-type reef. The assay values reflect 3PGM+Au. An area towards the south-west has been defined where resource estimation is not possible for the Merensky Reef. The reason is based on the diamond drilling information

having intersected the reefs at less than 50 m from surface resulted in an excessive core loss and often intersected units where a thinning of the reefs and/or stratigraphy occur leading to reef identification/correlation problems. No resource has been estimated for the northwestern part of the Feldspathic pegmatoidal pyroxenite reef type area since no grade data exist in this area. A mineral resource for the UG2 reef was based on 22 boreholes. The assay values reflect 3PGM+Au.

Both the Merensky and the UG2 reefs are on average 1m thick and therefore the full reef composites have been used for interpolation. The original borehole and deflections have been combined (weighted average) to represent a single intersection for each borehole. Borehole co-ordinates, reef width and PGM (3PGM+Au) grades used in the resource estimation exercises are depicted in Table 2.

The available borehole data consists of previously drilled AP holes and recently drilled PTM. The AP borehole PGM values consisted of Pt, Pd Rh and Au. Some of the AP drilled holes did not have Rh values and these were obtained from existing relationship of Pt and Rh values (Refer to Diagram 12). The following formula was used to calculate missing Rh values: Rh=0.1184x + 0.0083. The correlation coefficient for Rh vs. Pt is 0.7481.

Diagram 12: Scatter plot of Rh vs. Pt



In the evaluation process the metal content (3PGM+Au -cm g/t) and channel width (cm) are used. The channel width refers to the corrected reef width. The values have been interpolated into a 2D block model. The grade (g/t) has been calculated from the interpolated content and channel width values. All interpolated model cells for both the Merensky Reef and UG2 reefs of less than 1m have been diluted to reflect a minimum channel width cut of 1m. A regional dip of 6 degrees was used for channel width corrections.

The Merensky Reef was divided into two distinct geological domains or facies (Refer to Diagram 13) whereas the UG2 consists of only one geological domain (Refer to Diagram 14). Grade estimation was done in specific geological domains. The Merensky Reef in this area consists of two distinct reef types viz. Harzburgitic-type reef (interlayered harzburgite and feldspathic pegmatoidal pyroxenite units developed towards the north-east) and Feldspathic Pegmatoidal Pyroxenite-type reef occurring to the south-west with reef development deteriorating towards the west and abutting against a shear zone and/or the footwall to the Transvaal Supergroup.

Statistical Analysis

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

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

Table 4: Descriptive statistics for the Merensky Reef intersections

Variable	Descriptive Statistics (Spreadsheet1) Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.
DOM1ALL_MR_CW	10	1.147	0.4121	2.455	0	0.593
DOM1ALL_MR_Au	10	9.787	2.3348	16.151	19	4.402
DOM1ALL_MR_CMGT	10	1159.325	333.2103	3964.729	1136901	1066.256

Variable	Descriptive Statistics (Spreadsheet3) Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.
DOM2ALL_MR_CW	14	0.4271	0.019807	1.0296	0.12	0.3445
DOM2ALL_MR_Au	14	2.0906	0.090409	7.9333	5.51	2.3469
DOM2ALL_MR_CMGT	14	110.0872	1.505314	574.7434	31844.68	178.4508

Table 5: Descriptive statistics for the UG2 reef intersections

Variable	Descriptive Statistics (Spreadsheet5) Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.
DOM0ALL_UG2_CW	28	1.3818	0.606082	3.3109	0.51	0.7169
DOM0ALL_UG2_Au	22	1.9131	0.011954	5.7829	2.92	1.7085
DOM0ALL_UG2_CMGT	22	239.0512	1.039970	921.3384	51693.51	227.3621

The two domains for the Merensky Reef show different statistical relationships. Domain 1 is on the average thicker than Domain 2 lower grades. The thickness variation within the domains is small as can be seen in the variance values. The content (3PGM+Au cm g/t) values have a high variance as expected.

The histograms and normal probability plots indicate that the Domain 1 of the Merensky Reef did not have enough data point to create representative histograms and normal probability plots. Domain 2 of the Merensky Reef indicates that there might be two reef width populations but there were not enough information to separate the two populations. The grade histograms show the expected log normal distributions. The normal probability plots show no real outliers or anomalous values for grade.

No corrections were made to the data and the statistical analysis show the expected relationships for this type of reefs.

Variography

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

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

Table 6: Variogram parameters

Reef	Domain	Angle 1	Angle 2	Angle 3	Axis 1	Axis 2	Axis 3	Nugget %	Structure 1 Sill 1 %	Structure 1 Range 1	Structure 1 Range 2	Structure 1 Range 3	Structure 2 Sill 2 %	Structure 2 Range 1	Structure 2 Range 2
MR	1	0	0	0	0	0	0	25.25	80	245	245	1	100	510	510
MR	2	0	0	0	0	0	0	24.55	80	247	247	1	100	533	533
UG2	1	0	0	0	0	0	0	25	80	248	248	1	100	520	520

Grade Estimation

The full reef composite values (3PGM+Au content (cm g/t)) and channel width (cm) have been interpolated into a 2D block model. Both Simple Kriging ("SK") and Ordinary Kriging ("OK") techniques have been used. It has been shown that the SK technique is more efficient when limited data is available for the estimation process.

The 3PGM+Au concentration (g/t) was calculated from the interpolated kriging 3PGM+Au content (cm g/t) and channel width (cm). Detailed checks were done to validate kriging outputs including input data and kriged estimates checks, efficiency checks etc.

The simple kriging process uses a local or global mean as a weighting factor in the kriging process. For this exercise 750m x 750m blocks have been selected to calculate the local mean value for each block in respective domains. A minimum of 4 samples were required for a 750m x 750m block to be assigned a local mean value otherwise a domain global mean is assigned. The majority of the blocks used a global domain mean in the SK process with only a few blocks that used a local mean where there was enough data support.

The following parameters were used in the kriging process:

1. Point data - metal content (cm g/t) and channel width (cm)
2. 250m x 250m x 1m block size
3. Discretisation 25 x 25 x 1 for each 250m x 250m x 1m block
4. First search volume -1000m
    a. Minimum number of samples 4
    b. Maximum number of samples 40
5. Second search volume
    a. 1.5 x first search volume
    b. Minimum number of samples 2
    c. Maximum number of samples 40
6. Third search volume
    a. 3 x first search volume
    b. Minimum number of samples 1
    c. Maximum number of samples 20
7. Interpolation methods -simple kriging and ordinary kriging
8. Local and domain global mean values used in the simple kriging process

Diagrams 18 to 23 show the interpolated channel width, grade (g/t) and content (cm g/t) plots for the Merensky and UG2 Reefs.

Post Processing

During early stages of projects the data is invariably on a relatively large grid. This grid is much larger than the block size of a selective mining interest, i.e. selective mining units (SMU). Efficient kriging estimates for 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 over estimated. 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.

A selective mining unit (SMU) of 20m x 30m was selected with an expected future underground sampling configuration on a 20m x 20m grid. Information effects were calculated based on the SMU and the expected future production underground sampling configuration.

Within the parent blocks of 250m x 250m x 1m, the distribution of selective mining units has been estimated for various cut-offs. The latter has been estimated using lognormal distribution of SMU's within the large parent blocks -250m x 250m x 1m (See Assibey-Bonsu and Krige, 1999). This technique for post-processing has been used based on the observed lognormal distribution of the underlying 3PGM+Au 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 (on the basis of 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 200, 400, 500, 600, 700 and 1000 cmg/t.

A Specific Gravity (SG) of 3.2 for the Merensky Reef and 3.8 for the UG2 Reef was used for all tonnage calculations.

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 borehole 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. <10% deviation from mean -measured resource

b. 10 - 20% indicated resource

c. >20 inferred resource

Using the above criteria the current Merensky Reef and UG2 reefs in the delineated project area is classified as an Inferred Mineral Resource. Diagrams 24 to 30 show the different parameters that have been considered for the resource classification.

Item 19(g): Description of Potential Impact of the Reserve and Resource Declaration with respect to Environmental, Permits, Legal, Title, Taxation, Socio-economic, Marketing and Political Issues:

The intention of the report is to produce a Preliminary Assessment base on the inferred resource only. The confidence level is very low and thus the appropriate warning is hereby issued.

However in this report, assumptions are made regarding the environmental conditions, permitting, legal and political issues and assumed, with limited research are favourable. Taxation and marketing issues will be applied in real and un-escalated terms.

Item 19(h): Technical Parameters Effecting the Reserve and Resource Declaration which includes Mining, Metallurgy and Infrastructure:

Technical parameters specific to a planar and tabular precious metal deposit are well understood and are referred to as the "flow of ore" parameters. The results of the flow of ore parameters are detailed in Table 7 and Table 12.

The methodology takes into account the intentional and unintentional increase in tonnage due to mining. It also takes into account the unintentional and unaccounted loss of metal or metal not reaching the plant or recovered by the plant.

Item 19(i): 43-101 Rules Applicable to the Reserve and Resource Declaration:

In terms of which this report is issued, only the inferred resources can be used. The specific 43-101 regulations pertaining to this declaration are as specified in Item 4.

Item 19(j): Table showing the Quality, Quantity and Grade of the Multi-element Precious Metal Declaration:

Refer to Table 1a and Table 1b.

Item 19(k): Metal Splits for the Multi-element Precious Metal Declaration:

Refer to Table 1a and Table 1b.

ITEM 20 - OTHER RELEVANT DATA AND INFORMATION

RSA Reserve and Resource Declaration Rules

The South African Code for Reporting of Mineral Resources and Mineral Reserves (SAMREC Code) sets out minimum standards, recommendations and guidelines for Public Reporting of Exploration Results, Mineral Resources and Mineral Reserves in South Africa.

Documentation prepared for Public Report must be prepared by or under the direction of, and signed by, a Competent Person. A Competent Person 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 Competent 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 reasonable and realistic prospects for eventual economic extraction.

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

ITEM 21 - INTERPRETATION AND CONCLUSIONS

Results

A mineral resource estimate has been calculated for the Merensky Reef and UG2 Reef from available borehole information. The mineral resource for both the Merensky and UG2 reefs are classified as an Inferred Mineral Resource. The Merensky Reef was divided into two distinct domains based on different facies with specific lithological and mineralised characteristics. The in-situ interpolated grade models have been diluted where channel width was less than a 1m mining width.

	Cut-off Grade	Tonnes	Grade	Channel	Diluted	Tonnes

	(cm g/t)	(Mt)	3PGM+Au (g/t)	Width (metres)	Channel Width (metres)	3PGM+Au (t)
MR Domain1	200	13.87	9.67	1.11	1.12	134.11
MR Domain 2	400	0.53	6.47	0.42	1.00	3.46
UG2 Domain 1	400	2.21	4.32	1.35	1.35	9.57
TOTAL						**147.13**

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 is 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 Mineral Resource and could result in grade and variance relationships changes with additional data. With more data the variogram models will improve with resultant confidence in the estimation.

Reliability of the Data

The data has specifically inspected by the First QP and found to be reliable and consistent.

Strengths and Weaknesses with respect to the Data

Weaknesses: As a result of the limited drill data only an Inferred Resource level of confidence can be implied. Borehole surveys are not completed beyond a GPS position and this may impart a small amount of error in block sizes. Additional geotechnical work will be required to assess mineability. Although the metallurgical properties of the Merensky and UG2 reefs are well known, detailed metallurgical work will need to assess the recoverable amount of the reported grades.

Strengths: QA/QC work done on laboratory samples is of a high standard, including the insertion of blanks and standards. The data has been found to be consistent and well structured. The support of the digital data by paper originals, change of custody and drilling records is well assembled and of high quality.

Objectives of the Projects Adherence to the Scope of Study

The intention of this phase of the work program was to be able to have sufficient data and confidence to achieve a Preliminary Assessment report. This has been achieved and thus the objectives of the program have been met.

ITEM 22 - RECOMMENDATIONS

(a) Further Work Required

The current mineral resource is classified as an Inferred Mineral Resource. The Inferred category implies that there is not sufficient data to evaluate the resource with confidence. It is also expected that with more boreholes at a closer grid the grade and variance relationships will be different. The focus should be on Domain 1 of the Merensky Reef with borehole spacing of at least 500m, but preferably on a 250m grid for geostatistical considerations.

The current Merensky Reef and UG2 Reef mineral resource is classified as an Inferred Mineral resource. There is however the area (Refer to Diagram 31) that could be upgraded to an indicated mineral resource with the proposed additional drilling in this area. Current parameters, including but not being limited to kriging efficiency, 90% lower confidence limit, number of samples used in estimate and variogram ranges show that this area is just outside the criteria for indicated mineral resource. It is expected that with the proposed drilling in this area that the area would be upgraded to an Indicated Mineral Resource.

(b) Recommended Phases of Work

The main focus should be to upgrade the Inferred Mineral Resource within Domain 1 of the Merensky Reef to an indicated resource. Preliminary mineralogical and metallurgical work needs to be done on a selective and representative number of intersections in order to ensure that the Merensky Reef is likely to behave in a manner as reported by the mines to the north and south of the property.

(c) Objectives to be achieved in Future Work Programs

The objectives of the future work programs are to ensure the integrity of the resource by upgrading the confidence level to that of the Indicated Resource category. The drilling will also allow for investigation of opportunities for shallow mineralisation within the Elandsfontein Project area.

(d) Detailed Future Work Programs

To upgrade the resource (based on 500m x 500m grid) to an Indicated Resource additional boreholes are required to be drilled on a 250m x 250m grid. Geostatistical parameters derived from the modelled semi-variogram for Domain 1 of the Merensky Reef support a range of 200m as sufficient to upgrade the resource to reserve. Thirteen (13) high priority boreholes are planned as a first phase of upgrading:

No of Boreholes	Average depth (metre)	Total Inclusive Cost/metre	Total metres (plus deflection drilling)	Rate of Drilling	Total Cost
13	500m	R500/m	7865	80 days	R 3.93M

It is recommended that two deflections (one long 80m, one short 25m) apart from the original intersection be drilled on the Merensky Reef for statistical manipulation. The rate of drilling based on

5 Machines which average 25m/shift taking into account site moves and rehabilitation. Drilling will then take three months to complete and taking into account the assaying process, the data will be ready by the end of October 2005.

If needed, in the case of poor geological confidence, a second phase of infill drilling may be required. The most north-western area (Mining Block 8) of the Elandsfontein Project area also still requires some drilling to be done. That will be an additional five boreholes. The infill drilling could amount to another twelve boreholes. Thus the second phase follow-up drilling will be as follows:

No of Boreholes	Average depth (metres)	Total Inclusive Cost/metre	Total metres (plus deflection drilling)	Rate of Drilling	Total Cost

17	450m	R500/metre	9010	90 days	R 4.50M

It is recommended that only one long deflection (80m) be drilled apart from the original intersection. The rate of drilling is based on 5 machines averaging 25m/shift. This takes into account site moves and rehabilitation of the drill sites.

Drilling will thus take three months to complete commencing November 2005 and, taking into account the assaying process the data will be ready end of March 2006. This schedule takes into account the closure of business over the Christmas Break.

The above two phases of drilling will be sufficient to upgrade the resource and allow the project to be recommended for a pre-feasibility level of

(e) Declaration by QP with Respect to the Project Warranting Further Work

Domain 1 of the Merensky Reef has been shown to contain 13.87Mt at 9.68g/t (3PGM+Au). The current mineral resource classification is an Inferred Resource. At this stage only the global mean is of any value and any mine planning is of low confidence and could be incorrect for specific areas. The confidence in the project value will

improve if an area can be upgraded to an Indicated Mineral Resource. It is recommended that additional infill drilling be done in Domain 1 of the Merensky Reef.

ITEM 23 - REFERENCES

Assibey-Bonsu W. and Krige D.G (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.

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.

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 .

Siepker, EH and Muller CJ (2005). Geological Assessment and Resource Estimation of the Western BIC Project, North West Province of the Republic of South Africa.

Wagner, PA (1926). The preliminary report on the platinum deposits in thesouth-eastern portion of the Rustenburg district, Transvaal. Mem. Geol.Surv.S Afr., 24, 37pp.

ITEM 24 - DATE

The date of this report is 8 August 2005.

ITEM 25 - ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

ITEM 25(a): Technical Assessment:

INTRODUCTION and BACKGROUND

The Merensky Reef on the Elandsfontein property is said to outcrop on surface and dips in an easterly direction to the boundary at an average dip of about 6 degrees. The deepest point will be approximately 560 metres below surface. The UG2 reef is present on the property, is considered to be well developed but the grades are low and as such is excluded from this current assessment - this can be regarded as up-side potential for the project.

The current geological and structural model (as at 02 August 2005) has a total of 13 distinct blocks of ground, separated by faults and/or dykes, called Blocks 1 to 13. These blocks of reef will each need to be mined individually and thus the conceptual mining layout and anticipated production schedule has been based on this arrangement.

It is anticipated that the stoping width will be 10 cm above and below the defined channel width of the reef seam - this has been used for all tonnage and grade calculations in this assessment. The geological data indicates a channel width varying from 103 to 116 cm, and thus the stoping width will vary from 123 to 136 cm across the mine.

Platinum Group Metals (RSA) (Pty) Limited (PTM) has supplied the above information.

To adequately access this ore deposit, it is proposed that the most cost effective method will be via a twin vertical shaft system located in the centre to the deposit. The depth precludes the use of open pit techniques and multiple decline technology does not access the ore deposit quickly enough, whilst traversing un-pay blocks. This assessment is based on twin vertical shafts to a depth of 665 m below surface.

This economic assessment is based on Merensky Domain 1 Reef only and excludes Domain 2 and UG2 in their entirety.

The expected production rate will be 113 000 tonnes per month of reef plus working cost waste production of 28 000 tonnes per month. This mine is expected to produce 7 900 kg per annum 4PE's (Platinum, Palladium, Rhodium and Gold) in concentrate (254 000 oz 4PE's) and yield approximately 6 800 kg or 218 400 oz 4PE's after refining, subject to the terms of the toll treatment agreement.

DISCLAIMERS

In preparing this Technical Assessment report the authors relied upon:

- Geological and assay information were supplied by PTM.
- Drill hole analytical and survey data compiled by PTM.
- 'In-house'Turnberry experience and available date base information.

Other than as disclosed here in, the outside sources of information were relied upon without extensive inquiry and review. The authors make no particular representation to the degree of accuracy of that information and do not bear liability thereto. A dataset was compiled from all available data supplied by Anglo Platinum as well as data added collected during this assessment phase by PTM geological personnel.

This report was prepared as a Technical Assessment to provide an initial evaluation of the Elandsfontein Project and the report and conclusions are based on

- Information available at the time of preparation
- Data supplied from project data base and experience
- The assumptions, conditions, and qualifications set forth in this report

This report is intended to be used by PTM and the Joint Venture partners, subject to the terms and conditions of its contract with the authors and contributing persons. The contract permits PTM to file this report as a Technical Assessment but warns that there has been limited confirmation of the assumptions used in this evaluation and thus, any use of this Technical Assessment by any third party is at that party's sole risk.

GEOLOGY of the ELANDSFONTEIN PROJECT AREA

Geological Blocks

The current geological and structural model (as at 02 August 2005) indicates 13 mining blocks as detailed below and in the attached zonal plan below, for the Merensky Domain 1 Reef. The blocks have been labelled numerically and indicate the shallowest and deepest portion of the block. The approximate total tonnage for each block is also shown with the anticipated stoping tonnage available at a stoping width of 20 cm more than the channel width and a rock density of 3.2. The tonnage is based on the cut-off grades or an assumed mining extraction of 70%. This data is based on an Inferred Resource only and thus caution is to be exercised when reviewing this assessment.

ELANDSFONTEIN PROJECT - Mining Blocks from Resource Model

Block	Top	Bottom	Total Resource Tons	Stoping Tonnage at CW	Mineable Tons at CW	Corrected Tons for SW	Grade	Co
1	-50	-150	5,646,621	154,971	-	-	-	
2	-130	-260	1,463,903	54,815	-	-	-	
3	-300	-440	1,513,090	637,179	637,179	760,484	7.616	
4	-380	-520	1,615,399	1,130,779	1,130,779	1,335,742	8.643	
5	-250	-500	1,327,478	603,359	603,359	717,426	7.182	
6	-350	-500	2,501,556	1,634,276	1,634,276	1,934,861	6.098	
7	-400	-550	1,869,812	1,308,868	1,308,868	1,543,097	8.075	
8	-420	-560	2,088,127	1,461,689	1,461,689	1,725,199	7.820	
9	-420	-560	729,839	510,887	510,887	598,729	9.340	
10	-400	-510	1,357,299	950,110	950,110	1,125,779	7.999	
11	-430	-530	3,655,743	2,559,020	2,559,020	3,027,490	8.615	
12	-400	-520	1,122,635	761,182	761,182	902,969	8.278	
13	-120	-400	4,391,898	147,505	-	-	-	
			29,283,401	**11,914,640**	**11,557,349**	**13,671,775**	**7.928**	**1**

Table 8b - Mineable Tonnage for Elandsfontein Project from PTM -Merensky Domain 1

The above table (Table 8b) indicates that the total tonnage to be mined at a stoping width 20 cm more than the channel width and excluding any dilution factors should be about 13.7 million tonnes at a grade of 7.93 g/t 4PE's, containing 108.4 tonnes of 4PE's (an estimated 3.39 million oz). Including additional dilution factors plus Block Factor and Mine Call Factor, increases the tonnage processed to almost 15.7 million tonnes at a reduced grade of 6.55 g/t 4PE's.These estimated tonnages have been used in the corresponding financial models and will need to be more accurately defined as the geological and structural models develop with the drilling, possible seismic survey and geo-statistical analysis.

The geological data used for this interpretation is detailed in Table 8 and summarised in Table 8a. These data are indicating both Merensky Reef (MR) and UG2 reef horizons.

Block 1 is potentially a shallow mine, accessed by a decline system. Considering the tonnage and grade of the block at a cut-off of 400 cmg/t is 155 000 tonnes at 6.36 g/t (total block tonnage is 5.6 million tonnes), it is assumed that this block cannot be economically mined and is thus excluded from the mineable zone. The mineable portion only represents 2.7% of the resource based on current data. The potential working cost profit from this block is estimated at R48 million and will not cover the anticipated expenditure required to access this block of ground.

Block 2 is potentially a shallow mine, but would be accessed by the shaft system. Considering the tonnage and grade of the block at a cut-off of 400 cmg/t is 55 000 tonnes at 6.51 g/t (total block tonnage is 1.5 million tonnes), it is assumed that this block cannot be economically mined and is thus excluded from the mineable zone. The mineable portion only represents 3.7% of the resource based on current data. The potential working cost profit from this block is estimated at R18 million and will not cover the anticipated expenditure required to access this block of ground.

Block 13 is also potentially a shallow mine, which would be accessed by the shaft system. Considering the tonnage and grade of the block at a cut-off of 400 cmg/t is 148 000 tonnes at 6.32 g/t (total block tonnage is 0.9 million tonnes), it is assumed that this block cannot be economically mined and is thus excluded from the mineable zone. The mineable portion only represents 4.0% of the resource based on current data. The potential working cost profit from this block is estimated at R45 million and will not cover the anticipated expenditure required to access this block of ground.

The UG2 resource is not considered to be economically recoverable on a stand-alone basis, but could be recovered utilising the Merensky infrastructure. The potential recoverable zones are seen as Zones 6 to 10 and could be included at the end of the Merensky production profile. This has not been included in this assessment and could add some upside potential to the project, subject to economics.

The remaining blocks of ground are considered to be viable and mineable from a shaft complex located centrally in the deposit. The shaft position has been selected as the southern section of Zone 4, as indicated below.

The geological information associate with the Merensky Domain 1 area is categorised as an Inferred Resource and thus caution must be exercised in reviewing this evaluation.

Water Potential

To date, the geological drilling, which has been conducted by PTM, has not reported undue water production within the holes that have been drilled. It is thus reasonable to assume that underground water will not be a concern to any producing mine on this property. This will need to be confirmed during subsequent drilling campaigns.

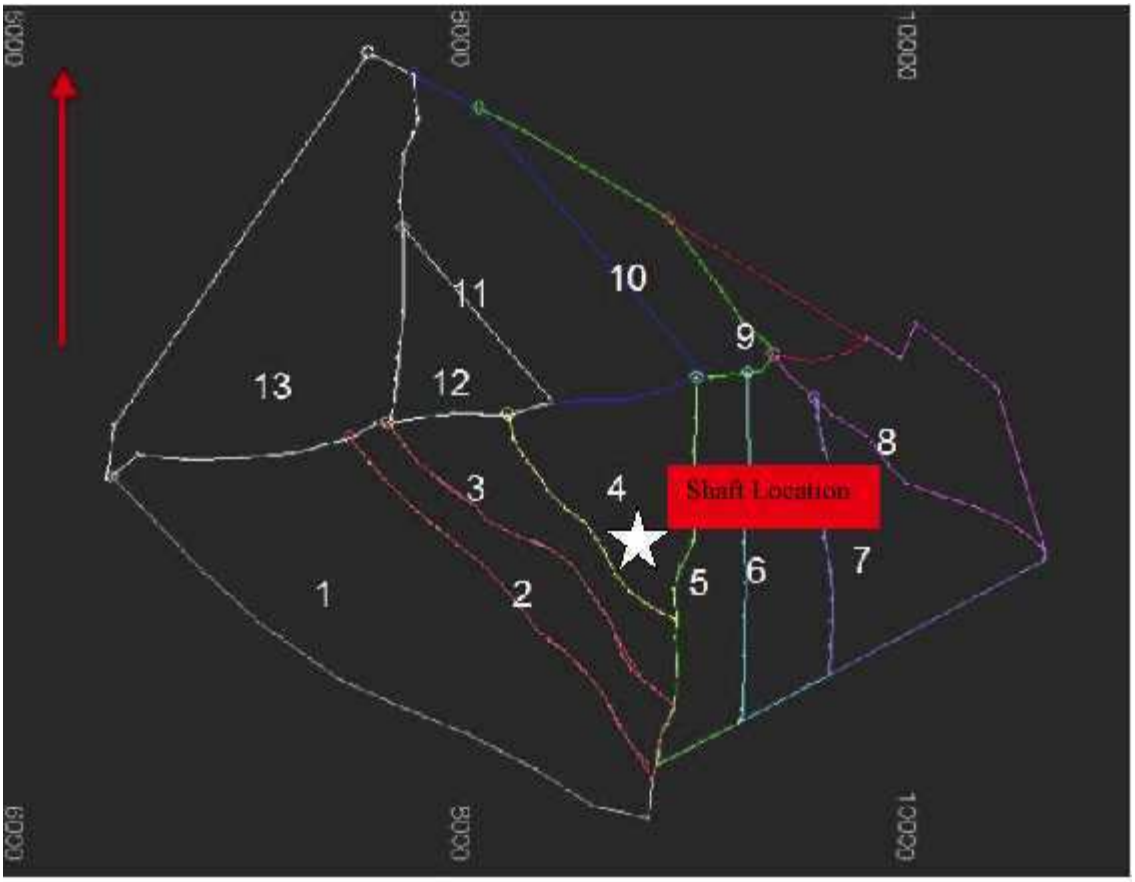


Diagram 33 - Geological Blocks at Elandsfontein Project as per PTM

CONCEPTUAL MINE DESIGN

Geology and Ore Deposit characteristics

Drilling is still in progress to confirm all aspects of the ore deposit.

At present the ore deposit can be described as a flat dipping (at approximately 6 degree) Merensky Reef that can be mined at a stoping width 20 centimetres more than the channel width, based on 10 cm above and below the defined channel of the reef seam. This factor has been used for all tonnage and grade calculations in this assessment. The geological data indicates a channel width varying from 103 to 116 cm, and thus the stoping width will vary from 123 to 136 cm across the mine. Where channel widths have been smaller than 100cm as in Blocks 1, 2 and 13 they have been re-calculated to an effective channel width of 100cm.

The Merensky Reef is present from surface down to approximately 560 metres below surface. The ore deposit is disturbed by several major faults and dykes breaking it up into discrete blocks, which may each require separate access development and mining and engineering infrastructure. Based on current information the mine has been divided into 13 such blocks as detailed in Table 8b and shown in Diagram 33. It is anticipated that only 10 such blocks are economically recoverable, based on current data.

The current interpretation may change in some of the details but it will still be valid as far as predicting the overall nature of the ore deposit and the infrastructure required to exploit it. The deeper, North Eastern section of the mine at this stage of the programme appears to have better grades. The grade weakens gradually to the shallower, southern and western parts of the mine.

This economic assessment is based on Merensky Domain 1 Reef only and excludes Domain 2 Reef and the UG2 Reef in their

entirety. The geological data upon which this economic assessment is based is categorised as an Inferred Resource only and thus the appropriate caution needs to be applied to this evaluation.

Surface considerations

A surface outcrop may exist and a shallow oxidized zone will be present down to about 50 metres below surface. For this assessment, this zone has been excluded due to expected poor metallurgical recoveries as well as complications with existing surface structures.

These surface structures also complicate the option of a starter mine with its surface infrastructure in the south eastern portion of the ore deposit, namely the Hotel and Lion Park Reserve.

The site chosen in this assessment for a vertical shaft to best exploit the resource is slightly South East of the centre of the ore deposit (in block 4) and does not present any surface complications. The surface rights are not currently owned by the project but by local farmers and are available, subject to negotiation. The shaft location is marked on Diagram 33.

Access to Ore Deposit

As the shallower portions of the ore deposit are considered to be un-economic to mine and the higher grade reef is located at depth, the option to access the ore deposit via a decline system has been rejected as inappropriate. The most satisfactory alternative will be to access the higher grade blocks as quickly as possible, and this will be via a surface shaft system.

Primary access to all parts of the mine is to be provided by a twin vertical shaft system down to 665 metres below surface. The shaft system will have intake and return facilities and will support production of 113 000 tonnes per month of reef. The Main Shaft will be 7.7 m in diameter and will have one Man Winder, one Service Winder and one Rock Winder whilst the Ventilation Shaft will be 5.5 m in diameter and will be used as the second outlet from the mine. The shaft sizes have been selected to suit the ventilation requirements of the mine.

The shaft system is situated south-east of the centre of the ore deposit, to the south of block 4 as indicated in Diagram 33, to give quick access to the richer portions of the mine. A shaft pillar of 150 m diameter will be left and possibly extracted at the end of the shaft's life. These dimensions must still be verified by normal Rock Engineering processes but are considered fair and reasonable.

Each of the 13 identified mining blocks can be developed for mining in a variety of ways, e.g. by accessing it on each Main Shaft level, or by accessing it top and bottom from the Main Shaft and then establishing a single decline system (4m wide x 4m high) to mine out the block, or top access only servicing a decline system. For the purpose of this assessment, it is assumed that all blocks will be exploited by service declines within the block, accessed from the Main Shaft at the top of the decline (3.4m x 3.2m station haulages) and using the lower access level to transport ore back to the Main Shaft.

Development of the declines will be done using LHD's to the tip positions on each level. The service declines will not initially be equipped with winders or rails but will be large enough to accommodate chairlifts and LHD's.

Shaft Configuration

The twin shaft system will be developed to a depth of 665 metres below surface.

The flat dipping nature of the reef results in a level spacing in the shaft of no more than 30 metres, such that the raise lengths in the stoping horizon will be about 200 metres.

Analysing the data in Table 8b, the first level in the shaft will need to be at about 300 metres below surface to allow top access to blocks 3 and 5, either directly or via an incline system. The shaft arrangement will be as indicated in the table below, showing the

locations of each level to be cut into the shaft barrel on both shafts.

ELANDSFONTEIN Shaft Level Spacing

No.	Level	Elevation (mbs)	Access to Blocks
0	Surface	0	Surface Bank
1	300	300	Top Access to Blocks 3 & 5
2	330	330	Intermediate Access
3	360	360	Top Access to Block 6
4	390	390	Top Access to Blocks 4, 7 & 10
5	420	420	Top Access to Blocks 8, 9, 11 & 12
6	450	450	Bottom Access to Block 3
7	480	480	Intermediate Access
8	510	510	Bottom Access to Blocks 5, 6 & 10
9	540	540	Bottom Access to Blocks 4, 11 & 12
10	570	570	Bottom Access to Blocks 7, 8 & 9
11	620	620	Transfer and Pumping Level
12	665	665	Shaft Bottom

Surface Shaft Level Spacing (mbs = metres below surface)

The up cast shaft will be concrete lined but not equipped, the stage winder and stage will remain in place after sinking and the emergency hoisting facility will operate on the rope guides for the stage.

The Main Shaft will have a finished dimension of 7.7 metres diameter whilst the ventilation shaft will be 5.5 metres finished diameter. The shafts will be located 30 metres apart, skin-to-skin to suit the Rock Engineering criteria. There is to be a 150m diameter shaft pillar to ensure the viability and safety of the shafts for the life of the mine.

The shaft dimensions are to be adequate to allow the transport of heavy mobile equipment such as LHD's underground. Each station is to be adequately sized to facilitate LHD movement and maintenance near the shaft. It is anticipated that 2.3m ^{3} LHD's will be utilised such as the Tamrock/Sandvik EJC116 or equivalent. These aspects need to be detailed during the next phase of the project.

Stoping method

This type of ore deposit, at the depths encountered at Elandsfontein, has been most successfully mined by either breast or down dip stoping, leaving 15% of the reef as in-stope support. This allows for the most cost effective way of supporting the stopes, as only elongate support is anticipated to be necessary in conjunction with these pillars. This support proposal is subject to the normal Rock Engineering checks that have not been carried out at this stage.

The decision to go breast or down dip mining is often influenced by the direction of fracturing in the rock, the ideal choice being to carry the face at right angles to this faulting or fracturing. Both methods have been used successfully, with Lonplats favouring down dip and Anglo Platinum and Impala Platinum favouring breast mining For the purpose of this initial assessment down dip stoping is assumed.

Drilling is to be by conventional, handheld pneumatic jackhammers. Face cleaning will be by face winches in each panel, scraping into a centre gully, which will be serviced by one tip at the bottom of the raise. Panel length will be determined by faulting and raise spacing, but will be between 30 and 35 metres. A panel length of 30 metres is used in this assessment. Face advance of 15 metres per month is planned, as per the industry norm. This results in a production of 1 640 tonnes per month per panel.

Block development

Footwall drives, 3.0m by 3.0m, are to be carried 15 metres below reef.

This assessment is based on all flat, block development being track bound. The potential exists to do this development with LHD's and to tram the waste rock with the LHD's or truck with trucks back to the decline tips, subject to the overall distance (LHD's are

adequate up to 200m).

Raise spacing is 40 metres and one panel is mined between raises. Raise dimensions are 3.0m by 2.0m. This allows for the anticipated rolling and grade cut-off nature of the reef horizon to be catered for in the mine planning and design.

At each raise position, a crosscut to reef for services and access is developed.

In the drive a boxhole, 2.4m by 1.2m, to reef is developed just past the crosscut position for reef removal from the stopes.

Level spacing can vary but given the flat dip of the reef, should not exceed 30 metres giving raise lengths of approximately 200 metres.

Return Air Way's on strike are not required, as the return air will move to the top of the block through the worked out areas.

On each half level (i.e. north and south), six raises will be in various stages of stoping. This allows production of 12 500 reef tonnes per month per half level or 25 000 reef tonnes per month per level.

The above layout results in 15 square metres / total metre developed, or 48 square metres per waste metre developed.

Rock Engineering

It has been assumed that the stoping environment will be relatively stable from a rock engineering and hanging wall support aspect. The hanging wall will be supported by rock pillars, wooden sticks and where appropriate composite wooden/concrete packs. To achieve the necessary in-stope support, it is expected that 15% of the reef horizon will remain in the stope, as stated above.

In addition, due to the broken nature of the ore deposit, extra regional and 'fault line'support will be required to ensure a safe working environment -to achieve this aspect, it has been assumed that an additional 10% of the reef horizon will be required to remain in position in the mining areas.

As a result of the geological uncertainty and the effects of potholes on the mining operations, an allowance of 5% of the reef horizon has been allocated to geological losses. Based on the current geological information provided by PTM, this is regarded as fair and reasonable.

Combining the above allowances results in a 30% 'loss'of reef horizon to factors associated with geology and a safe working environment. This factor has been applied to the tonnages calculated and displayed in Tables 8, 8a and 8b and summarised in Table 7a below.

As these areas are primarily allocated for support reasons to ensure a safe working environment, no allowance has or will be made to reclaim these pillars at any time during the life of the mine.

As stated previously, the expected shaft pillar size will be 150m diameter and the shaft will be spaced 30m apart, skin-to-skin. These parameters need to be confirmed by normal rock engineering principals and evaluation.

Ventilation Requirements

The duty of the shaft combined with the shallow depth means that ventilation requirements will determine the ultimate dimensions

of the shaft.

It has been assumed that 500 cubic metre/sec of air will be delivered into the mine, corresponding to 3.5 m^3 /sec per ktonne per month -this is an acceptable design parameter and within industry norms. The down cast Main Shaft will be 7.7 metres diameter whilst the up cast Ventilation Shaft will be 5.5 metres diameter, resulting in a down flow velocity of 10.7m/sec and an up flow velocity of 21.0m/sec in the respective shafts.

Each block of ore will be accessed by twin haulages on the top and bottom levels -one for intake and the other for return air. If the vertical extent of the block is greater than 60 metres, an intermediate level with return airway may be developed to improve access to the ore deposit and for development of the infrastructure.

It is not anticipated that any form of refrigeration will be required during the operation of the mine, due to the shallow nature of the ore deposit. This will need to be confirmed during any subsequent definitive feasibility study.

Modus Operandi for Sinking and Development

The conceptual modus operandi for the project has been used to develop the Project Schedule (Appendix B) and can be described as being fast tracked from implementation into production. This means that most early equipment for shaft sinking, (e.g. sinking winders, compressors, etc.) will be supplied as part of the sinking contractors equipment and not an early purchase by the Elandsfontein Project.

It is assumed that no construction work will commence on the project until the go-ahead has been received from the Joint Venture Project Team. This means that only conceptual drawings have been developed for the mine layout and associated infrastructure. In addition, no ordering has been done for long lead items such as winders, compressors, mills etc.

It is anticipated that the Definitive Feasibility Study (DFS) will commence in early 2006 calendar year and will be completed in approximately nine months, ahead of the decision point to proceed with the project implementation. During the DFS, sufficient detail will have been requested and received from the major sinking contractors to allow a final decision to be taken for the shaft sinking contractor selection, process plant design and construct and major equipment (e.g. compressors, etc.), without the necessity to again submit enquiry documents -this is expected to occur in mid to late December 2006.

It is expected that both the Main and Ventilation Shafts will commence at approximately the same time. The Ventilation Shaft will progress slightly more rapidly that the Main Shaft, as a result of the reduced concrete lining requirements due to there being no equipment to install in the shaft. As a result, a number of the stations will be cut from the Ventilation Shaft rather than the Main Shaft as the sinking progresses. In the programme, it is proposed that every second station and station infrastructure will be cut from alternative shafts.

As the Ventilation Shaft will not be equipped, apart from the sinking rope guides, this shaft will be complete approximately one year ahead of the Main Shaft. During this time, the 'in circle'development for the ore passes and dams is to be completed using the hoisting facilities of the Ventilation Shaft. In addition, some of the critical primary development to reef will also be completed.

It is assumed that no reef mining will occur whilst the Main Shaft is not available for hoisting. Reef mining, even 'on-reef' development is to be delayed until after the Main Shaft and the ore pass system is available for continuous production.

When the Main Shaft is available, the primary development will be accelerated on the critical levels to allow a more rapid build up of access points to reef.

Ore Flow Factors

The ore flow factors used in the production and financial models for Elandsfontein can be summarised as follows:

- **Channel Width** (CW) is the width of the reef zone containing 4PE's as reported by the geologists during the drilling campaign.
- **Stoping Width** (SW) is defined as the width required to achieve the correct recovery of reef from the stope face. The SW consists of the channel width of the reef zone plus a 10cm overbreak on the hanging and footwall horizons, thus making a total of 20cm additional material beyond the Channel Width. As can be seem in Appendix A **Error! Reference source not found.** , the channel width for zone 6 is 108.7cm and the stoping width has been increased to 128.7cm. This results in additional tonnage at zero grade and a corresponding reduction in stope grade.
- **Stope Dilution** is defined as the additional tonnage generated at zero grade as a direct result of the mining activities. In the production forecast, a figure of 15% has been used for stope dilution. This is made up of 10% as a direct result of the necessity for stope gullies, winch beds, etc., resulting from the necessity to access the reef horizon without discarding any reef. In addition, the platinum industry has a norm of 4% overbreak within the stoping horizon (due to hanging and foot wall breakage), plus an allowance of 1% for the mine tonnage excess / shortfall calculation. This results in the allocation of the 15% dilution factor, increasing tonnage, decreasing grade but maintaining the same kilogram allocation.
- **Block Factor** for this project has been set at 100%. The block factor is the reconciliation between the resource grade and the currently mined grade as defined by in-stope sampling. At Elandsfontein, it is assumed that the resource and the mined grade will be the same and thus the factor is 100%.
- **Mine Call Factor** for this project has been set at 95%. The mine call factor is the reconciliation between the currently mined grade as defined by the in-stope sampling and the grade presented to the processing plant. The platinum industry has a historical mine call factor of between 95 and 100%. There is no reason to expect that the mine call factor will be greater than 95%.
- **Concentrator Recovery** for this project has been calculated, based on anticipated tailings values and comparing to industry standards and norms. It is anticipated that Elandsfontein will have a Concentrator Recovery of between 87 and 87.5% resulting in a tailings value of approximately 0.86 g/t 4PE's. This is regarded as fair and reasonable until such time as metallurgical testwork confirms these recoveries.

Applying all of these factors to the Elandsfontein geological and ore flow models, results in changes of tonnages, metal contents and grades. Table 7a below summarises the effects of these changes.

Elandsfontein Project
Summary of Ore Flow Calculations

		Tonnage tonnes	Kilograms 4E's
1	**Total Resource at 0 cmg/t cut-off**	29,283,401	143,9
2	**Total Resource at economic cut-off grade**	11,914,640	110,6
3	**Total Resource excluding un-economic blocks**	11,557,349	108,3
4	**Economic Resource corrected for SW**	13,671,775	108,3
5	**Delivered to Mill with Mining Factors included**	15,722,542	102,9

6	**Concentrate Recovered based on Tons Milled**	15,722,542	89,6
7	**Smelter Recovered based on Tons Milled**	15,722,542	77,1

Based on the Inferred Resource for the Merensky Domain 1 zone only

Table 7a - Summary of Ore Flow - Tonnages and Grade Comparison

To apply the ore flow factors from the Total Resource (categorised only as an Inferred Resource) excluding un-economic blocks as indicated in line 3 of the table, and comparing to the delivered to the mill parameters in line 5, there is a 30% loss of grade but only a 5% loss of contained metal. Lines 6 and 7 are based on tonnage milled, although the concentrate tonnage will be about 384 589 tonnes whilst the tonnage within the smelter plant is immaterial to this profile.

It is noted that the predicted mill head grade (Line 5 above) for this assessment is somewhat higher than that which has been reported by the neighbouring and district mines. This seems to be due to higher in-situ grades being predicted as the ore flow factors are based on industry norms. Whilst this is encouraging for the project, this apparent anomaly needs to be investigated and verified in the next phase of the project.

Production Scheduling

As indicated above, each level within a mining block will be capable of delivering 12 500 tonnes per month from one side of the decline. This implies that each level will be able to produce 25 000 tonnes per month. For this assessment, it has been assumed that only one level will be in production at any one time from a particular block of ore.

The total available stoping tonnage, as indicated in **Error! Reference source not found.** above, is 13.7 million tonnes. Using ore flow parameters as detailed elsewhere, this will result in a milled tonnage in excess of 15.7 million tonnes. The proposed shaft complex envisaged for Elandsfontein will be capable of delivering approximately 113 000 tonnes per month of reef and thus the anticipated mine life on Merensky Reef will be 148 months or more than 12 years. This is regarded as fair and reasonable for this project.

Analysing the above information, the desired throughput of 113 000 tonnes per month will be achieved from four and a half production levels or units in the mine -again for this assessment, this represents five blocks of ore. This tonnage will come from the five separate mining blocks and there will be at least one new block in preparation, acting as 'spare'in case of loss of production due to equipment failure, geological losses caused by potholes or unexpected faults, etc.

It is anticipated that the timing to production from mining blocks from the top level will be as follows: -

- 7 months to the first level
- 15 months to full production (25 000tpm) on that level / block

From the Main Shaft two blocks must be accessed immediately and the development to the next three must be started simultaneously. The overall production schedule would then be

- Shaft commissioning - Month 1
- Development to Blocks 3 and 4 - Month 4 (150m)
- Production from Blocks 3 and 4 (50 000 tpm) - Month 26
- Development to Blocks 5,6 and 10 - Month 25 (1000m)
- Full production plus reserve levels - Month 47

Steady State Production

It is expected that the mine will be capable of consistently producing 113 000 tonnes of reef per month from between four and five producing areas. Reviewing the attached production schedule (Graph 3), it can be seem that at times, more than five blocks will need to be in production. This production profile has been selected, as it does not have the disadvantage of a long production tail

at the end of the mine life.

The mining and development parameters used in this evaluation can be summarised as

- 113 000 reef tpm at the planned stoping width
- 480 m/month block development waste at an average 30 tonnes per metre
- 900 m/month block development reef at an average 20 tonnes per metre
- 240 m/month access and ore pass development at an average 30 tonnes per metre
- 120 m/month decline development (4m x 4m) at an average 36 tonnes per metre

The above parameters result in 28 000 tonnes per month of waste being produced.

Waste Rock Storage

It is anticipated that a total of 3.7 million tonnes of Working Cost waste will be produced during the life of the mine. In addition, the capital development is expected to generate in excess of 2.5 million tonnes of waste. A facility to store in excess of 6.2 million tonnes of waste rock will be required.

It would be advantageous to consider contracting a local waste rock crushing operator to reduce the size of the storage facility by producing aggregate for the construction industry, subject to the waste rock being suitable as aggregate and local demand.

MINE ENGINEERING

Engineering Infrastructure

The engineering infrastructure that will be required for this project will be typical of any similar sized mine associated with the Bushveld Igneous Complex (BIC), utilising a similar operating methodology, namely:

- Electrical supply from the local generating authority, ESKOM
- Electrical reticulation on surface and underground
- Water supply from the local authority, Magalies / Rand Water Board
- Hoisting capacity for the Main Shaft consisting of a Rock Winder, a Man Winder and a Service Winder
- Hoisting capacity for the Ventilation Shaft consisting of a Stage Winder and an Emergency Winder for men
- Conveyor and transfer facilities in the Main Shaft headgear for rock hoisting

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- Ventilation fans on surface attached to the Ventilation Shaft
- Settling and pumping for underground water
- Water storage on surface for treatment and distribution of service water
- Compressors to provide the necessary compressed air for the mining operation
- Workshops to repair the mining fleet and other equipment -whilst a significant proportion of this service can be outsourced, it will still be necessary to have repair facilities on site
- Management and supervisory offices, stores, etc
- First aid, proto facility, lamp-room and associated facilities
- Warehouse and garage facilities for both surface and underground supplies equipment
- Explosives magazine
- Change house facilities for all employees and associated laundry and Sewage Plant

As a result of the proximity of the mine to existing accommodation, it is not expected that the mine will provide any form of accommodation to its employees, but will pay a gate wage, adequate to cover the accommodation requirements.

Capital Requirement

Provision has been made in the Capital Forecast for all the above aspects, as well as a provision for the underground mining fleet of R100 million.

There is an additional requirement from about year 6, to allow the introduction of additional infrastructure to sustain the production from the mine. This is additional to that required in achieving the initial production targets during the first few years of production.

The total engineering infrastructure capital requirement is estimated to be R473 million for the equipment supply (excluding the Concentrator and Tailings Dam at R215 million) plus the EPCM portion of the contract in Phase 1, estimated to be about R70 million. In Phase 2, the capital is estimated at R75 million plus the EPCM costs at about R12 million.

Operating Costs

The cost to operate the engineering infrastructure detailed above is included in the current Operating Cost estimate.

The services to the mine, namely electric power and water, are costed separately and shown as a line item in Table 11.

The power requirement is based on 55 kWhr/tonne milled for the Concentrator and 65 kWhr/tonne mined for the remainder for the other infrastructure. In addition, it has been assumed that the water consumed by the mine will be 0.5m ^{3} /tonne milled for the concentrator and 0.4 m ^{3} /tonne mined for the remainder of the mine. During the next phase of the project, these service requirements will need to be determined from a complete mine power and water balance.

METALLURGY

Metallurgical Processing

The reef from the Elandsfontein underground mine will require metallurgical processing to recover the associated platinum group metals and base metals in the ore. There are a number of possible process routes to consider for the operation, namely:

1. Conventional processing consisting of:
    - Base metal Concentrator producing an upgraded product (concentrate) in 4PE's
    - The concentrate is fed into a smelter producing an upgraded smelter matte
    - The matte is further processed in a base metal removal plant producing copper and nickel by-products and a 4PE rich sludge
    - The final sludge is fed into a precious metal refinery producing the final precious metals in either metal or salt form
2. Toll treatment of the reef from Elandsfontein at a neighbouring mine, provided that there is sufficient capacity available and a mutually beneficial toll treatment agreement can be reached
3. Leaching of the concentrate by either the Panton or the Platsol processes -these options are not industrially proven and thus are not recommended for Elandsfontein
4. Leaching of the reef or whole ore by either the Panton or the Platsol processes - these options are not industrially proven and thus are not recommended for Elandsfontein

Considering the above options, the only viable process route is the first or second, as they are extremely well proven and reliable technology. The capital infrastructure associated with smelter, base metal removal plant and the precious metal refinery are extremely high and require high levels of production to even consider them as an option. In the case of Elandsfontein, the levels of

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production do not warrant even considering the metallurgical processing of the reef beyond concentrate production.

Considering that one of the joint-venture partners is a major platinum producer with significant infrastructure within a reasonable distance of Elandsfontein, the process option selected as appropriate for this project is to

1. Mill the reef and produce concentrate on the mine site
2. Dispatch the concentrate under contract to the third party toll treatment facility, probably located near Rustenburg, South Africa.
3. Have the toll treatment facility, process the concentrate to produce the final products and purchase them from Elandsfontein as per the contract terms

The option of toll milling the reef is potentially viable, but would require transportation of the reef from the Elandsfontein Main Shaft to the Concentrator Plant that would be doing the toll milling. The economics will need to be carefully examined and for this Technical Assessment, this option is considered to be inappropriate for 113 000 tonnes per month of reef.

Thus, the process option to be recommended for Elandsfontein will consist of a conventional multistage milling and flotation circuit with concentrate being upgraded and thickened prior to dispatch. The tailings will be stored on a local tailings dam. The Concentrator will have a capacity of 113 000 tonnes per month or 4 310 tonnes per day or 180 tonnes per hour. This size of processing plant is within industry norms, is based on standard, proven technology and is considered to have little technical risk.

Process Description

Whilst no mineralogical or metallurgical testwork has been performed on this particular reef, it is reasonable to expect that the metallurgical performance will be similar to that of the neighbouring mines and to the associated mines in this area of the Bushveld Igneous Complex. Whilst this expectation cannot be guaranteed, the assumption is considered reasonable for this Technical Assessment.

Waste rock will be hoisted separately from the mine and discharged into a receiving bin in surface. The rock will be trucked or conveyed away to the waste rock dump for storage or additional processing to manufacture aggregate.

Reef will be hoisted from underground via the Main Shaft and deposited into a receiving bin next to the headgear. A conveyor will remove the reef from the headgear and it will be primary crushed to smaller than 150mm. This crushed ore will be stockpiled ahead of the Concentrator to de-couple the underground mine from the processing plant.

The crushed ore will be reclaimed from the stockpile and fed into the primary Semi-Autogeneous Grinding (SAG) circuit in closed circuit with a classifier. The finer fraction will be directed to a primary flotation circuit whilst the coarser fraction could be subjected to 'flash' flotation prior to returning back to the Primary SAG Mill. The tailings from the primary flotation circuit could be subjected to additional secondary ball milling and secondary flotation. Even tertiary ball milling and flotation may be considered, but this is most unlikely. The tailings from the secondary flotation circuit are expected to be of low enough value to be discarded to the tailings dam.

The concentrates from the primary and secondary flotation circuits will be upgraded in the cleaner flotation plant to produce a higher grade concentrate with reduced levels of chrome contamination. The tailings from the cleaner circuit will be subject to tertiary milling before being returned to the main flotation circuit.

Metallurgical Performance

The plant performance is expected to be good, as a result of the predicted high grade of the ore to be produced from the mine. The average mill feed grade for the life of the mine will be over 6.5 g/t 4PE's. This high head grade is expected to result in recoveries in excess of 87% for 4PE's into concentrate containing 250 g/t 4PE's or more. The average mass pull to concentrate will be less than 2.5%.

Tailings Disposal

Considering that the expected milled Merensky tonnage will be 15.7 million tonnes plus any potential UG2 reef that might be milled in future, the tailings dam capacity requirement will need to be approximately 22 million tonnes, less the mass pull of about 2.5% to concentrate. This will require a dam with a capacity of 9.75 million m^3. A footprint of about 42 hectares would be required for a tailings dam of 30 m in height.

Current tailings disposal regulations may require the dam to be lined to prevent the contamination of ground water. Water run off from the dam will be contained and returned to the processing plant.

Process Plant Costs

A metallurgical plant with the above processing capability is expected to cost approximately R200 million (in July 2005 money terms), including all associated infrastructure. In addition, the first phase tailings dam is expected to cost R15 million with a second phase expansion of the tailings dam costing an additional R10 million, 6 to 7 years after the commencement of production.

The operating cost of the plant is expected to be approximately R40 per tonne milled, excluding concentrate transport, toll refining charges and services such as electricity and water supply.

Toll Treatment Conditions

The anticipated concentrate toll refining terms and conditions are subject to a confidential contract with the Toll Refiner, but for the sake of this Technical Assessment, it has been assumed that the metal recovery will be 86% with a treatment charge of R500 per tonne of concentrate and a refining charge of R2 500 per kilogram of contained 4PE's. These terms and charges are subject to negotiation and do not necessarily reflect the final condition. These assumptions are considered to be reasonable within the context of industry practice and are based on smelting and refining cost and recovery data published by South African producers. These terms also assume that the third party toll refiner either has the capacity or will develop the capacity to consume the concentrate produced from the Elandsfontein Project.

There will be minimum quality conditions applied to the concentrate, and these have been assumed to be the following

- Concentrate grade to be better than 200 g/t 4PE's
- Contained chromite to be better than 1%
- Concentrate moisture to be better than 15%

If these are not achieved, penalties could be applied and for this Technical Assessment, penalties have been assumed for contained chromite and moisture but with no penalty for the concentrate grade. The penalties amount to approximately R1.5 million per annum.

These assumed conditions have been used in the associated economic evaluation, and amount to approximately R24.50 per tonne milled, and are considered to be reasonable within the context of industry practice.

Analytical Facilities

It is anticipated that all analytical requirements will be outsourced to experienced analytical laboratories located in the Rustenburg area.

Metallurgical Testwork

As previously stated, no mineralogical examination of the geological core from the current drilling campaign at Elandsfontein have been conducted to date, and thus there is no indication of the likely metallurgical performance from the Concentrator to be located at Elandsfontein. In addition, no metallurgical testwork has been conducted on any of the produced geological core.

It is necessary that during the next phase of the project, some initial mineralogical and metallurgical examinations be conducted to verify the assumed likelihood of similar performance to neighbouring or district processing plants.

The concern with Elandsfontein, it that the predicted head grade is somewhat higher than has been reported by the neighbouring and district mines, and this reflects in the predicted overall Concentrator recovery.

PROJECT SCHEDULE

The entire project has been scheduled as per the attached Gantt Chart in Graph 3.

The capital expenditure programme has been based on this schedule.

The mining rates that are applicable to this project schedule are summarised as

- Prepare to sink -207 days
- Shaft Pre-sink -1.5 m/day
- Shaft Sinking in the barrel -4m per day
- Station development -8 to 14 days

- Main Shaft equipping -25m per day
- Station equipping -20 days nominally
- Flat 'in circle'development -3m per day
- Flat block development for haulage and RAW -40m per month

These rates are regarded as fair and reasonable for this project, provided that the sinking is in dry shaft conditions (with normal cover drilling protection for water intersections), as currently anticipated. If water is encountered, these sinking rates will need to be revised.

Production time estimates for the mining project are expected to be: -

- Time to first level from shaft -12 month
- Time to next level in block -7 months
- Time to full production on a particular level -15 months
- Time to full production across the mine i.e. 113 000 tpm -54 month

The highlights of this proposed project schedule include:

- Project start date of December 2006
- Ventilation Shaft completed -December 2008
- Main Shaft equipped -December 2009
- Begin Stoping -April 2010
- Full Production (113 000 tpm) achieved -October 2011

Analysing the Valuation Model, the following comments are also evident:

- Project Start date -December 2006
- The maximum draw down of project financing -June 2010
- Project is cash neutral -November / December 2014
- Project achieves R1 billion return on investment -August 2017

ENVIRONMENTAL

The proposed Elandsfontein Project will be situated between the towns of Boshoek and Sun City in the North West Province. The project is a greenfields site that will be adjoining a similar mining site in the area. The greater Rustenburg district is a heavily industrialised area with a strong mining culture.

A detailed environmental study will need to be undertaken prior to work commencing on the project site. The objectives of the environmental study will be to:

- Identify the issues associated with the Elandsfontein Project, which are most likely to affect the biophysical and socio-economic aspects of the surrounding environment;
- Conduct a review of the applicable environmental legislation;
- Determine and document the aspects of the project, which will require further detailed investigation.

In order to meet the objectives the following activities will need to be undertaken:

- Site visits;
- Review of existing information;
- Review of the applicable legislation;
- Compilation of a Scoping Report according to the requirements of the Minerals and Petroleum Resources Development Act:
- Brief description of the environmental setting;
- Envisaged impacts on the environmental aspects of concern;
- Nature and extent of proposed specialist investigations.
- Outline of the environmental processes and authorisations applicable to the Elandsfontein Project.

The environmental consequences of the proposed project, both positive and negative, are to be addressed in the Environmental Impact Assessment / Environmental Management Programme. The specific requirements, which must be implemented to prevent unnecessary environmental degradation, whilst promoting economical and social upliftment are to be included in these documents. The process is to be conducted in an open and transparent manner to ensure that all aspects and issues of concern are taken into account.

No public participation meetings with the Interested and Affected Parties (IAP) have been held at this stage of the Technical Assessment. As soon as the official 'go ahead'for the Definitive Feasibility Study (if not earlier) has been given, this activity as well as the compilation of the EMPR and other documents will need to be instigated, whilst the engineering design is progressing and before activities have commenced on site.

Ground Water Removal

The removal of ground water as the dewatering of the lower workings of the mine begins development, is most likely to result in the lowering of the water table in the immediate vicinity of the mine. In addition, this could cause the water table to be lowered in the general area of the mine during production ramp up, but may be especially noticeable when the mine is in full production. This aspect could thus render any neighbouring surface borehole dry. This aspect is likely to cause concern during the public meetings with the IAP's.

Rehabilitation Fund

The potential funding of the requirements for the Mine's Rehabilitation Fund has not been taken into account as a separate cost element within the current working cost model. The rehabilitation fund may also be assisted with financing from the sale of assets (although a minimal revenue is expected) at the end of the mine's life and any 4PE's recovered from plant clean up.

Current Environmental Concerns

There are no current environmental concerns on the property, apart from the surface infrastructure at the Hotel and Lion Park, to the east of the proposed shaft position. In addition, the proprietor of the Hotel and independent farmers currently owns the land. No significant agricultural activities are practiced on the property.

Tailings Dam

A tailings disposal facility will be required to contain up to 21.5 million tonnes of ore. Considering the bulk density of this material and the anticipated height restriction of 30m, the footprint of the tailings dam will need to be approximately 42 hectares with the entire impoundment facility requiring up to 65 hectares.

Waste Rock Dump

A waste rock dump will be required to contain in excess of 6.2 million tonnes of waste rock. Considering the bulk density of the rock and the anticipated height restriction of 30m, the footprint of the waste rock dump will need to be approximately 15 hectares with the entire impoundment area being 20 hectares. A smaller facility could be required if an agreement can be entered into with a local producer for crushing rock for aggregate purposes.

Infrastructure

The infrastructure required for the Mine Operation, apart from the tailings dam and waste rock dump will require approximately 75 hectares and will include, but is not limited to the following: -

- Shaft system
- Shaft bank area
- Compressor house and cooling towers
- Workshops and stores
- Offices
- Water storage and treatment plant
- Mud and dirty water storage
- Ventilation plant
- Parking area for mine vehicles
- Access roads
- Parking area for private vehicles

In addition, the Concentrator will require an area of about 25 hectares as a footprint.

CAPITAL EXPENDITURE

The capital expenditure as detailed in Table 10 is a current estimate of the required funding to achieve the desired level of production and sustainability for the project. The capital estimate is based on developing a mine and concentrator facility only, with no reference to smelting or further downstream processing. The costs indicated are based on data base information and no quotations have been received from contractors or vendors to support the indicated costs. Cost estimates as such are 'order of magnitude' estimates only.

There are a number of types of costs included in Table 10, namely

- "PTM" Costs - Costs prior to Definitive Feasibility Study
- "Project" Costs -Definitive Feasibility Study
- Phase 1 Capital Costs
- Phase 2 Capital Costs
- On going Capital Cost Requirements

These costs are defined and further explained as below:

The " **PTM" Costs** are those costs which will be incurred by PTM up to the time when a Pre-Feasibility Study has been completed and will include current geological drilling and seismic survey data purchase, initial geological modelling and some geo-statistical evaluation, initial metallurgical testwork, initial mine planning based on the geological modelling and structural plans, the costs of the scoping and pre-feasibility studies and the ongoing PTM management costs. Included in these costs are the historical costs already incurred on the project by PTM, but excludes the purchase of the mineral or surface rights.

The " **Project" Costs** are those that will be incurred between the time when the JV partners agree to a Definitive Feasibility Study proceeding and the completion of the DFS. This will include comprehensive metallurgical testwork, additional geological drilling, finalisation of the geological model, structural model and geo-statistical analysis, additional mine planning details, the definitive feasibility study costs, land purchase for surface infrastructure and ongoing project management costs by the JV partners. Included in these costs will be the EMPR and EIA documentation requirements.

The **Phase 1 Capital Costs** details the expected capital cost to achieve full production from the underground mine and will include all necessary surface infrastructure, metallurgical plant, shaft systems, mining equipment, underground development to the initial production areas and the access to the stoping areas to achieve 113 000 tonnes per month.

The **Phase 2 Capital Costs** details the expected capital cost on infrastructure to sustain the levels of production for the life of the mine. This includes any additional surface infrastructure required, completion of the underground development to access all production blocks; additional tailings dam

capacity and additional mining equipment. This expenditure should be funded out of revenue generated by production and the positive mine cash flow.

The **On-going Capital Cost Requirement** details the costs associated with upgrading the infrastructure during the production life of the mine and includes mining fleet, processing plant and infrastructure upgrade. This expenditure should be funded out of revenue generated by production and the positive mine cash flow.

The total capital expenditure in July 2005 money terms is estimated to be R1 961 million for the entire project excluding the PTM costs and this is summarised in the table below.

As previously stated, these costs exclude any provision for housing of employees.

Elandsfontein Project - Capital Summary

PTM Costs	R 23,400,000		
Project Costs	R 31,000,000	**31,000,000**	**R**
Phase 1 Finance Capital		**R**	
1,428,901,321			
Plant & Surface Infrastucture	R 688,000,000		
Shafts & Incircle Development	R 326,611,866		
Underground Development	R 227,911,022		
Engineering, Procurement, etc	R 186,378,433		
Phase 2 Finance Capital		**R**	
429,678,119			
Plant & Surface Infrastructure	R 75,000,000		
Shafts & Incircle Development	R 26,508,375		
Underground Development	R 272,124,772		
Engineering, Procurement, etc	R 56,044,972		
Ongoing Capital		**R 71,700,**	
Combined	R 71,700,000		
Total Capital (excluding PTM Costs)		**R**	
1,961,279,440			

Capital Expenditure Summary

The accuracy of the above capital estimate is anticipated to be plus or minus 30%.

This cost is comparable to the reported cost estimates of similar sized projects within the South African mining industry and is considered reasonable for a project of this size and scope.

WORKING COSTS

The operating cost estimates have been derived by benchmarking the operating costs of similar projects in the South Africa platinum mining industry, The impact of mining depth has been factored in, using prior experience of the study team. The cost estimates used for this study are summarised below and indicate a total 'On Mine'cost of marginally less than R300 per tonne milled to produce and transport concentrate to a smelting facility. Total costs per tonne milled are estimated at under R324 per tonne. These costs exclude any royalty payments or penalties incurred.

Elandsfontein Project - OPEX Summary

WORKING COST SUMMARY	R/ton Milled
Mining Cost	R 206.96
Processing Cost (excluding services)	R 41.22
Services	R 24.56
Administration & Overhead Costs	R 26.36
TOTAL On Mine Costs	**R 299.09**
Smelting & Treatment Charges	R 24.49
TOTAL Costs	**R 323.59**

Operating Cost Summary

As discussed above, these study estimates are factored or benchmarked cost estimates only. It is believed that they are within 25% accuracy. More accurate zero based estimates should be done during the phase of the overall project.

As stated previously, no specific provision has been made in these costs for the rehabilitation fund requirement. It is currently assumed to be included in the Operating Cost estimate.

STAFFING

It is expected that the mine will provide employment for about 2 800 people. This should be verified by a detailed manning study in the next phase of the work. This aspect is extremely important as the project area has a high level of unemployment.

The area in which the mine is located has been near a mining district and supplying labour to the mines for many decades. The available labour is expected to be sufficiently industrialised to provide

trained manpower to Elandsfontein and this labour will be available to work with minimal additional training having been conducted, without compromising safety.

As stated previously, it is anticipated that Elandsfontein will pay a gate wage, adequate to cover all accommodation expenses and thus there will be no accommodation units constructed for any employees. This aspect has been excluded from the capital schedules, but is implicitly included in the Operating Cost estimate.

ECONOMIC VALUATION

Metal Prices and Revenue

The revenue to the mine is based on metal prices and exchange rates as at 01 July 2005 and is indicated in Table 9.

ELANDSFONTEIN PROJECT

PGM Basket Calculation

	Base Price	Exchange Rate	Discount	Metal Price	Split	Heat Grade	Assumed Price

		$US/oz	R/$US		R/kg	%	g/t	R/kg
Basket Metal	Pt	$871	6.55		R183,422	61.0%	4.00	
	Pd	$184	6.55		R 38,748	30.0%	1.97	
	Rh	$1,930	6.55		R 406,434	4.0%	0.26	
	Au	$426	6.55		R 89,710	5.0%	0.33	
	3PGEs's & Au	$685	Basket Price		R 144,254	100.0%	6.55	R 144,000
	Ir	$154	6.55		R 32,430			R 32,000
	Ru	$68	6.55		R 14,320			R 14,000
	Os	$750	6.55		R 157,941			R 150,000
		$US/ton			R/ton	Prices 01-Jul-05		
	Cu	$3,500.00	6.55	$150.00	R 21,943	Date 01-Jul-05		R 21,000
	Ni	$14,500.00	6.55	$70.00	R 94,517			R 94,000

Table 9 - Metal Prices used for Project evaluation

The metal splits are as reported by PTM from the borehole results and are regarded as fair and reasonable for this portion of the Bushveld Igneous Complex ores. It is also reasonable to expect that the metal ratios indicated will be maintained in the final product.

The basket price used in the economic evaluation is R144 000 per kg of 4PE's.

The other metals used for revenue purposes in this evaluation are copper and nickel only. The OPM's (other precious metals such as Ruthenium, Iridium and Osmium) have not been included in the revenue calculations.

Marketing of the Product

The markets for the product are not a concern to the WBJV. The conditions of the JV are such that a "Sale of Ore" and/or "Sale of Concentrate" proforma agreement is in place. The absolute conditions and costs of the agreement still need to be agreed upon but the proforma and willingness of AP to market the metal is open to the WBJV.

Taking this into account, the average "basket price" for the product is R144,000 per kilogram 4E's. A full breakdown of the prices, metal spilt and exchange rates used are shown in Table 9. As shown in Table 9, the platinum value is US$871/oz (United States Dollar "US$"), the palladium value at US$184/oz, the rhodium value at US$1,930/oz and the gold is taken at US$426/oz. The average PGM plus gold value has been calculated at USS685/oz. Using an exchange rate of R6.55 (South African Rands "R") to the US$ results in R183,422/kg for platinum, R38,748/kg for palladium, R406,434/kg for rhodium and R86,710/kg for gold. Given the average of metal splits, platinum at 61%, palladium at 30%, rhodium at 4% and gold at 5%, the average basket price R144.000/kg (prices as of 1 July 2005).

Royalties and Penalties

Considering the South African Government's current position on the Money Bill in which the mining industry will be subjected to a royalty payment (probably based on revenue) for precious metal production, the economic evaluation has assumed that the following royalties will apply from 2008:

- 4% royalty for platinum
- 3% royalty for palladium
- 3% royalty for Gold
- 0% royalty for Other PGM's
- 0% royalty for base metals

Penalties have been discussed previously, but are expected to reduce the overall income by about R1.5 million per annum.

Escalation and Inflation

The economic evaluation can make provision for price inflation, exchange rate escalation and metal price escalation, but for this Technical Assessment, all escalations have been eliminated and the economics are based in July 2005 money terms only.

This decision has been taken so as not to 'cloud'the economics with inflationary data. The valuation results can be improved or worsened by minor changes to the differences between inflation and escalation parameters chosen.

Economic Evaluation

The production profiles developed above and detailed in Table 12 form the basis of the economic evaluation for the Elandsfontein Project. In addition, the Capital Cost Estimate detailed in Table 10 and the Operating Cost schedules in Table 11 provide the necessary financial inputs to the model. Caution is to be exercised when reviewing this evaluation, as the production data is based on an Inferred Geological Resource only.

The actual Financial Model is available in Table 14 but is summarised in Table 13, along with the potential sensitivities associated with the major inputs into the model.

The project has a base case Pre-tax NPV of R1.91 billion at a 5% discount rate with an IRR of 18.9%. The NPV calculation has a base date of July 2005 and excludes the capital costs attributable to the initial PTM Costs. Assigning a corporate tax rate of 29% to the cash flow, the NPV at 5% drops to R1.29 billion with an IRR of 16.0%. The Pre-tax and Post-tax NPV at both 10% and 15% discount rates are detailed below.

No escalation or inflationary effects have been included in the economic evaluation to improve the financials in any way.

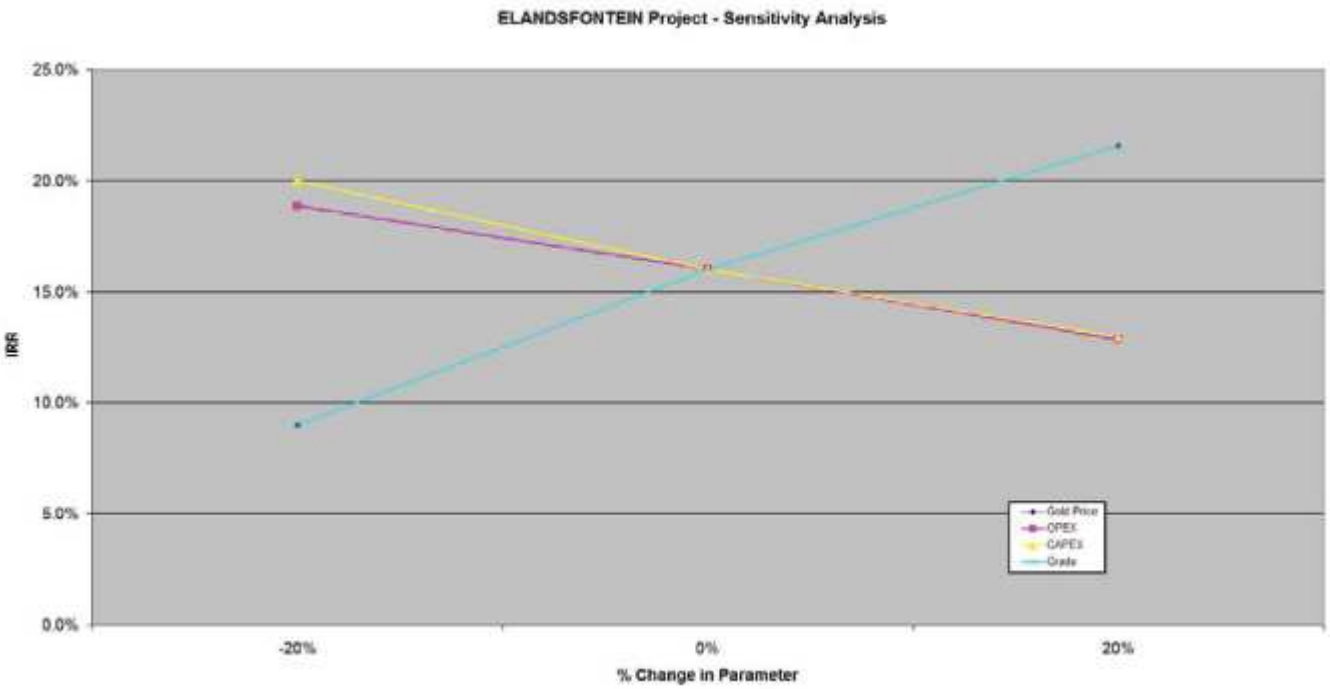
The Payment terms for concentrate delivered to the toll smelter have been assumed to be immediate and not incur a waiting period, which may be negotiated with the smelter operator.

ELANDSFONTEIN PROJECT

SENSITIVITY ANALYSIS

Parameter	Change in Parameter	Base case	Change in Parameter
PGM Basket Price	-20%	0%	20%
NPV @ 5% (before Tax)	R 696,621	R 1,908,960	R 3,121,299
NPV @ 5% (after Tax)	R 409,626	R 1,288,285	R 2,157,648
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
Opex	-20%	0%	20%
NPV @ 5% (before Tax)	R 2,490,825	R 1,908,960	R 1,327,095
NPV @ 5% (after Tax)	R 1,706,661	R 1,288,285	R 867,937
IRR (before Tax)	22.1%	18.9%	15.3%
IRR (after tax)	18.9%	16.0%	12.8%
Capex	-20%	0%	20%
NPV @ 5% (before Tax)	R 2,217,628	R 1,908,960	R 1,600,291
NPV @ 5% (after Tax)	R 1,525,722	R 1,288,285	R 1,048,523
IRR (before Tax)	23.4%	18.9%	15.5%
IRR (after tax)	20.0%	16.0%	12.9%
Grade	-20%	0%	20%
NPV @ 5% (before Tax)	R 696,621	R 1,908,960	R 3,121,299
NPV @ 5% (after Tax)	R 409,626	R 1,288,285	R 2,157,648
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
MCF Change	-3%	95%	3%
NPV @ 5% (before Tax)	R 1,717,538	R 1,908,960	R 2,100,382
NPV @ 5% (after Tax)	R 1,150,410	R 1,288,285	R 1,426,160
IRR (before Tax)	17.8%	18.9%	20.0%
IRR (after tax)	15.0%	16.0%	17.0%

Table 13 - Economic Evaluation - Sensitivity Data at 5% Discount



Sensitivity Analysis Graph

The above table and graph indicate the sensitivity of the project economics (at a discount rate of 5%) to changes in the input parameters. The factors are based on revenue and expenditures and flexed by plus and minus 20%. In addition, due to the sensitivity of the project to MCF, the changing of the MCF by plus or minus 3% is also indicated. The same sensitivity table, at a discount rate of 10% and 15%, is shown in the following tables.

The project will be cash neutral in November / December 2014, some 96 months from committing to the project. The cash neutral position is 56 months from the commencement of production, all subject to the zero escalation and zero inflation aspects of the project.

The maximum project funding requirement will be R1.37 billion at the end of June 2010, as first stoping will commence from April 2010.

These results indicate that the project is robust and is likely to be financially viable.

ELANDSFONTEIN PROJECT
SENSITIVITY ANALYSIS - 10% Discount Rate

Parameter	Change in Parameter	Base case	Change in Parameter
PGM Basket Price	-20%	0%	20%
NPV @ 10% (before Tax)	R 75,029	R 795,618	R 1,516,206
NPV @ 10% (after Tax)	-R 69,922	R 464,483	R 987,596
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
Opex	-20%	0%	20%
NPV @ 10% (before Tax)	R 1,143,676	R 795,618	R 447,559

NPV @ 10% (after Tax)	R 718,575	R 464,483	R 207,959
IRR (before Tax)	22.1%	18.9%	15.3%
IRR (after tax)	18.9%	16.0%	12.8%
Capex	-20%	0%	20%
NPV @ 10% (before Tax)	R 1,044,529	R 795,618	R 546,707
NPV @ 10% (after Tax)	R 667,753	R 464,483	R 258,329
IRR (before Tax)	23.4%	18.9%	15.5%
IRR (after tax)	20.0%	16.0%	12.9%
Grade	-20%	0%	20%
NPV @ 10% (before Tax)	R 75,029	R 795,618	R 1,516,206
NPV @ 10% (after Tax)	-R 69,922	R 464,483	R 987,596
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
MCF Change	-3%	95%	3%
NPV @ 10% (before Tax)	R 681,841	R 795,618	R 909,395
NPV @ 10% (after Tax)	R 381,124	R 464,483	R 547,842
IRR (before Tax)	17.8%	18.9%	20.0%
IRR (after tax)	15.0%	16.0%	17.0%

Economic Evaluation - Sensitivity Data at 10% Discount

ELANDSFONTEIN PROJECT
SENSITIVITY ANALYSIS - 15% Discount Rate

Parameter	Change in Parameter	Base case	Change in Parameter
PGM Basket Price	-20%	0%	20%
NPV @ 15% (before Tax)	-R 211,008	R 236,984	R 684,976
NPV @ 15% (after Tax)	-R 286,902	R 53,450	R 383,273
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
Opex	-20%	0%	20%
NPV @ 15% (before Tax)	R 454,782	R 236,984	R 19,186
NPV @ 15% (after Tax)	R 215,165	R 53,450	-R 110,568
IRR (before Tax)	22.1%	18.9%	15.3%
IRR (after tax)	18.9%	16.0%	12.8%
Capex	-20%	0%	20%
NPV @ 15% (before Tax)	R 441,758	R 236,984	R 32,209
NPV @ 15% (after Tax)	R 228,365	R 53,450	-R 124,209
IRR (before Tax)	23.4%	18.9%	15.5%
IRR (after tax)	20.0%	16.0%	12.9%
Grade	-20%	0%	20%
NPV @ 15% (before Tax)	-R 211,008	R 236,984	R 684,976
NPV @ 15% (after Tax)	-R 286,902	R 53,450	R 383,273
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
MCF Change	-3%	95%	3%
NPV @ 15% (before Tax)	R 166,248	R 236,984	R 307,720
NPV @ 15% (after Tax)	R 639	R 53,450	R 106,260
IRR (before Tax)	17.8%	18.9%	20.0%
IRR (after tax)	15.0%	16.0%	17.0%

Economic Evaluation - Sensitivity Data at 15% Discount

PROJECT RISKS and POTENTIAL

The identified risks associated with this project can be summarised as

- Geological risk associated with the grade evaluation
- Geological structure risk associated with the interpretation of the blocks
- Rock Engineering and hanging wall stability
- Geo-hydrological risk associated with the potential for ground water inflows
- Potholes in the mining environment

- Normal risks associated with underground mining
- Mineralogical evaluation has not been implemented
- Metallurgical testwork has not been implemented
- Less beneficial toll treatment terms with the third party refiner, reducing either recovery or increasing charges within the contract

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- Increased penalty payments due to not achieving product quality
- Delayed payments for concentrate dispatched to the smelter
- The stoping width of 20 cm more than the channel width cannot be achieved, thus causing additional dilution
- The Block Factor of 100% is not achieved due to overvaluation of the drilling results
- The Mine Call Factor of 95% (industry norm) is not achieved
- The capital cost estimate accuracy
- The operating cost estimate accuracy
- The project schedule is not achieved
- The production schedule is delayed
- The production ramp-up is not achieved
- Metal price, exchange rate and operating cost volatility
- Country risk
- Political Risk and Mining Charter
- AIDS risk with the impact on the workforce
- Industrial Action
- Environmental, both surface and underground

These project risks are not materially different to those facing any South African platinum project with similar depth and mineralogy.

The up-side potential associated with the project can be summarised as

- Improved information in Blocks 1, 2 and 13 could bring additional resources into account
- The UG2 potential associated with Blocks 6 to 10 could be brought to account with limited additional infrastructure
- The Block Factor of 100% is exceeded
- The project has used a Mine Call factor of 95% whilst some underground operations in the platinum industry report MCF's in excess of 100%
- Stoping width reduction to better than 20 cm more than the channel width
- Improved treatment terms for the processing of the concentrate with the third party refiner
- Improved concentrator recoveries as a result of the high head grades
- Additional tonnage processed through the plant as a result of improved availability and control
- Shallow surface or decline mine for Blocks 1 and 2

- Open casting of the shallow UG2 potential
- Tribute mining potential of neighbouring properties
- Utilisation of existing neighbouring processing facilities to treat the ore
- Chromitite recovery potential of both MR and UG2
- Chromitite recovery potential of other portions of the BIC such the chromitite rich LG1 and LG2 seams
- Rare Earth potential associated with the BIC
- Simpler geological / structural than the current interpretation
- Increased mechanisation in the stoping environment to reduce costs

RECOMMENDATIONS

As a result of the apparent robustness of the project as indicated by the economic evaluation, it is recommended that the following be continued for the Elandsfontein Project area.

1. Geological drilling is to continue to improve the confidence in the geological model

2. Improve the resource from Inferred to Indicated category

3. Review the core currently drilled for geo-technical competence and understand the support requirements for stoping activities and conducting preliminary rock engineering modelling to better understand the in-stope and regional support requirements
4. Obtain larger diameter diamond drill core samples for mineralogical examination and initial metallurgical testing

5. Participate in the currently planned seismic survey to be conducted across the Elandsfontein property

6. Conduct initial Mine Planning and design using Mine24D or equivalent 3D software packages

When the above has been adequately completed, it is proposed that a more detailed Pre-Feasibility study be completed to decide the way forward for the project.

CONCLUSIONS

The Elandsfontein Project can produce at a rate of 1.37 million tonnes per annum for a period of 11 years at full production with a 2-year ramp up phase and a one-year closure phase, i.e. a total of 14 years in production. The total Merensky tonnage to be milled will be 15.7 million tonnes.

The mine will be a stand-alone operation and will not rely on any mining or Concentrator infrastructure from other sources. There is to be a twin shaft system from surface to a depth of 665m below surface. A processing plant to treat 113 000 tonnes per month will be constructed with an associated tailings disposal facility.

The concentrate produced from the processing plant will be toll treated at a smelter in the Rustenburg area, subject to satisfactory negotiations between the parties.

The project is robust and is likely to be economically viable under the current cost and revenue scenarios.

It is recommended that the following be continued for the Elandsfontein Project area.

- Geological drilling is to continue to improve the confidence in the geological model
- Improve the resource from inferred to indicated category
- Review the core currently drilled for geo-technical competence and understand the support requirements for stoping activities and conducting preliminary rock engineering modelling to better understand the in-stope and regional support requirements
- Obtain larger diameter diamond drill core samples for mineralogical examination and initial metallurgical testing
- Participate in the currently planned seismic survey to be conducted across the Elandsfontein property
- Conduct initial Mine Planning and design using Mine24D or equivalent 3D software packages

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ITEM 26 - ILLUSTRATIONS

ADDENDUM (a) - CERTIFICATES

ADDENDUM (b) - SUPPLEMENTS

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PO Box 9026
CENTURION
0046

Professional Geoscience Services

CERTIFICATE of AUTHOR

I, Charles J Muller B Sc. (Hons), Pr.Sci.Nat. do hereby certify that:

1. I am currently employed as Director by:

   Global Geo Services (Pty) Ltd
   PO Box 9026
   CENTURION
   South Africa,
   0046

2. I graduated from the Rand Afrikaanse University (B Sc (1988) and B Sc Hons (1992)).

3. I am a member in good standing of the South African Council for Natural Scientific Professions (SACNASP).

4. I have worked as a geoscientist for a total of 16 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 a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of the report on the "Preliminary Assessment – Scoping Study Report and Resource Update - Western Bushveld Joint Venture – Elandsfontein Project (Project 1), dated August 2005 (the "Report"). I visited the property and viewed the core and discuss the geology of the project with PTM.

7. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. I am independent of the issuer, Platinum Group Metals RSA (Pty) Ltd., applying all of the tests in Section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and any other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 5 August 2005.

Charles Johannes Muller
B Sc (Hons), Pr.Sci.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 by:
   Turnberry Projects (Pty) Ltd PO Box 2199 Rivonia, SANDTON South Africa, 2128
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.
4. I am a member in good standing of the South Africa Institute of Mining and Metallurgy -Membership No. 19584.
5. I have worked as a Metallurgist in production for a total of 20 years since my graduation from university.
6. I have worked as a consulting metallurgist for 5 years since graduation
7. I have been working for Turnberry Projects for 5 years as a Project Engineer and Director, primarily associated with mining and metallurgical projects.
8. 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 a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

the Joint Venture, appl

Dated this 5 August 2005.

Gordon Cunningham
B Eng (Chemical), Pr.Eng.

9. I am responsible for the preparation of the Technical Assessment relating to the Western BIC Project Joint venture property. I have visited the property and viewed the core and discussed the geology of the project with PTM personnel.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Assessment that is not reflected in the Technical Assessment, the omission to disclose which makes the Technical Assessment misleading.
11. I am independent of the issuer, Platinum Group Metals RSA (Pty) Ltd. or any member of applying all of the tests in Section 1.5 of National Instrument 43-101.

North Building, 272 Kent Avenue, Ferndale, Randburg, South Africa. Email: turnbery@iafrica.com

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Director: G.I.Cunningham

Turnberry Projects
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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
   South Africa,
   2128

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.

4. I am a member in good standing of the South Africa Institute of Mining and Metallurgy.

5. I have worked as a Mining Engineer in production for a total of 20 years prior to and since my graduation from university.

6. I have worked as a consulting mining engineer for 12 years since graduation

7. I have been associated with Turnberry Projects for 4 years as a Principal Mining Engineer.

8. 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 a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

9. I am responsible for the preparation of the Technical Assessment relating to the Western BIC Project Joint venture property. I have visited the property and viewed the core and discussed the geology of the project with PTM personnel.

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Assessment that is not reflected in the Technical Assessment, the omission to disclose which makes the Technical Assessment misleading.

11. I am independent of the issuer, Platinum Group Metals RSA (Pty) Ltd. or any member of the Joint Venture, applying all of the tests in Section 1.5 of National Instrument 43-101.

Dated this 5 August 2005.

Tim Spindler
B Sc (Mining), Pr.Eng.

North Building, 272 Kent Avenue, Ferndale, Randburg, South Africa. Email: turnbery@iafrica.com

Merensky Reef

Intersection No.	Borehole No.	Defl.	New	Reef	Comment	Facies	BRC (m)	Interval (m)	Pt (ppb)	Pd (ppb)	Rh (ppb)	Au (ppb)		4E (g/t)	4E (cmgt)	X Coordinate UTM	Y Coordinate UTM
1	ELN01	D0		MR	Pass	Htz	491.59	1.00						8.08	808	9866.310	-2812667.580
2	ELN01	D3		MR	Pass	Htz	491.58	1.00						12.82	1282	9866.310	-2812667.580
3	ELN06	D2	New	MR	Pass	Htz	400.57	1.45	1314	3873	1783	272	7.24	7.24	1047	9219.720	-2812880.870
4	ELN12	D0	New	MR	Pass	Htz	334.81	1.00	2469	1979	188	54	4.69	4.69	469	8892.174	-2813404.000
5	ELN12	D1		MR	Pass	Htz	334.71	2.03						7.93	1611	8892.174	-2813404.000
6	ELN12	D2		MR	Pass	Htz	334.61	2.14						11.51	2464	8892.174	-2813404.000
7	ELN15	D0	New	MR	Pass	Htz	432.80	1.00	363	172	34	8	0.58	0.58	58	8367.607	-2812800.541
8	FG02	D0		MR	Pass	Htz	521.79	2.46						13.57	3338	9268.620	-2812035.890
9	FG02	D2		MR	Pass	Htz	522.00	2.50						18.69	4673	9268.620	-2812035.890
10	FG29	D0		MR	Pass	CR	468.10	1.13	1585	841	82	182	2.69	2.69	304	9749.469	-2812950.226
11	FG29	D1		MR	Pass	CR	469.40	1.77	1219	666	43	172	2.10	2.10	372	9749.469	-2812950.226
12	FG30	D0		MR	Pass	CR	506.00	1.02	2781	1586	167	272	4.81	4.81	491	8622.793	-2811718.804
13	FG30	D3		MR	Pass	CR	505.60	1.00	4871	2506	257	448	8.08	8.08	808	8622.793	-2811718.804
14	FG31	D0		MR	Pass	FPP	336.50	1.25	4691	2148	224	552	7.62	7.61	951	7781.225	-2811039.799
15	FG33	D0		MR	Pass	Pxnt	395.16	1.32	2234	912	88	312	3.55	3.55	469	9664.449	-2813583.619
16	FG33	D6		MR	Pass	Pxnt	395.60	1.00	2972	1328	174	289	4.76	4.76	476	9664.449	-2813583.619
17	FG34	D7		MR	Pass	FPP	890.20	1.23	4111	1789	299	325	6.52	6.52	848	6890.892	-2809868.915
18	FG34	D9		MR	Pass	FPP	889.70	1.07	9555	2881	971	97	13.50	13.50	1464	6890.892	-2809868.915
19	WBJV01	D0	New	MR	Pass	FPP	448.65	1.00	2854	1307	-	171	4.33	4.33	433	8855.941	-2812165.626
20	WBJV01	D2	New	MR	Pass	FPP	442.14	1.00	3058	1392	-	276	4.73	4.73	473	8855.941	-2812165.626
21	WBJV02	D0	New	MR	Pass	FPP	465.74	1.00	4054	1948	-	406	6.41	6.41	641	8573.092	-2812564.088
22	WBJV02	D1	New	MR	Pass	FPP	459.84	1.00	2029	1252	-	180	3.46	3.46	346	8573.092	-2812564.088
23	WBJV06	D0	New	MR	Pass	Htz	460.98	1.00	10051	4525	-	448	15.02	15.02	1502	8608.128	-2813259.832
24	WBJV06	D1	New	MR	Pass	Htz	457.37	1.00	8483	4252	-	555	13.29	13.29	1329	8608.128	-2813259.832
25	WBJV08	D0	New	MR	Pass	Pxnt	243.98	1.00	1472	691	-	127	2.29	2.29	229	8072.823	-2813337.816
26	WBJV08	D1	New	MR	Pass	Pxnt	240.67	1.00	2310	1299	32	547	4.19	4.19	419	8072.823	-2813337.816
27	WBJV09	D1	New	MR	Pass	Htz	265.05	1.00	963	502	9	27	1.50	1.50	150	5733.700	-2811297.983
28	WBJV10	D1	New	MR	Pass	FPP	422.81	1.00	1312	523	74	143	2.05	2.05	205	9358.783	-2813598.879
29	WBJV12	D0	New	MR	Pass	Pxnt	65.22	1.00	340	143	0	2	0.49	0.49	49	7999.966	-2814091.185
30	WBJV14	D0	New	MR	Pass	Pxnt	235.65	1.00	926	328	22	73	1.35	1.35	135	8511.950	-2813841.210
31	WBJV14	D1	New	MR	Pass	Pxnt	238.59	1.00	239	100	30	3	0.37	0.37	37	8511.950	-2813841.210
32	WBJV15	D0	New	MR	Pass	FPP	390.73	1.00	6612	2403	300	368	9.68	9.68	968	9320.907	-2813217.668
33	WBJV15	D1	New	MR	Pass	FPP	392.05	1.00	2943	1238	142	195	4.52	4.52	452	9320.907	-2813217.668
34	WBJV16	D0	New	MR	Pass	Pxnt	118.06	1.17	605	330	22	108	1.07	1.07	125	7768.690	-2813571.546
35	WBJV16	D1	New	MR	Pass	Pxnt	117.71	1.00	309	153	11	92	0.57	0.57	57	7768.690	-2813571.546
36	WBJV17	D0	New	MR	Pass	CR	78.15	1.00	26	11	1	3	0.04	0.04	4	8151.766	-2813894.581
37	WBJV17	D1	New	MR	Pass	CR	77.65	1.00	29	10	1	1	0.04	0.04	4	8151.766	-2813894.581
38	WBJV18	D0	New	MR	Pass	Pxnt	231.49	1.11	378	237	18	132	0.77	0.77	85	8761.791	-2813921.102
39	WBJV18	D1	New	MR	Pass	Pxnt	232.12	1.00	6799	2996	242	665	10.70	10.70	1070	8761.791	-2813921.102

Coordinates in WGS84, Hartebeeshoek datum and UTM 35S

Entry:	Explanation
-	**Designates No Value**
Pass	**QAQC**
SNV	**Sampled but no value Return-Lost**
Insufficient Sample	**Not enough material to accuratley assay**
Faulted	**Stratigraphy eliminated**
Rejected	**Core Loss or Core Mixed**
Not Recognized	**Lithologies/stratigraphy not recognised**
Not Drilled	**Deflection drilled for UG2**
Not Sampled	**Core not Sampled at all**
Beyond Subcrop	**Borehole position beyond possible intersection of reef**
Stopped Short	**Borehole stopped above the reef horison**
Not located	**Core could not be traced or found**

Facies:	Explanation
Htz	Harzburgitic Facies
Pxnt	Pyroxenite Facies
FPP	Pegmatoidal Feldspathic Pyroxenite Facies
CR	Contact Facies

Wrong Stratigraphic	**TOW of deflection in wrong position to intersect reef**
To be sampled	**In the process of completion, will be sampled**
Awaiting Assay	**Sampled but awaiting assay return from Lab**
Disturbed	**Lithology/stratigraphy not recognised but still useful for markers**
Dyked	**Reef eliminated/brecciated by dyke**
Drilling	**Borehole/deflection in progress**
To be drilled	**Deflection still to be drilled, machine on site**
Sited	**Borehole position laid out on ground, drill rig moving to site**

TABLE 1a

UG2 Reef

Intersection No.	Borehole No.	Defl.	New	Reef	Comment	BRC (m)	Interval (m)	Pt (ppb)	Pd (ppb)	Rh (ppb)	Au (ppb)		4E (g/t)	4E (cmgt)	X Coordinate UTM	Y Coordinate UTM
1	ELN01	D0		UG2	Pass	543.48	1.83						2.81	514	9866.310	-2812667.580
2	ELN01	D1		UG2	Pass	543.47	1.80	-	-	-	-		2.43	437	9866.310	-2812667.580
3	ELN01	D2		UG2	Pass	542.95	1.68	-	-	-	-		2.71	455	9866.310	-2812667.580
4	ELN05	D1		UG2	Pass	532.61	1.80	2106	691	298	12		3.11	560	8070.813	-2813335.600
5	FG02	D0		UG2	Pass	593.06	1.70						0.20	34	9268.620	-2812035.890
6	FG02	D1		UG2	Pass	592.94	1.80	-	-	-	-		0.79	142	9268.620	-2812035.890
7	FG29	D0		UG2	Pass	543.84	0.64	1412	410	143	20	1.99	1.99	127	9749.469	-2812950.226
8	FG30	D0		UG2	Pass	559.67	0.89	2284	891	354	23	3.55	3.55	316	8622.793	-2811718.804
9	FG30	D1		UG2	Pass	559.65	0.96	2372	999	373	25	3.77	3.77	362	8622.793	-2811718.804
10	FG30	D2		UG2	Pass	560.05	1.10	1963	496	197	12		2.67	294	8622.793	-2811718.804
11	WBJV01	D0	New	UG2	Pass	474.70	2.20	541	187	-	0	0.73	0.73	161	8855.941	-2812165.626
12	WBJV01	D1	New	UG2	Pass	475.86	3.66	260	109	-	6	0.38	0.38	139	8855.941	-2812165.626
13	WBJV01	D2	New	UG2	Pass	468.68	1.93	505	180	-	1	0.69	0.69	133	8855.941	-2812165.626
14	WBJV02	D0	New	UG2	Pass	557.62	1.70	2029	728	-	13	2.77	2.77	471	8573.092	-2812564.088
15	WBJV02	D1	New	UG2	Pass	550.00	1.00	2123	747	-	10	2.88	2.88	288	8573.092	-2812564.088
16	WBJV02	D2	New	UG2	Pass	554.10	1.19	2224	730	-	3	2.96	2.96	352	8573.092	-2812564.088
17	WBJV03	D0	New	UG2	Pass	537.68	0.84	3167	1067	-	26	4.26	4.26	358	9215.668	-2812492.532
18	WBJV03	D1	New	UG2	Pass	538.59	1.48	1973	1165	-	30	3.17	3.17	469	9215.668	-2812492.532
19	WBJV03	D2	New	UG2	Pass	557.32	0.64	587	170	-	1	0.76	0.76	49	9215.668	-2812492.532
20	WBJV05	D0	New	UG2	Pass	486.03	2.14	507	159	-	0	0.67	0.67	143	8309.380	-2812942.636
21	WBJV06	D0	New	UG2	Pass	477.84	0.62	473	50	-	0	0.52	0.52	32	8608.128	-2813259.832
22	WBJV07	D0	New	UG2	Pass	256.78	0.92	2514	728	253	28	3.52	3.52	324	8322.021	-2813639.167
23	WBJV08	D0	New	UG2	Pass	352.31	0.29	800	469	-	10	1.28	1.28	37	8072.823	-2813337.816
24	WBJV08	D1	New	UG2	Pass	323.86	1.15	1545	643	192	15	2.40	2.40	276	8072.823	-2813337.816
25	WBJV09	D0	New	UG2	Pass	280.36	0.43	360	111	68	1	0.54	0.54	23	5733.700	-2811297.983
26	WBJV09	D3	New	UG2	Pass	281.05	0.91	522	198	92	4	0.82	0.81	74	5733.700	-2811297.983
27	WBJV10	D0	New	UG2	Pass	457.19	1.48	463	126	70	5	0.66	0.66	98	9358.783	-2813598.879
28	WBJV10	D1	New	UG2	Pass	456.46	1.96	495	247	108	13	0.86	0.86	169	9358.783	-2813598.879
29	WBJV12	D0	New	UG2	Pass	70.71	0.61	340	143	0	2	0.49	0.18	11	7999.966	-2814091.185
30	WBJV13	D0	New	UG2	Pass	475.20	4.91	377	133	94	2	0.61	0.61	300	9160.170	-2813418.072
31	WBJV13	D1	New	UG2	Pass	471.21	1.10	258	68	64	7	0.40	0.40	44	9160.170	-2813418.072
32	WBJV14	D0	New	UG2	Pass	248.36	0.78	389	111	96	1	0.60	0.60	47	8511.950	-2813841.210
33	WBJV14	D1	New	UG2	Pass	248.17	0.26	409	106	80	19	0.61	0.61	16	8511.950	-2813841.210
34	WBJV15	D0	New	UG2	Pass	435.24	1.27	2636	1115	349	36	4.14	4.14	526	9320.907	-2813217.668
35	WBJV15	D1	New	UG2	Pass	438.31	1.18	2976	979	360	17	4.33	4.33	511	9320.907	-2813217.668
36	WBJV16	D0	New	UG2	Pass	134.41	0.96	3403	1256	430	41	5.13	5.13	492	7768.690	-2813571.546
37	WBJV16	D1	New	UG2	Pass	117.71	1.36	2190	568	294	17	3.07	3.07	418	7768.690	-2813571.546
38	WBJV18	D0	New	UG2	Pass	245.61	2.08	2100	1041	307	24	3.47	3.47	722	8761.791	-2813921.102
39	WBJV18	D1	New	UG2	Pass	246.72	1.08	1900	803	303	20	3.03	3.03	327	8761.791	-2813921.102

Entry:	Explanation
-	**Designates No Value**
	QAQC

Coordinates in WGS84, Hartebeeshoek

Pass	
SNV	**Sampled but no value Return-Lost**
Insufficient Sample	**Not enough material to accuratley assay**
Faulted	**Stratigraphy eliminated**
Rejected	**Core Loss or Core Mixed**
Not Recognized	**Lithologies/stratigraphy not recognised**
Not Drilled	**Deflection drilled for UG2**
Not Sampled	**Core not Sampled at all**
Beyond Subcrop	**Borehole position beyond possible intersection of reef**
Stopped Short	**Borehole stopped above the reef horison**
Not located	**Core could not be traced or found**
Wrong Stratigraphic	**TOW of deflection in wrong position to intersect reef**
To be sampled	**In the process of completion, will be sampled**
Awaiting Assay	**Sampled but awaiting assay return from Lab**
Disturbed	**Lithology/stratigraphy not recognised but still useful for markers**
Dyked	**Reef eliminated/brecciated by dyke**
Drilling	**Borehole/deflection in progress**
To be drilled	**Deflection still to be drilled, machine on site**
Sited	**Borehole position laid out on ground, drill rig moving to site**

TABLE 1b

Table 2: Merensky and UG2 Intersections used in estimation

BHID	Channel Width	3PGE&Au	
	m	g/t	cmg/t
	Merensky Reef Domain 1		
ELN01	0.69	15.30	1048
ELN06	1.45	7.24	1047
ELN12	1.52	9.29	1411
FG02	2.45	16.15	3965
FG29	1.43	2.33	333
FG30	0.79	7.87	625
WBJV001	0.41	10.93	451
WBJV002	0.65	7.33	475
WBJV006	1.05	14.49	1526
WBJV015	1.03	6.93	711
	Merensky Reef Domain 2		
ELN15	0.27	0.58	16
FG33	0.95	4.82	458
WBJV005	0.43	1.86	79
WBJV007	0.19	1.15	22
WBJV008	0.19	7.93	151
WBJV010	0.66	0.76	50
WBJV011	0.47	0.09	4
WBJV012	0.18	2.32	42
WBJV013	0.19	1.44	28
WBJV014	0.24	0.96	23
WBJV016	1.00	0.90	90
WBJV017	0.17	0.11	2
WBJV018	1.03	5.58	575
WBJV022	0.02	0.76	2
	UG2		
ELN01	1.75	2.65	464
FG02	1.71	0.50	86
FG07	1.59	5.78	921
FG29	0.64	1.99	126
FG30	0.92	3.66	336
WBJV001	3.31	0.51	170
WBJV002	1.28	3.18	406

WBJV003	0.96	3.26	314
WBJV005	2.12	0.74	158
WBJV006	0.89	0.52	46
WBJV007	0.89	0.96	85
WBJV008	0.78	2.05	159
WBJV009	1.26	0.80	101
WBJV010	1.71	0.77	132
WBJV011	0.77	0.67	51
WBJV012	0.87	0.01	1
WBJV013	3.00	0.58	173
WBJV014	0.61	0.52	32
WBJV015	1.22	4.23	516
WBJV016	1.15	3.92	453
WBJV018	1.10	4.61	507
WBJV019	0.84		
WBJV020	0.73		
WBJV021	0.87		
WBJV022	1.21	0.18	21
WBJV023	2.46		
WBJV025	1.59		
WBJV028	2.47		

TABLE 2

Inferred Mineral Resource (diluted to 1m minimum mining width)

Cut-Off g/t	Tonnage	Av 3PGE&Au Grade	Av Channel Width	Av Mining Width (1m minimum)	Metal Content 3PGE&Au	Metal
cm g/t	t	g/t	m	m	g	Moz
		Merensky Reef - Domain 1				
0	13,870,586	9.67	1.11	1.12	134,112,425	4.312
200	13,869,781	9.67	1.11	1.12	134,111,228	4.312
400	13,671,466	9.77	1.11	1.12	133,509,878	4.292
500	13,203,917	9.97	1.11	1.12	131,634,208	4.232
600	12,363,873	10.31	1.11	1.12	127,522,342	4.100
700	11,195,722	10.79	1.11	1.12	120,763,773	3.883
1000	6,978,111	12.73	1.11	1.12	88,808,675	2.855
		Merensky Reef - Domain 2				
0	15,474,713	1.06	0.42	1.00	16,383,388	0.527
200	1,991,262	3.73	0.42	1.00	7,423,431	0.239
400	534,406	6.47	0.42	1.00	3,454,966	0.111
500	321,585	7.80	0.42	1.00	2,508,726	0.081
600	206,025	9.12	0.42	1.00	1,878,574	0.060
700	138,019	10.43	0.42	1.00	1,439,376	0.046
1000	50,502	14.30	0.42	1.00	722,368	0.023
		UG2 Reef Domain 1				
0	28,227,481	1.48	1.35	1.35	41,749,715	1.342
200	10,353,612	2.51	1.35	1.35	26,023,949	0.837
400	2,212,977	4.32	1.35	1.35	9,568,189	0.308
500	1,113,588	5.27	1.35	1.35	5,869,863	0.189
600	591,167	6.23	1.35	1.35	3,683,004	0.118

700	328,570	7.20	1.35	1.35	2,364,131	0.076
1000	69,796	10.11	1.35	1.35	705,429	0.023

TABLE 3

Descriptive statistics for the Merensky reef intersections

Variable	Descriptive Statistics (Spreadsheet1)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM1ALL_MR_CW	10	1.147	0.4121	2.455	0	0.593	1.114805	1.614057
DOM1ALL_MR_Au	10	9.787	2.3348	16.151	19	4.402	0.088974	-0.680499
DOM1ALL_MR_CMGT	10	1159.325	333.2103	3964.729	1136901	1066.256	2.367082	6.302458

Variable	Descriptive Statistics (Spreadsheet3)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM2ALL_MR_CW	14	0.4271	0.019807	1.0296	0.12	0.3445	0.901821	-0.685947
DOM2ALL_MR_Au	14	2.0906	0.090409	7.9333	5.51	2.3469	1.625291	1.873528
DOM2ALL_MR_CMGT	14	110.0872	1.505314	574.7434	31844.68	178.4508	2.138561	3.633475

TABLE 4

Descriptive statistics for the UG2 reef intersections

Variable	Descriptive Statistics (Spreadsheet5)							
	Valid N	Mean	Minimum	Maximum	Variance	Std.Dev.	Skewness	Kurtosis
DOM0ALL_UG2_CW	28	1.3818	0.606082	3.3109	0.51	0.7169	1.294066	1.059923
DOM0ALL_UG2_Au	22	1.9131	0.011954	5.7829	2.92	1.7085	0.793769	-0.604609
DOM0ALL_UG2_CMGT	22	239.0512	1.039970	921.3384	51693.51	227.3621	1.447032	2.335884

TABLE 5

Variogram parameters

									Structure 1				Structure 2			
Reef	Domain	Angle 1	Angle 2	Angle 3	Axis 1	Axis 2	Axis 3	Nugget %	Sill 1 %	Range 1	Range 2	Range 3	Sill 2 %	Range 1	Range 2	Range 3
MR	1	0	0	0	0	0	0	25.25	80	245	245	1	100	510	510	1
MR	2	0	0	0	0	0	0	24.55	80	247	247	1	100	533	533	1
UG2	1	0	0	0	0	0	0	25	80	248	248	1	100	520	520	1

TABLE 6

SIMPLIFIED FLOW OF ORE PARAMETERS (eg taken from Tables 11 and 12 for 2009/10)

	PARAMETERS	TONS	TONS % Increase	GRADE g/t	% g/t Loss	CONTENT Grams
Insitu Tons and Content						
Channel Width (cm)	1.06					
Average Grade (g/t)	9.2					
SG	3.2					
Reef Tons		46,276	100%	9.80	100%	444,40
Diluted Tons and Content (+20cm)	0.2	8,724				
Stoping Width (cm)	1.26					
SG	3.2					
Reef Tons due to 20cm dilution		55,000	119%	8.08	84%	444,40
Dilution from Development (+10%)						
Waste Dilution	10%					
Waste Tonnage to Mill		60,500	131%	7.35	76%	444,40
Dilution from Over-break (+4%)	4%					
Waste Tonnage to Mill		62,700	135%	7.09	74%	444,40
Dilution from Shortfall (+1%)	1%					
		63,250	137%	7.03		444,40
Mine Call Factor	95%	63,250	137%	6.68	70%	422,64
Metallurgical Factors						
Recovery on Concentrator	87%	63,250	137%	5.81	61%	367,69
Recovery on Smelter	86%	63,250	137%	5.00	52%	316,20
Total Metallurgical Recovery	75%	63,250	137%	5.00	52%	316,20

TABLE 7

Elandsfontein Project Summary of Ore Flow Calculations

		Tonnage tonnes	Kilograms 4E's	Grade 4E's g/tonne
1	Total Resource at 0 cmg/t cut-off	29,283,401	143,921.5	4.915
2	Total Resource at economic cut-off grade	11,914,640	110,668.3	9.288
3	Total Resource excluding un-economic blocks	11,557,349	108,394.3	9.379
4	Economic Resource corrected for SW	13,671,775	108,394.3	7.928
5	Delivered to Mill with Mining Factors included	15,722,542	102,976.2	6.550
6	Concentrate Recovered based on Tons Milled	15,722,542	89,670.6	5.703
7	Smelter Recovered based on Tons Milled	15,722,542	77,116.7	4.905

Based on the Inferred Resource for the Merensky Domain 1 zone only

TABLE 7a

MR DILUTED MBLK1

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	Mining Width
mr_blk1_m	07-28-2005	0	5,646,621	0.94	5,069,159	0.171	1.00
mr_blk1_m	07-28-2005	200	605,162	3.65	2,111,599	0.071	1.00
mr_blk1_m	07-28-2005	400	154,971	6.36	942,050	0.032	1.00
mr_blk1_m	07-28-2005	500	91,623	7.68	672,787	0.023	1.00
mr_blk1_m	07-28-2005	600	57,785	8.98	496,367	0.017	1.00
mr_blk1_m	07-28-2005	700	38,189	10.28	375,361	0.013	1.00
mr_blk1_m	07-28-2005	1000	13,489	14.11	182,044	0.006	1.00

MR DILUTED MBLK 2

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk2_m	07-28-2005	0	1,463,903	1.08	1,515,734	0.051	1.00
mr_blk2_m	07-28-2005	200	197,677	3.78	714,518	0.024	1.00
mr_blk2_m	07-28-2005	400	54,815	6.51	341,466	0.011	1.00
mr_blk2_m	07-28-2005	500	33,295	7.85	249,942	0.008	1.00
mr_blk2_m	07-28-2005	600	21,437	9.18	188,096	0.006	1.00
mr_blk2_m	07-28-2005	700	14,451	10.49	144,951	0.005	1.00
mr_blk2_m	07-28-2005	1000	5,354	14.38	73,640	0.002	1.00

MR DILUTED MBLK3

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk3_m	07-28-2005	0	1,513,090	4.20	6,077,651	0.204	1.03
mr_blk3_m	07-28-2005	200	637,179	9.09	5,539,150	0.186	1.03
mr_blk3_m	07-28-2005	400	523,117	10.47	5,237,223	0.176	1.03
mr_blk3_m	07-28-2005	500	497,236	10.79	5,128,997	0.172	1.03
mr_blk3_m	07-28-2005	600	470,811	11.10	4,997,927	0.168	1.03
mr_blk3_m	07-28-2005	700	437,765	11.49	4,808,888	0.162	1.03
mr_blk3_m	07-28-2005	1000	301,841	13.18	3,804,180	0.128	1.03

MR DILUTED MBLK 4

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk4_m	07-28-2005	0	1,615,399	8.02	12,385,210	0.417	1.10
mr_blk4_m	07-28-2005	200	1,244,345	10.21	12,148,930	0.408	1.10
mr_blk4_m	07-28-2005	400	1,204,661	10.45	12,044,100	0.405	1.10
mr_blk4_m	07-28-2005	500	1,183,249	10.57	11,958,910	0.402	1.10
mr_blk4_m	07-28-2005	600	1,140,716	10.78	11,757,670	0.395	1.10
mr_blk4_m	07-28-2005	700	1,070,344	11.11	11,369,090	0.382	1.10
mr_blk4_m	07-28-2005	1000	740,248	12.70	8,993,035	0.302	1.10

MR DILUTED MBLK 5

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk5_m	07-28-2005	0	1,327,478	4.26	5,406,096	0.182	1.06
mr_blk5_m	07-28-2005	200	603,359	8.54	4,927,464	0.166	1.06
mr_blk5_m	07-28-2005	400	518,139	9.49	4,704,754	0.158	1.06
mr_blk5_m	07-28-2005	500	496,746	9.72	4,618,569	0.155	1.06
mr_blk5_m	07-28-2005	600	468,754	10.01	4,487,366	0.151	1.06
mr_blk5_m	07-28-2005	700	429,948	10.40	4,277,608	0.144	1.06
mr_blk5_m	07-28-2005	1000	275,443	12.14	3,197,301	0.108	1.06

MR DILUTED MBLK 6

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk6_m	07-28-2005	0	2,501,556	4.97	11,893,070	0.400	1.09
mr_blk6_m	07-28-2005	200	1,634,276	7.22	11,283,990	0.379	1.09
mr_blk6_m	07-28-2005	400	1,450,616	7.78	10,791,090	0.363	1.09
mr_blk6_m	07-28-2005	500	1,318,398	8.16	10,289,230	0.346	1.09
mr_blk6_m	07-28-2005	600	1,141,007	8.69	9,480,744	0.319	1.09
mr_blk6_m	07-28-2005	700	943,622	9.33	8,423,607	0.283	1.09
mr_blk6_m	07-28-2005	1000	444,268	11.69	4,967,937	0.167	1.09

MR DILUTED MBLK 7

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk7_m	07-28-2005	0	1,869,812	8.70	15,559,280	0.523	1.12
mr_blk7_m	07-28-2005	200	1,695,442	9.52	15,435,610	0.519	1.12
mr_blk7_m	07-28-2005	400	1,640,423	9.74	15,286,030	0.514	1.12
mr_blk7_m	07-28-2005	500	1,576,032	9.98	15,042,760	0.506	1.12
mr_blk7_m	07-28-2005	600	1,472,645	10.34	14,566,630	0.490	1.12
mr_blk7_m	07-28-2005	700	1,335,106	10.82	13,816,280	0.464	1.12
mr_blk7_m	07-28-2005	1000	844,302	12.75	10,294,040	0.346	1.12

MR DILUTED MBLK 8

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk8_m	07-28-2005	0	2,088,127	9.23	18,433,920	0.620	1.11
mr_blk8_m	07-28-2005	200	2,087,908	9.23	18,433,590	0.620	1.11
mr_blk8_m	07-28-2005	400	2,039,679	9.38	18,293,170	0.615	1.11
mr_blk8_m	07-28-2005	500	1,938,615	9.66	17,902,350	0.602	1.11
mr_blk8_m	07-28-2005	600	1,775,951	10.09	17,132,790	0.576	1.11
mr_blk8_m	07-28-2005	700	1,570,455	10.64	15,981,070	0.537	1.11
mr_blk8_m	07-28-2005	1000	921,051	12.75	11,226,080	0.378	1.11

MR DILUTED MBLK 9

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk9_m	07-28-2005	0	729,839	10.95	7,639,385	0.257	1.16
mr_blk9_m	07-28-2005	200	729,839	10.95	7,639,385	0.257	1.16
mr_blk9_m	07-28-2005	400	727,818	10.97	7,633,373	0.257	1.16
mr_blk9_m	07-28-2005	500	719,744	11.05	7,602,586	0.256	1.16
mr_blk9_m	07-28-2005	600	699,727	11.22	7,510,406	0.252	1.16
mr_blk9_m	07-28-2005	700	664,348	11.52	7,319,129	0.246	1.16
mr_blk9_m	07-28-2005	1000	486,240	13.02	6,055,113	0.204	1.16

MR DILUTED MBLK 10

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk10_m	07-28-2005	0	1,357,299	9.48	12,301,320	0.414	1.08
mr_blk10_m	07-28-2005	200	1,357,226	9.48	12,301,210	0.414	1.08
mr_blk10_m	07-28-2005	400	1,335,206	9.58	12,235,270	0.411	1.08
mr_blk10_m	07-28-2005	500	1,281,135	9.81	12,020,290	0.404	1.08
mr_blk10_m	07-28-2005	600	1,183,713	10.20	11,546,590	0.388	1.08
mr_blk10_m	07-28-2005	700	1,050,746	10.73	10,781,400	0.362	1.08
mr_blk10_m	07-28-2005	1000	603,670	12.85	7,420,789	0.249	1.08

MR DILUTED MBLK 11

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk11_m	07-28-2005	0	3,655,743	10.19	35,631,340	1.198	1.09
mr_blk11_m	07-28-2005	200	3,655,743	10.19	35,631,340	1.198	1.09
mr_blk11_m	07-28-2005	400	3,630,950	10.24	35,553,100	1.195	1.09
mr_blk11_m	07-28-2005	500	3,547,074	10.38	35,212,570	1.184	1.09
mr_blk11_m	07-28-2005	600	3,365,831	10.66	34,323,710	1.154	1.09
mr_blk11_m	07-28-2005	700	3,083,892	11.09	32,700,900	1.099	1.09
mr_blk11_m	07-28-2005	1000	1,954,328	12.97	24,240,710	0.815	1.09

MR DILUTED MBLK 12

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk12_m	07-28-2005	0	1,122,635	6.88	7,387,521	0.248	1.07
mr_blk12_m	07-28-2005	200	761,182	9.82	7,148,066	0.240	1.07
mr_blk12_m	07-28-2005	400	715,689	10.27	7,027,991	0.236	1.07
mr_blk12_m	07-28-2005	500	696,592	10.43	6,951,293	0.234	1.07
mr_blk12_m	07-28-2005	600	663,861	10.70	6,794,920	0.228	1.07
mr_blk12_m	07-28-2005	700	614,130	11.10	6,517,123	0.219	1.07
mr_blk12_m	07-28-2005	1000	405,682	12.88	4,996,965	0.168	1.07

MR DILUTED MBLK 13

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz	
mr_blk13_m	07-28-2005	0	4,391,898	1.10	4,621,795	0.155	1.00
mr_blk13_m	07-28-2005	200	568,640	3.62	2,036,234	0.068	1.00
mr_blk13_m	07-28-2005	400	147,505	6.32	891,083	0.030	1.00
mr_blk13_m	07-28-2005	500	86,458	7.64	631,651	0.021	1.00
mr_blk13_m	07-28-2005	600	54,271	8.94	463,869	0.016	1.00
mr_blk13_m	07-28-2005	700	35,574	10.24	348,427	0.012	1.00
mr_blk13_m	07-28-2005	1000	12,423	14.08	167,308	0.006	1.00

UG2 DILUTED MBLK 1

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz
ug2_blk1_m	07-28-2005	0	4,164,097	1.27	5,042,289	0.170
ug2_blk1_m	07-28-2005	200	1,126,399	2.34	2,528,405	0.085
ug2_blk1_m	07-28-2005	400	182,351	3.96	690,397	0.023
ug2_blk1_m	07-28-2005	500	80,466	4.78	367,568	0.012
ug2_blk1_m	07-28-2005	600	37,629	5.59	201,294	0.007
ug2_blk1_m	07-28-2005	700	18,526	6.41	113,559	0.004
ug2_blk1_m	07-28-2005	1000	2,806	8.85	23,737	0.001

UG2 DILUTED MBLK 2

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz
ug2_blk2_m	07-28-2005	0	1,775,184	1.42	2,414,173	0.081
ug2_blk2_m	07-28-2005	200	507,730	2.51	1,216,342	0.041
ug2_blk2_m	07-28-2005	400	77,713	4.22	313,248	0.011
ug2_blk2_m	07-28-2005	500	34,118	5.07	165,417	0.006
ug2_blk2_m	07-28-2005	600	16,020	5.92	90,744	0.003
ug2_blk2_m	07-28-2005	700	7,997	6.77	51,768	0.002
ug2_blk2_m	07-28-2005	1000	1,280	9.30	11,391	0.000

UG2 DILUTED MBLK 3

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz
ug2_blk3_m	07-28-2005	0	1,899,486	1.37	2,482,810	0.084
ug2_blk3_m	07-28-2005	200	569,880	2.47	1,344,806	0.045
ug2_blk3_m	07-28-2005	400	107,820	4.17	429,852	0.014
ug2_blk3_m	07-28-2005	500	51,236	5.03	246,560	0.008
ug2_blk3_m	07-28-2005	600	25,609	5.90	144,412	0.005
ug2_blk3_m	07-28-2005	700	13,367	6.76	86,434	0.003
ug2_blk3_m	07-28-2005	1000	2,334	9.35	20,876	0.001

UG2 DILUTED MBLK 4

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz
ug2_blk4_m	07-28-2005	0	2,454,088	1.25	2,936,874	0.099
ug2_blk4_m	07-28-2005	200	754,808	2.17	1,567,417	0.053
ug2_blk4_m	07-28-2005	400	114,738	3.71	407,610	0.014
ug2_blk4_m	07-28-2005	500	49,105	4.51	211,562	0.007
ug2_blk4_m	07-28-2005	600	22,416	5.30	113,567	0.004
ug2_blk4_m	07-28-2005	700	10,808	6.09	62,987	0.002
ug2_blk4_m	07-28-2005	1000	1,603	8.47	12,988	0.000

UG2 DILUTED MBLK 5

File	Date	Cut-Off	Tonnage	Av grade(g/t)	Metal Content	Moz
ug2_blk5_m	07-28-2005	0	2,206,866	1.31	2,768,486	0.093
ug2_blk5_m	07-28-2005	200	808,876	2.14	1,653,116	0.056
ug2_blk5_m	07-28-2005	400	141,518	3.61	486,041	0.016
ug2_blk5_m	07-28-2005	500	63,311	4.37	264,342	0.009
ug2_blk5_m	07-28-2005	600	29,855	5.13	146,544	0.005
ug2_blk5_m	07-28-2005	700	14,767	5.91	83,406	0.003
ug2_blk5_m	07-28-2005	1000	2,264	8.26	17,880	0.001

ELANDSFONTEIN PROJECT - Mining Blocks from Resource Mode

File	Cut-Off	Tonnage	Av grade(g/t)	Channel Width	Possible Tons (70%)	Mineable Tons	Grade g/t	Conte
mr_blk1_m	400	154,971	6.360	1.000	154,971	-	6.360	
mr_blk2_m	400	54,815	6.510	1.000	54,815	-	6.510	
mr_blk3_m	200	637,179	9.090	1.034	637,179	637,179	9.090	5
mr_blk4_m	200	1,244,345	10.210	1.103	1,130,779	1,130,779	10.210	11
mr_blk5_m	200	603,359	8.540	1.058	603,359	603,359	8.540	5
mr_blk6_m	200	1,634,276	7.220	1.087	1,634,276	1,634,276	7.220	11
mr_blk7_m	200	1,695,442	9.520	1.118	1,308,868	1,308,868	9.520	12
mr_blk8_m	200	2,087,908	9.230	1.109	1,461,689	1,461,689	9.230	13
mr_blk9_m	200	729,839	10.945	1.163	510,887	510,887	10.945	5
mr_blk10_m	200	1,357,226	9.478	1.082	950,110	950,110	9.478	9
mr_blk11_m	200	3,655,743	10.192	1.093	2,559,020	2,559,020	10.192	26
mr_blk12_m	200	761,182	9.820	1.074	761,182	761,182	9.820	7
mr_blk13_m	400	147,505	6.317	1.000	147,505	-	6.317	
					11,914,640	**11,557,349**	**9.379**	**108**

Merensky Domain 1 - Summary of Geological Data

TABLE 8a

ELANDSFONTEIN PROJECT - Mining Blocks from Resource Model

Block	Top	Bottom	Total Resource Tons	Stoping Tonnage at CW	Mineable Tons at CW	Corrected Tons for SW	Grade	Content kg
1	-50	-150	5,646,621	154,971	-	-	-	-
2	-130	-260	1,463,903	54,815	-	-	-	-
3	-300	-440	1,513,090	637,179	637,179	760,484	7.616	5,792
4	-380	-520	1,615,399	1,130,779	1,130,779	1,335,742	8.643	11,545
5	-250	-500	1,327,478	603,359	603,359	717,426	7.182	5,153
6	-350	-500	2,501,556	1,634,276	1,634,276	1,934,861	6.098	11,799
7	-400	-550	1,869,812	1,308,868	1,308,868	1,543,097	8.075	12,460
8	-420	-560	2,088,127	1,461,689	1,461,689	1,725,199	7.820	13,491
9	-420	-560	729,839	510,887	510,887	598,729	9.340	5,592
10	-400	-510	1,357,299	950,110	950,110	1,125,779	7.999	9,005
11	-430	-530	3,655,743	2,559,020	2,559,020	3,027,490	8.615	26,082
12	-400	-520	1,122,635	761,182	761,182	902,969	8.278	7,475
13	-120	-400	4,391,898	147,505	-	-	-	-
			29,283,401	**11,914,640**	**11,557,349**	**13,671,775**	**7.928**	**108,394**

Mineable Tonnage for Elandsfontein Project - Merensky Domain 1

TABLE 8b

ELANDSFONTEIN PROJECT

"Basket Price" Calculation

		Base Price	Exchange Rate	Discount	Metal Price	Split	Head Grade	Assumed Price
		$US/oz	R/$US		R/kg	%	g/t	R/kg
Basket Meta	Pt	$871	6.55		R 183,422	61.0%	4.00	
	Pd	$184	6.55		R 38,748	30.0%	1.97	
	Rh	$1,930	6.55		R 406,434	4.0%	0.26	
	Au	$426	6.55		R 89,710	5.0%	0.33	
	3PGE's &	$686	Basket Price		R 144,509	100.0%	6.55	R 144,000
	Ir	$154	6.55		R 32,430			R 32,000
	Ru	$68	6.55		R 14,320			R 14,000
	Os	$750	6.55		R 157,941			R 150,000
		$US/ton			R/ton			
	Cu	$3,500.00	6.55	$150.00	R 21,943	Prices 01-Jul-05		R 21,000
	Ni	$14,500.00	6.55	$70.00	R 94,517	Date 01-Jul-05		R 94,000

TABLE 9

PTM - Platinum Group Metals Ltd.
ELANDSFONTEIN PROJECT - CAPTEX SCHEDULE
Scoping Study - July 2005 - Shaft System - VALUATION MODEL

Production Schedule of: ELANDSFONTEIN PROJECT
MR UNDERGROUND MINE with SHAFT

		Totals	-2 2004/05	-1 2005/06	1 2006/07	2 2007/08	3 2008/09	4 2009/10	5 2010/11	6 2011/12	7 2012/13	8 2013/14	9 2014/15	10 2015/16	11 2016/17	12 2017/18	13 2018/19
Capital Expenditure																	
PTM Costs (not included in CAPEX Schedule)																	
Sunk Costs from previous Work on Project	Rands	7,000,000		7,000,000													
Geology	Rands	8,000,000		8,000,000													
Seismic Survey	Rands	4,000,000		3,000,000	1,000,000												
Metallurgical Testwork	Rands	200,000		200,000													
Mine Planning etc. Mine24D	Rands	500,000		500,000													
Scoping & Pre-Feasibility Study	Rands	700,000		700,000													
Owners Costs	Rands	3,000,000		1,500,000	1,500,000												
Project Costs																	
Metallurgical Testwork	Rands	1,000,000			1,000,000												
Geology	Rands	11,000,000		2,000,000	7,000,000	2,000,000											
Mine Planning etc. Mine24D	Rands	1,500,000			1,500,000												
Definitive Feasibility Study	Rands	5,000,000		3,000,000	2,000,000												
Land Purchase	Rands	5,000,000		1,000,000	4,000,000												
Owners Costs	Rands	7,500,000				1,500,000	1,500,000	1,500,000	1,500,000	1,000,000	500,000						
Phase 1 Financing																	
Shaft Access with In-Circle Development																	
Main Shaft - 7.7 m final diameter																	
Shaft P&Gs @ 15%	Rands	25,641,079		0	5,250,000	12,941,400	5,199,680	2,250,000	0	0	0	0	0	0	0	0	0
Preparation to Sink	Rands	60,000,000			35,000,000	25,000,000											
Pre Sink - surface to 100m	Rands	17,969,436				17,969,436											
Sink, Line & Equip Main Shaft - 100m to 685m (Shaft Bottom)	Rands	76,771,250				32,500,000	29,271,250	15,000,000									
Cut Station - Level 130	Rands	-															
Cut Station - Level 270	Rands	-															
Cut Station - Level 300	Rands	1,348,320				1,348,320											
Cut Station - Level 330	Rands	1,348,320				1,348,320											
Cut Station - Level 360	Rands	1,348,320				1,348,320											
Cut Station - Level 390	Rands	1,348,320				1,348,320											
Cut Station - Level 420	Rands	1,348,320				1,348,320											
Cut Station - Level 450	Rands	1,348,320				1,348,320											
Cut Station - Level 480	Rands	1,348,320				1,348,320											
Cut Station - Level 510	Rands	1,348,320				1,348,320											
Cut Station - Level 540	Rands	1,348,320					1,348,320										
Cut Station - Level 570	Rands	1,348,320					1,348,320										
Cut Station - Transfer Level	Rands	1,348,320					1,348,320										
Cut Shaft Bottom	Rands	1,348,320					1,348,320										
Ventilation Shaft - 3.5 m final diameter																	
Shaft P&Gs @ 15%	Rands	12,159,281		0	3,375,000	7,897,520	886,760	0	0	0	0	0	0	0	0	0	0
Preparation to Sink	Rands	30,000,000			22,500,000	7,500,000											
Pre Sink - surface to 100m	Rands	9,716,751				9,716,751											
Sink, Line & Equip Vent Shaft - 100m to 685m (Shaft Bottom)	Rands	34,602,176				31,500,000	3,102,176										
Cut Station - Level 130	Rands	-															
Cut Station - Level 270	Rands	-															
Cut Station - Level 300	Rands	561,912				561,912											
Cut Station - Level 330	Rands	561,912				561,912											
Cut Station - Level 360	Rands	561,912				561,912											
Cut Station - Level 390	Rands	561,912				561,912											
Cut Station - Level 420	Rands	561,912				561,912											
Cut Station - Level 450	Rands	561,912				561,912											
Cut Station - Level 480	Rands	561,912				561,912											
Cut Station - Level 510	Rands	561,912					561,912										
Cut Station - Level 540	Rands	561,912					561,912										
Cut Station - Level 570	Rands	561,912					561,912										
Cut Station - Transfer Level	Rands	561,912					561,912										
Cut Shaft Bottom	Rands	561,912					561,912										
In-Circle Development																	
Development P&Gs @ 15%	Rands	11,191,420			0	0	11,191,420	0	0	0	0	0	0	0	0	0	0
Level 130	Rands	-															
Level 270	Rands	-															
Level 300	Rands	5,666,263					5,666,263										
Level 330	Rands	3,437,946					3,437,946										
Level 360	Rands	2,895,113					2,895,113										
Level 390	Rands	9,228,172					9,228,172										
Level 420	Rands	5,066,447					5,066,447										
Level 450	Rands	7,961,560					7,961,560										
Level 480	Rands	2,442,751					2,442,751										
Level 510	Rands	9,228,172					9,228,172										
Level 540	Rands	11,037,617					11,037,617										
Level 570	Rands	7,056,837					7,056,837										
Transfer Level	Rands	6,483,846					6,483,846										
Shaft Bottom	Rands	904,723					904,723										
Primary & Secondary Development																	
Development P&Gs @ 15%	Rands	34,188,693		0	0	0	6,953,573	18,005,253	5,002,846	2,500,073	1,375,073	350,036	0	0	0	0	0
Area 2 Horizontal	Rands	-															
Area 2 Decline	Rands	-															
Area 2 Other Mining	Rands	-															
Area 3 Horizontal	Rands	55,272,131					20,000,000	35,272,131									
Area 3 Decline	Rands	11,833,575					1,500,000	3,500,000	3,500,000	3,333,575							
Area 3 Other Mining	Rands	925,487						925,487									
Area 12 Horizontal	Rands	-															
Area 12 Decline	Rands	-															
Area 12 Other Mining	Rands	-															
Area 4 Horizontal	Rands	17,857,150					17,857,150										
Area 4 Decline	Rands	11,833,575					2,000,000	3,500,000	3,500,000	2,833,575							
Area 4 Other Mining	Rands	925,487						925,487									
Area 11 Horizontal	Rands	17,857,150						17,857,150									
Area 11 Decline	Rands	11,833,575						2,000,000	3,500,000	3,500,000	2,833,575						
Area 11 Other Mining	Rands	925,487							925,487								
Area 10 Horizontal	Rands	-															
Area 10 Decline	Rands	-															
Area 10 Other Mining	Rands	-															
Area 8&9 Horizontal	Rands	-															
Area 8&9 Decline	Rands	-															
Area 8&9 Other Mining	Rands	-															
Area 7 Horizontal	Rands	-															
Area 7 Decline	Rands	-															
Area 7 Other Mining	Rands	-															
Area 6 Horizontal	Rands	17,857,150						17,857,150									
Area 6 Decline	Rands	11,833,575						2,000,000	3,500,000	3,500,000	2,833,575						
Area 6 Other Mining	Rands	925,487							925,487								
Area 5 Horizontal	Rands	55,272,131					5,000,000	35,272,131	15,000,000								
Area 5 Decline	Rands	11,833,575							2,500,000	3,500,000	3,500,000	2,333,575					
Area 5 Other Mining	Rands	925,487						925,487									
Area 1 Horizontal	Rands	-															
Area 1 Decline	Rands	-															
Area 1 Other Mining	Rands	-															
Area 13 Horizontal	Rands	-															
Area 13 Decline	Rands	-															
Area 13 Other Mining	Rands	-															
Ventilation																	
Ventilation Raise North	Rands	-															
Ventilation Raise South	Rands	-															
Supply Portion																	
Supply P&Gs @ 15%	Rands	103,200,000			14,625,000	29,550,000	47,700,000	11,325,000	0	0	0	0	0	0	0	0	0
Supply Estimates - Surface Electrical Installation	Rands	35,000,000			15,000,000	15,000,000	5,000,000										
Supply Estimates - Winders	Rands	170,000,000			50,000,000	75,000,000	45,000,000										
Supply Estimates - Water Supply	Rands	7,000,000			1,500,000	2,500,000	3,000,000										
Supply Estimates - Compressed Air	Rands	60,000,000			5,000,000	30,000,000	25,000,000										
Supply Estimates - Ventilation Fans	Rands	16,000,000					16,000,000										
Supply Estimates - Offices and Workshops	Rands	5,000,000			500,000	1,500,000	2,500,000	500,000									
Supply Estimates - Underground Equipment	Rands	100,000,000			10,000,000	30,000,000	45,000,000	15,000,000									
Supply Estimates - Vehicles, etc.	Rands	5,000,000			500,000	1,000,000	3,500,000										
Other Costs	Rands	75,000,000			15,000,000	30,000,000	20,000,000	10,000,000									
Concentrator Plant	Rands	200,000,000				10,000,000	140,000,000	50,000,000									
Tailings Dam - Phase 1	Rands	15,000,000				2,000,000	13,000,000										
Phase 2 Financing																	
Mining Aspects																	
In-Circle Development																	
Development P&Gs @ 15%	Rands	3,976,256			0	0	0	0	1,677,801	1,905,622	392,834	0	0	0	0	0	0
Level 130	Rands	-															
Level 270	Rands	-															
Level 300	Rands	-															
Level 330	Rands	3,618,891							3,618,891								
Level 360	Rands	5,066,447							5,066,447								
Level 390	Rands	1,809,445							500,000	1,309,445							
[illegible]	Rands	[illegible]							[illegible]	[illegible]							

TABLE 10

Production Schedule of: ELANDSFONTEIN PROJECT
NR UNDERGROUND MINE with SHAFT

			Totals	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12	2012/13	2013/14	2014/15	2015/16	2016/17	20
1			Year Number	-2	-1	1	2	3	4	5	6	7	8	9	10	11	
2	PGM Basket Price	R/kg	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	
3	*New Project Profile*																
4	*WASTE TONS MINED*	tons	-														
5	*Topsoil*	tons	-		-	-	-	-	-	-	-	-	-	-	-	-	
6	*Oxide Reef*	tons	-		-	-	-	-	-	-	-	-	-	-	-	-	
7	*Waste*	tons	3,417,944	-	-	-	-	-	13,750	123,750	288,750	295,000	295,000	295,000	295,000	295,000	
8	*Working Cost*	tons	3,417,944	-	-	-	-	-	13,750	123,750	288,750	295,000	295,000	295,000	295,000	295,000	
9	*REEF TONS MINED*	tons	15,722,542	-	-	-	-	-	63,250	569,250	1,236,250	1,357,000	1,357,000	1,357,000	1,357,000	1,357,000	1
10	*Stope Tons*	tons	13,671,776	-	-	-	-	-	55,000	495,000	1,075,000	1,180,000	1,180,000	1,180,000	1,180,000	1,180,000	1
11	*Stope Dilution Tons*	tons	2,050,766	-	-	-	-	-	8,250	74,250	161,250	177,000	177,000	177,000	177,000	177,000	
12	*Development Reef Tons*	tons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
13	*TOTAL REEF TONS TO SURFACE*	tons	15,722,542	-	-	-	-	-	63,250	569,250	1,236,250	1,357,000	1,357,000	1,357,000	1,357,000	1,357,000	1
14	*Stockpile Tons*	tons		-	-	-	-	-	1,250	5,500	1,750	2,750	2,750	2,750	2,750	2,750	
15	*TONS MILLED*	tons	15,722,542						62,000	565,000	1,240,000	1,356,000	1,357,000	1,357,000	1,357,000	1,357,000	1
16	*In situ Grade (corrected to SW)*	3PGE's&Au (g/t)		-	-	-	-	-	6.66	7.39	7.55	7.70	7.77	7.96	8.05	7.97	
17		%Ni		0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	0.078%	
18		%Cu		0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	
19	*Geological Losses*	%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	
20	*Mine Call Factor (MCF)*	%		95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	
21	*Effective Stope Grade*	3PGE's&Au (g/t)	7.53	0.00	0.00	0.00	0.00	0.00	7.88	7.02	7.17	7.31	7.38	7.56	7.65	7.57	
22	*Stockpile Grade*	3PGE's&Au (g/t)		-	-	-	-	-	6.88	6.10	6.24	6.36	6.42	6.59	6.65	6.58	
23	*MILL HEAD GRADE*	3PGE's&Au (g/t)	6.55	0.00	0.00	0.00	0.00	0.00	6.68	6.10	6.24	6.36	6.42	6.58	6.65	6.58	
24	*PGM's to Surface*	3PGE's&Au (g/t)	102974.6	-	-	-	-	-	422.3	3,473.6	7,712.8	8,631.2	8,714.0	8,947.8	9,021.5	8,931.6	
25	*PGM's in Mill Feed*	3PGE's&Au (g/t)	102976.2	-	-	-	-	-	414.0	3,447.7	7,736.2	8,624.6	8,714.0	8,947.8	9,021.5	8,931.6	
26																	
27																	
28	Tons Milled	tons	15,722,542	-	-	-	-	-	62,000	565,000	1,240,000	1,356,000	1,357,000	1,357,000	1,357,000	1,357,000	1
29	Head Grade	3PGE's&Au (g/t)		0.00	0.00	0.00	0.00	0.00	6.68	6.10	6.24	6.36	6.42	6.58	6.65	6.58	
30		Cu (%)		0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	0.012%	
31		Ni (%)		0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	0.074%	
32	Concentrator Recovery	3PGE's&Au (g/t)		0.0%	0.0%	0.0%	0.0%	80.0%	80.0%	84.0%	86.8%	87.8%	87.8%	87.3%	87.3%	87.2%	
33		Cu (%)		0.0%	0.0%	0.0%	0.0%	50.0%	50.0%	50.0%	55.0%	60.0%	60.0%	60.0%	60.0%	60.0%	
34		Ni (%)		0.0%	0.0%	0.0%	0.0%	55.0%	55.0%	55.0%	58.0%	65.0%	65.0%	70.0%	70.0%	70.0%	
35	Concentrate Tonnage	tons	384,569		0	0	0	0	2298	16549	33486	37518	34460	31245	31503	31154	
36	Mass Pull	2.45%		0.0%	0.0%	0.0%	0.0%	0.0%	3.6%	2.9%	2.7%	2.8%	2.5%	2.3%	2.3%	2.3%	
37	Calculated Tailings Grade	3PGE's&Au (g/t)		0.00	0.00	0.00	0.00	0.00	1.38	1.01	0.88	0.85	0.88	0.88	0.88	0.88	
38		Cu (%)		0.000%	0.000%	0.000%	0.000%	0.000%	0.006%	0.006%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	
39		Ni (%)		0.000%	0.000%	0.000%	0.000%	0.000%	0.033%	0.033%	0.031%	0.028%	0.028%	0.022%	0.022%	0.022%	
37	Concentrate Grade	3PGE's&Au (g/t)	233.2		75.0	150.0	150.0	150.0	150.0	175.0	200.0	200.0	220.0	250.0	250.0	250.0	
38		Cu (%)	0.281%		0.0%	0.0%	0.0%	0.0%	0.2%	0.2%	0.2%	0.3%	0.3%	0.3%	0.3%	0.3%	
39		Ni (%)	2.037%		0.0%	0.0%	0.0%	0.0%	1.1%	1.4%	1.6%	1.7%	1.9%	2.2%	2.2%	2.2%	
40		Ru (g/t)	1.0		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
41		Ir (g/t)	0.0		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
42		Os (g/t)	0.0		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
43		Cr2O3 (%)							4.0%	2.5%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	
44	Concentrate Contents	3PGE's&Au (kg)	89671		0.0	0.0	0.0	0.0	331.2	3886.1	6699.6	7503.6	7581.2	7811.2	7875.6	7788.4	
45		Cu (tons)	1079		0.0	0.0	0.0	0.0	3.6	32.7	79.0	94.3	94.4	94.4	94.4	94.4	
46		Ni (tons)	7834		0.0	0.0	0.0	0.0	25.1	229.1	530.2	649.8	650.2	700.3	700.3	700.3	
47		Ru (kg)	385		0.0	0.0	0.0	0.0	2.2	16.5	33.5	37.5	34.5	31.2	31.5	31.2	
48		Ir (kg)	0		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
49		Os (kg)	0		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
50	Smelter Recovery	3PGE's&Au (%)			88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	
51		Cu (%)			67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	67.5%	
52		Ni (%)			72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	72.5%	
53		Ru (%)			45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	
54		Ir (%)			55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	
55		Os (%)			0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	
56	Metal Production	3PGE's&Au (kg)	77117		0	0	0	0	285	2491	5762	6463	6520	6718	6773	6699	
57		Cu (tons)	728		0	0	0	0	2	22	53	64	64	64	64	64	
58		Ni (tons)	5660		0	0	0	0	18	166	384	471	471	508	508	508	
59		Ru (kg)	173		0	0	0	0	1	7	15	17	15	14	14	14	
60		Ir (kg)	0		0	0	0	0	0	0	0	0	0	0	0	0	
61		Os (kg)	0		0	0	0	0	0	0	0	0	0	0	0	0	
62	Metal Prices	3PGE'S&Au (R/kg)		R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	R 144,000	
63		Cu (R/tons)		R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	R 21,000	
64		Ni (R/tons)		R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	R 94,000	
65		Ru (R/kg)		R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	R 14,000	
66		Ir (R/kg)		R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	R 32,000	
67		Os (R/kg)		R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	R 150,000	
68	Metal Revenue	3PGE's&Au (R)		R 0	R 0	R 0	R 0	R 0	R 41,013,912	R 358,648,859	R 829,672,561	R 929,247,862	R 938,859,057	R 967,344,352	R 975,340,485	R 994,515,224	R 98
69		Cu (R)		R 0	R 0	R 0	R 0	R 0	R 50,929	R 464,114	R 1,120,446	R 1,336,849	R 1,337,635	R 1,337,635	R 1,337,635	R 1,337,635	R
70		Ni (R)		R 0	R 0	R 0	R 0	R 0	R 1,713,190	R 15,812,136	R 36,132,737	R 44,261,895	R 44,314,351	R 47,723,147	R 47,723,147	R 47,723,147	R 4
71		Ru (R)		R 0	R 0	R 0	R 0	R 0	R 13,910	R 164,259	R 211,036	R 236,394	R 217,069	R 198,843	R 198,470	R 198,298	R
72		Ir (R)		R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	
73		Os (R)		R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	
74		TOTAL		R 0	R 0	R 0	R 0	R 0	R 42,791,941	R 374,830,367	R 867,136,779	R 975,101,799	R 984,727,141	R 1,016,601,977	R 1,024,599,736	R 1,013,772,273	R 1,03
76	Tons Mined (Total)	tons		-	-	-	-	-	77,000	693,000	1,505,000	1,652,000	1,652,000	1,652,000	1,652,000	1,652,000	1
77	Working Cost Waste (including Dev. Reef)	tons		-	-	-	-	-	13,750	123,750	288,750	295,000	295,000	295,000	295,000	295,000	
78	Stope Tons Mined	tons		-	-	-	-	-	63,250	569,250	1,236,250	1,357,000	1,357,000	1,357,000	1,357,000	1,357,000	1
79	Mining Cost																
80	Waste Tons Mined	R/ton Waste Mined		R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	
81	Reef Tons Mined	R/ton Reefing		R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	
82	Working Cost Waste (including Dev. Reef)	Rands		R 0	R 0	R 0	R 0	R 0	R 2,337,500	R 21,037,500	R 49,687,500	R 50,150,000	R 50,150,000	R 50,150,000	R 50,150,000	R 50,150,000	R 5
83	Stope Tons Mined	Rands		R 0	R 0	R 0	R 0	R 0	R 10,762,500	R 96,772,500	R 210,162,500	R 230,690,000	R 230,690,000	R 230,690,000	R 230,690,000	R 230,690,000	R 23
84	**TOTAL MINING COSTS**	**Rands**		**R 0**	**R 0**	**R 0**	**R 0**	**R 0**	**R 13,000,000**	**R 117,810,000**	**R 255,850,000**	**R 280,840,000**	**R 280,840,000**	**R 280,840,000**	**R 280,840,000**	**R 280,840,000**	**R 29**
85		R/ton Mined		R 0.00	R 0.00	R 0.00	R 0.00	R 0.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	R 170.00	
86	Processing Cost (excluding services)	R/ton milled		R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	R 40.00	
87		**Rands**		**R 0**	**R 0**	**R 0**	**R 0**	**R 0**	**R 2,480,000**	**R 22,600,000**	**R 49,600,000**	**R 54,240,000**	**R 54,280,000**	**R 54,280,000**	**R 54,280,000**	**R 54,280,000**	**R 5**
88	Concentrate Transport Costs	Tons Produced		0	0	0	0	0	2298	16549	33486	37518	34460	31245	31503	31154	
89		Moisture (%)		17%	17%	17%	17%	17%	17%	17%	17%	17%	17%	17%	17%	17%	
90		Tons Shipped		0	0	0	0	0	2680	19669	40368	45202	41518	37645	37966	37534	
91		Cost (R/ton km)		R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	R 0.55	
92		Distance (km)		75	75	75	75	75	75	75	75	75	75	75	75	75	
93		Cost (R/ton shipped)		R 41	R 41	R 41	R 41	R 41	R 41	R 41	R 41	R 41	R 41	R 41	R 41	R 41	
94		**Cost (Rand)**		**R 0**	**R 0**	**R 0**	**R 0**	**R 0**	**R 109,730**	**R 822,465**	**R 1,664,802**	**R 1,864,802**	**R 1,712,825**	**R 1,552,837**	**R 1,565,673**	**R 1,548,298**	**R**
95		Cost (R/ton milled)		R 0.00	R 0.00	R 0.00	R 0.00	R 0.00	R 1.77	R 1.46	R 1.34	R 1.38	R 1.26	R 1.14	R 1.15	R 1.14	
96	Smelter & Refinery Charges	Charges (R/ton dry conc.)		R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	R 500.00	
97		Charges (R/kg 4PE's)		R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R 2,500.00	R
98		**Total Charges (Rand)**		**R 0**	**R 0**	**R 0**	**R 0**	**R 0**	**R 1,015,986**	**R 14,901,058**	**R 31,152,918**	**R 34,891,784**	**R 33,529,860**	**R 32,416,659**	**R 32,684,617**	**R 32,321,852**	**R 3**
99		Cost (R/ton milled)		R 0.00	R 0.00	R 0.00	R 0.00	R 0.00	R 29.29	R 25.67	R 25.12	R 25.73	R 24.71	R 23.89	R 24.09	R 23.82	
100	Services	Water (R/m3)		R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	R 4.50	
101		m3/ton mined		0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	
102		m3/ton milled		0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	
103		Water (m3)		0	0	0	0	0	61,800	559,700	1,222,000	1,336,800	1,339,300	1,339,300	1,339,300	1,339,300	
104		Power (R/kWhr)		R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	R 0.15	
105		kWh/ton mined		65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	
106		kWh/ton milled		55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	
107		Power (kWhr)		0	0	0	0	0	8,415,000	76,120,000	166,025,000	181,960,000	182,015,000	182,015,000	182,015,000	182,015,000	18
108		Cost (Rand)		R 0	R 0	R 0	R 0	R 0	R 1,540,350	R 13,938,650	R 30,402,750	R 33,316,800	R 33,329,100	R 33,329,100	R 33,329,100	R 33,329,100	R 3
109		Cost (R/ton milled)		R 0.00	R 0.00	R 0.00	R 0.00	R 0.00	R 24.84	R 24.67	R 24.52	R 24.57	R 24.56	R 24.56	R 24.56	R 24.56	
110	Administration Costs	Cost (Rand)		R 0	R 0	R 0	R 0	R 0	R 1,000,000	R 8,000,000	R 8,000,000	R 8,000,000	R 8,000,000	R 8,000,000	R 8,000,000	R 8,000,000	R
111		Cost (R/ton milled)		R 0.00	R 0.00	R 0.00	R 0.00	R 0.00	R 16.13	R 14.16	R 6.45	R 5.90	R 5.90	R 5.90	R 5.90	R 5.90	
112	Overheads	Cost (Rand)		R 0.00	R 0	R 0	R 0	R 0	R 1,240,000	R 11,300,000	R 24,800,000	R 27,120,000	R 27,140,000	R 27,140,000	R 27,140,000	R 27,140,000	R 2
113		Cost (R/ton milled)		R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	R 20.00	
114	TOTAL Production Costs	Cost (Rand)		R 0	R 0	R 0	R 0	R 0	R 21,276,076	R 188,970,174	R 401,470,466	R 440,274,986	R 438,831,385	R 437,558,597	R 437,839,391	R 437,459,248	R 43
115		Cost (R/ton milled)		R 0.00	R 0.00	R 0.00	R 0.00	R 0.00	R 343.16	R 334.46	R 323.77	R 324.69	R 323.38	R 322.45	R 322.65	R 322.37	
117	PENALTIES			R 0	R 0	R 0	R 0	R 0	R 4,205,594	R 12,755,967	R 1,548,887	R 1,725,838	R 1,585,164	R 1,437,269	R 1,449,149	R 1,433,065	R
118		Moisture (less than 15%)		R 0	R 0	R 0	R 0	R 0	R 22,079	R 165,490	R 334,576	R 375,180	R 344,601	R 312,450	R 315,032	R 311,536	
119		Chromite		R 0	R 0	R 0	R 0	R 0	R 2,132,829	R 3,924,230	R 1,205,919	R 1,350,659	R 1,240,563	R 1,124,819	R 1,134,117	R 1,121,528	R
120		Conc Grade (better than 200 g/t)		R 0	R 0	R 0	R 0	R 0	R 2,050,686	R 8,966,246	R 0	R 0	R 0	R 0	R 0	R 0	
121		Tonnage Supply		R 0	0	0	0	0	0	0	0	0	0	0	0	0	
122	ROYALTIES			R 0	R 0	R 0	R 0	R 0	R 1,431,386	R 12,516,890	R 28,955,572	R 32,436,739	R 32,796,140	R 33,769,318	R 34,039,383	R 33,661,581	R 3
123		Pt Royalty (%)						4.00%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%	4.00%	
124		Pd Royalty (%)						3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	
125		Rh Royalty (%)						0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	
126		Au Royalty (%)						3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	3.00%	
127		Base Metal Royalty (%)						0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	
128	WORKING COST SUMMARY																
129	Mining Cost	Cost (Rand)	R 3,253,582,666	R 0	R 0	R 0	R 0	R 0	R 13,000,000	R 117,810,000	R 255,850,000	R 280,840,000	R 280,840,000	R 280,840,000	R 280,840,000	R 280,840,000	R 29
130	Processing Cost (excluding services)	Cost (Rand)	R 648,013,314	R 0	R 0	R 0	R 0	R 0	R 2,589,730	R 23,422,465	R 51,264,798	R 56,104,602	R 55,922,625	R 55,832,837	R 55,845,673	R 55,828,298	R 5
131	Smelting & Treatment Charges	Cost (Rand)	R 385,086,567	R 0	R 0	R 0	R 0	R 0	R 1,015,986	R 14,901,058	R 31,152,918	R 34,891,784	R 33,529,860	R 32,416,659	R 32,684,617	R 32,321,852	R 3
132	Services	Cost (Rand)	R 386,159,309	R 0	R 0	R 0	R 0	R 0	R 1,540,350	R 13,938,650	R 30,402,750	R 33,316,800	R 33,329,100	R 33,329,100	R 33,329,100	R 33,329,100	R 3
133	Administration & Overhead Costs	Cost (Rand)	R 414,450,640	R 0	R 0	R 0	R 0	R 0	R 2,240,000	R 19,300,000	R 32,800,000	R 35,120,000	R 35,140,000	R 35,140,000	R 35,140,000	R 35,140,000	R 3
134	TOTAL	Cost (Rand)	R 5,087,394,736	R 0	R 0	R 0	R 0	R 0	R 21,276,076	R 188,970,174	R 401,470,466	R 440,274,986	R 438,831,385	R 437,558,597	R 437,839,391	R 437,459,248	R 43
135																	

Production Schedule of: ELANDSFONTEIN PROJECT
MR UNDERGROUND MINE with SHAFT

		Totals	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12	20012/13	20013/14	2014/15
		Year Number	-2	-1	1	2	3	4	5	6	7	8	9
PGM Basket Price	R/kg	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,00
	Haulage Height (m)	3.40											
	HaulageWidth (m)	3.20											
	RAW Height (m)	3.00											
	RAW Width (m)	3.00											
	Decline Height (m)	4.20											
	Decline Width (m)	4.50											
	Overbreak	10%											
	Resource Width	1.19											
	Stope Width (m)	1.19											
	Waste SG	3.00											
	Ore SG	3.20											
	Waste % of Stope Tons	25%											
WASTE TONS MINED - Working Cost													
Zone 2													
Production	tons	-	-	-	-	-	-	-	-	-	-	-	-
Zone 3													
Production	tons	190,121	-	-	-	-	-	7,500	31,250	67,500	62,500	21,371	-
Zone 12													
Production	tons	225,742	-	-	-	-	-	-	-	-	-	5,000	31,25
Zone 4													
Production	tons	333,936	-	-	-	-	-	6,250	31,250	67,500	67,500	67,500	56,19
Zone 11													
Production	tons	756,873	-	-	-	-	-	-	-	31,250	63,956	67,500	67,50
Zone 10													
Production	tons	281,445	-	-	-	-	-	-	-	-	-	-	-
Zone 8 & 9													
Production	tons	580,982	-	-	-	-	-	-	-	-	8,211	36,456	67,50
Zone 7													
Production	tons	385,774	-	-	-	-	-	-	-	-	-	-	13,40
Zone 6													
Production	tons	483,715	-	-	-	-	-	-	31,250	67,500	67,500	67,500	54,09
Zone 5													
Production	tons	179,357	-	-	-	-	-	-	30,000	35,000	25,334	29,673	5,05
Zone 1													
Production	tons	-	-	-	-	-	-	-	-	-	-	-	-
Zone 13													
Production	tons	-	-	-	-	-	-	-	-	-	-	-	-
Waste Tons Mined - Working Costs													
Production	tons	3,417,944	-	-	-	-	-	13,750	123,750	268,750	295,000	295,000	295,00
REEF TONS MINED													
Zone 2													
Production	tons	-											
Grade	g/t (3PGE's & Au)	-		-	-	-	-	-	-	-	-	-	-
Zone 3													
Production	tons	760,484						30,000	125,000	270,000	250,000	85,484	
Grade	g/t (3PGE's & Au)	760,484	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.62	7.6
Zone 12													
Production	tons	902,969							-			20,000	125,00
Grade	g/t (3PGE's & Au)	902,969	8.28	8.28	8.28	8.28	8.28	8.28	8.28	8.28	8.28	8.28	8.2
Zone 4													
Production	tons	1,335,742						25,000	125,000	270,000	270,000	270,000	224,77
Grade	g/t (3PGE's & Au)	1,335,742	8.64	8.64	8.64	8.64	8.64	8.64	8.64	8.64	8.64	8.64	8.6
Zone 11													
Production	tons	3,027,490								125,000	255,823	270,000	270,00
Grade	g/t (3PGE's & Au)	3,027,490	8.62	8.62	8.62	8.62	8.62	8.62	8.62	8.62	8.62	8.62	8.6
Zone 10													
Production	tons	1,125,779											
Grade	g/t (3PGE's & Au)	1,125,779	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.0
Zone 8 & 9													
Production	tons	2,323,928									32,843	145,623	270,00
Grade	g/t (3PGE's & Au)	2,323,928	8.21	8.21	8.21	8.21	8.21	8.21	8.21	8.21	8.21	8.21	8.2
Zone 7													
Production	tons	1,543,097											53,62
Grade	g/t (3PGE's & Au)	1,543,097	8.08	8.08	8.08	8.08	8.08	8.08	8.08	8.08	8.08	8.08	8.0
Zone 6													
Production	tons	1,934,861							125,000	270,000	270,000	270,000	216,37
Grade	g/t (3PGE's & Au)	1,934,861	6.10	6.10	6.10	6.10	6.10	6.10	6.10	6.10	6.10	6.10	6.1
Zone 5													
Production	tons	717,426							120,000	140,000	101,334	118,693	20,22
Grade	g/t (3PGE's & Au)	717,426	7.18	7.18	7.18	7.18	7.18	7.18	7.18	7.18	7.18	7.18	7.1
Zone 1													
Production	tons	-											
Grade	g/t (3PGE's & Au)	-		-	-	-	-	-	-	-	-	-	-
Zone 13													
Production	tons	-											
Grade	g/t (3PGE's & Au)	-		-	-	-	-	-	-	-	-	-	-
Total Stoped		13,671,776											
Production	tons	13,671,776	-	-	-	-	-	55,000	496,000	1,075,000	1,180,000	1,180,000	1,180,00
Grade	g/t (3PGE's & Au)		0.00	0.00	0.00	0.00	0.00	8.08	7.39	7.55	7.70	7.77	7.

TABLE 12

ELANDSFONTEIN PROJECT

SENSITIVITY ANALYSIS

Parameter	Change in Parameter	Change in Parameter	Change in Parameter
PGM Basket Price	-20%	0%	20%
NPV @ 5% (before Tax)	R 696,621	R 1,908,960	R 3,121,299
NPV @ 5% (after Tax)	R 409,626	R 1,288,285	R 2,157,648
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%
Opex	-20%	0%	20%
NPV @ 5% (before Tax)	R 2,490,825	R 1,908,960	R 1,327,095
NPV @ 5% (after Tax)	R 1,706,661	R 1,288,285	R 867,937
IRR (before Tax)	22.1%	18.9%	15.3%
IRR (after tax)	18.9%	16.0%	12.8%
Capex	-20%	0%	20%
NPV @ 5% (before Tax)	R 2,217,628	R 1,908,960	R 1,600,291
NPV @ 5% (after Tax)	R 1,525,722	R 1,288,285	R 1,048,523
IRR (before Tax)	23.4%	18.9%	15.5%
IRR (after tax)	20.0%	16.0%	12.9%
Grade	-20%	0%	20%
NPV @ 5% (before Tax)	R 696,621	R 1,908,960	R 3,121,299
NPV @ 5% (after Tax)	R 409,626	R 1,288,285	R 2,157,648
IRR (before Tax)	11.0%	18.9%	25.2%
IRR (after tax)	9.0%	16.0%	21.6%

MCF Change	-3%	95%	3%
NPV @ 5% (before Tax)	R 1,717,538	R 1,908,960	R 2,100,382
NPV @ 5% (after Tax)	R 1,150,410	R 1,288,285	R 1,426,160
IRR (before Tax)	17.8%	18.9%	20.0%
IRR (after tax)	15.0%	16.0%	17.0%

TABLE 13

VALUATION MODEL PROJECT

PTM - Platinum Group Metals Ltd. ELANDSFONTEIN Scoping Study - July 2005 - Shaft System -

Production Schedule of:

ELANDSFONTEIN PROJECT MR UNDERGROUND MINE with SHAFT

Calculated Mining Royalty	3.34%
Cost Inflation	0.00%
Metal Price Escalation	0.00%
Exchange Rate Escalation	0.00%

		Totals	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12	20012
			-2	-1	1	2	3	4	5	6	7
Cost Inflation Factor	%			1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.00
Metal Price Factor	%			1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.00
Exchange Rate factor	%			1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.00
PGM Basket Factor	%			1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.00
PGM Basket Price	R/kg	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,000	144,
New Project Profile											
Tons Milled	tons	15,722,542	-	-	-	-	-	62,000	565,000	1,240,000	1,356,
Concentrator Recovered PGM's	kg	89,670.6	-	-	-	-	-	331	2,896	6,700	7,
Smelter Recovered PGM's	kg	77,116.7	-	-	-	-	-	285	2,491	5,762	6,
Overall Recovery Grade	g/t	4.90	-	-	-	-	-	4.59	4.41	4.65	
Costs per Ton	R	324	-	-	-	-	-	343.16	334.46	323.77	32
Project Capital Expenditure	R' 000	1,961,279	-	6,000	193,750	389,815	552,974	243,615	75,717	117,741	112,
Financial Evaluation (Stand-Alone Basis)											
Revenue	R' 000	11,656,443	-	-	-	-	-	42,792	374,830	867,137	975,
Mining Royalty	R' 000	387,558	-	-	-	-	-	1,431	12,517	28,956	32,
Working Costs	R' 000	5,087,595	-	-	-	-	-	21,276	188,970	401,470	440,
Penalties	R' 000	33,790	-	-	-	-	-	4,206	12,756	1,541	1,
Working Profit	R' 000	6,147,501	-	-	-	-	-	15,879	160,587	435,170	500,
Total Capital Expenditure	R' 000	1,961,279	-	6,000	193,750	389,815	552,974	243,615	75,717	117,741	112,
Pre-Tax Net Cash Flow	R' 000	4,186,221	-	(6,000)	(193,750)	(389,815)	(552,974)	(227,736)	84,871	317,429	387,
Pre-Tax Cumulative Cash Flow	R' 000		-	*(6,000)*	*(199,750)*	*(589,565)*	*(1,142,539)*	*(1,370,276)*	*(1,285,405)*	*(967,977)*	*(580,*
After Tax Net Cash Flow	R' 000	2,972,217	-	(6,000)	(193,750)	(389,815)	(552,974)	(227,736)	84,871	317,429	387,
After Tax Cumulative Cash Flow	R' 000	*R 0*	-	*(6,000)*	*(199,750)*	*(589,565)*	*(1,142,539)*	*(1,370,276)*	*(1,285,405)*	*(967,977)*	*(580,*
Tax Calculation:											
Unred.Cap.Bal b/f	R' 000		-	-	*6,000*	*199,750*	*589,565*	*1,142,539*	*1,370,276*	*1,285,405*	*967,*
Total Capital for Tax	R' 000	*8,269,992*	-	*6,000*	*199,750*	*589,565*	*1,142,539*	*1,386,155*	*1,445,993*	*1,403,146*	*1,080,*
Redemption	R' 000	*1,961,279*	-	-	-	-	-	*15,879*	*160,587*	*435,170*	*500,*
Unred.Cap.Bal c/f	R' 000		-	*6,000*	*199,750*	*589,565*	*1,142,539*	*1,370,276*	*1,285,405*	*967,977*	*580,*
Taxable Income	R' 000	*4,186,221*	-	-	-	-	-	-	-	-	
Y Tax Rate	%		-	-	-	-	-	-	-	-	
Tax Payable	R' 000	*1,214,004*	-	-	-	-	-	-	-	-	

Financial analysis (as at 01 July 2005):

Pre-Tax Net Present Value @	5.0%	10.0%	15.0%
R' 000	1,908,960	**795,618**	236,984
Pre-Tax IRR	**18.9%**		
Payback			

After Tax Net Present Value @	5.0%	10.0%	1
R' 000	1,288,285	**464,483**	53,
After-Tax IRR	**16.0%**		
Payback			

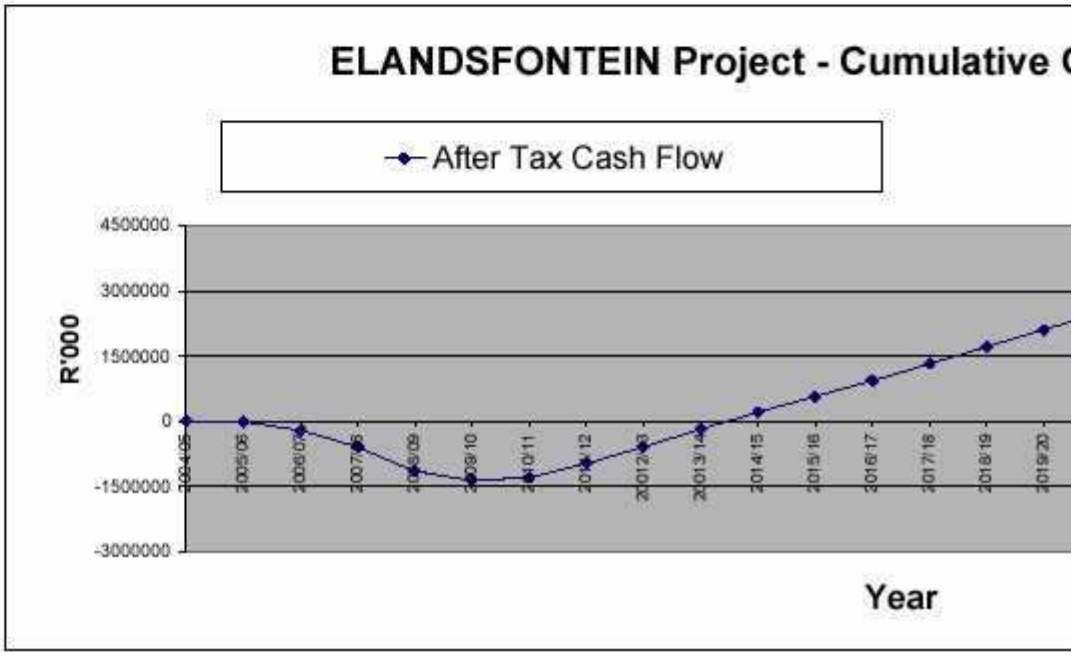
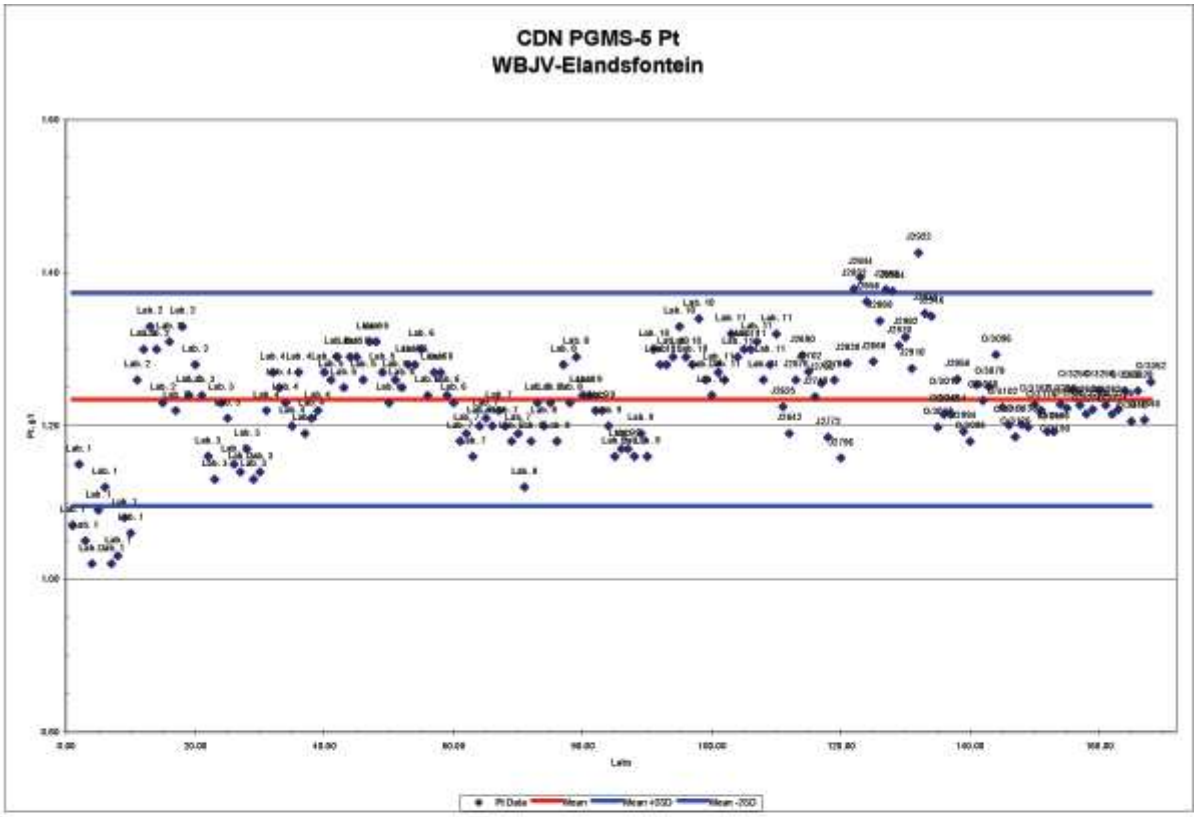
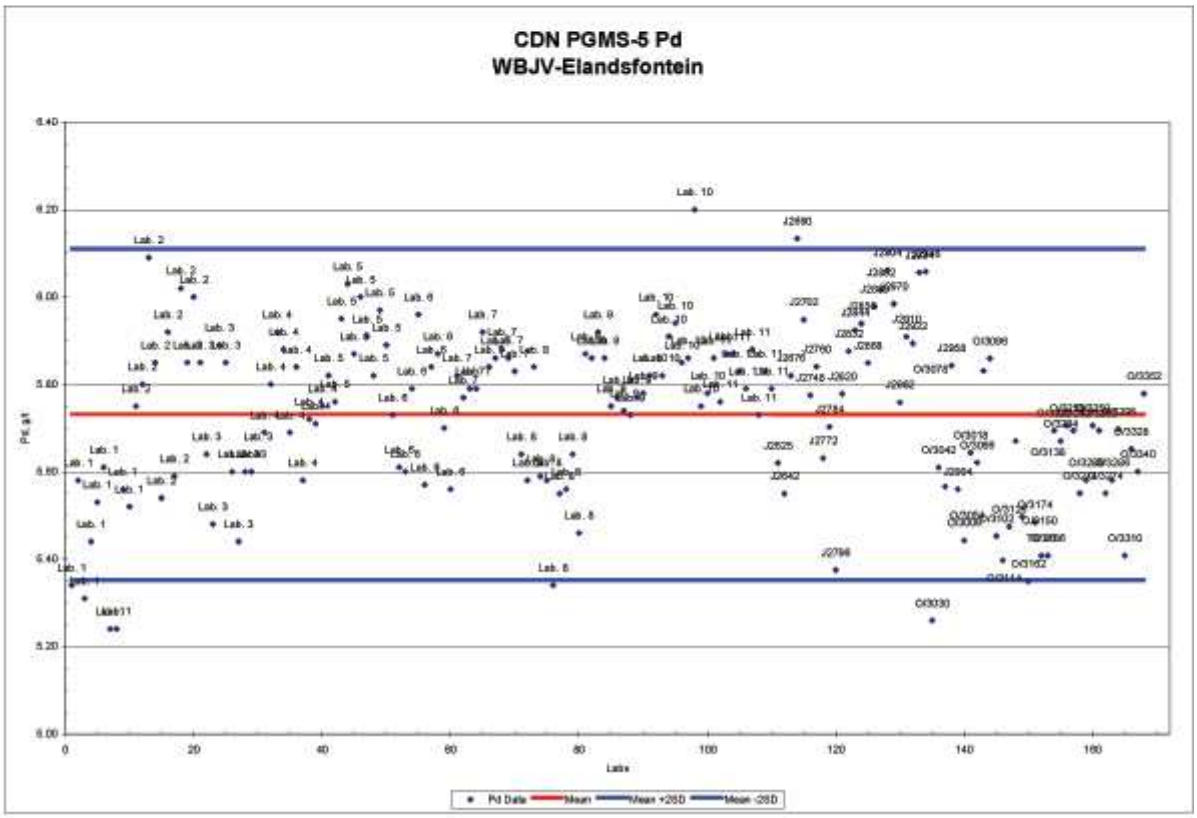
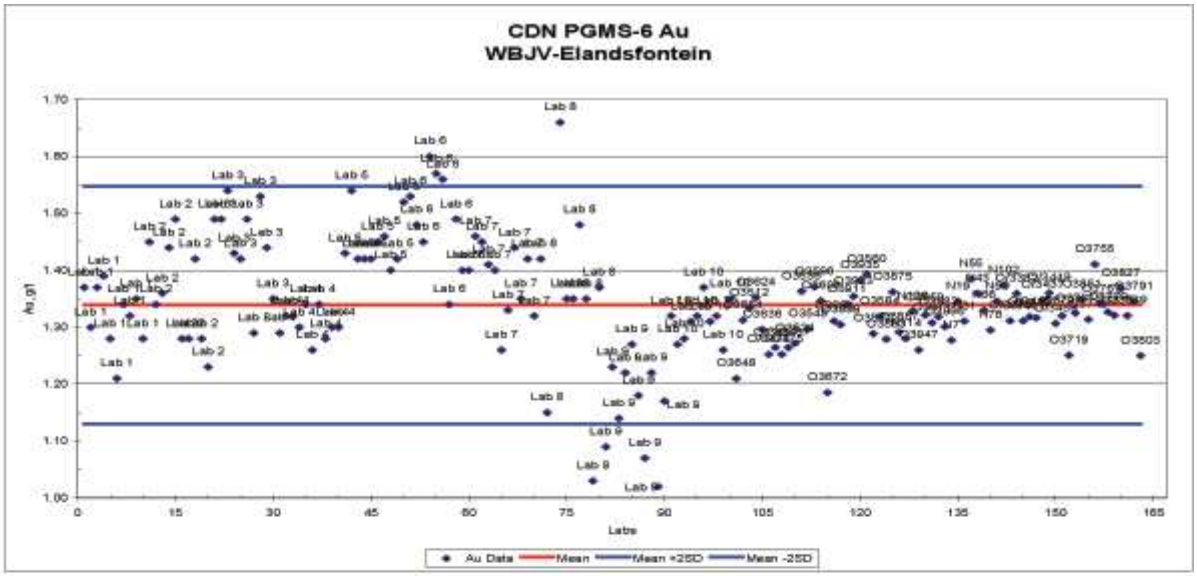


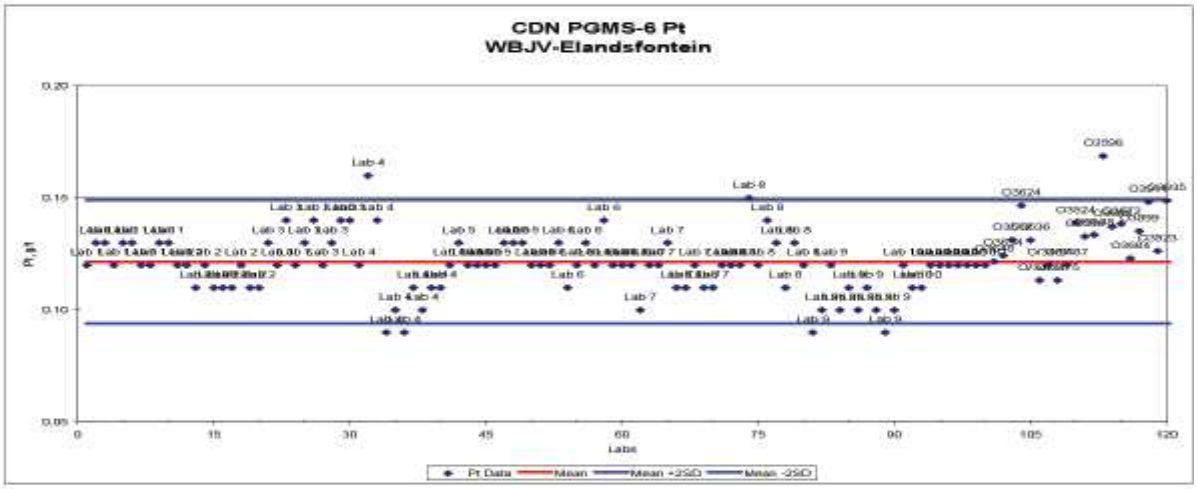
TABLE 14

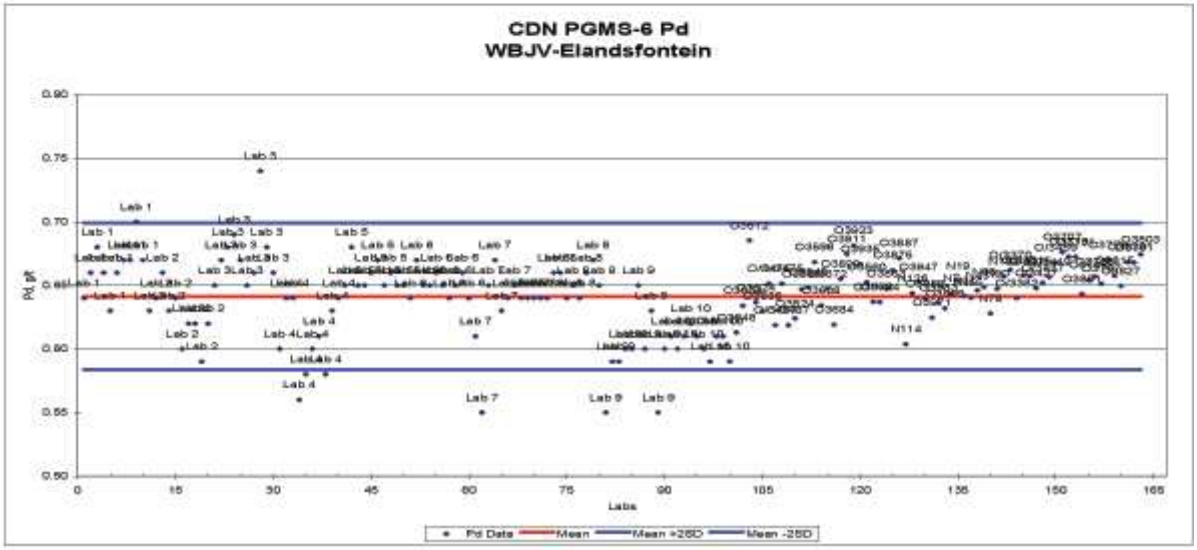




Graph 1 : QAQC WBJV Elandsfontein (CDN PGMS-5)



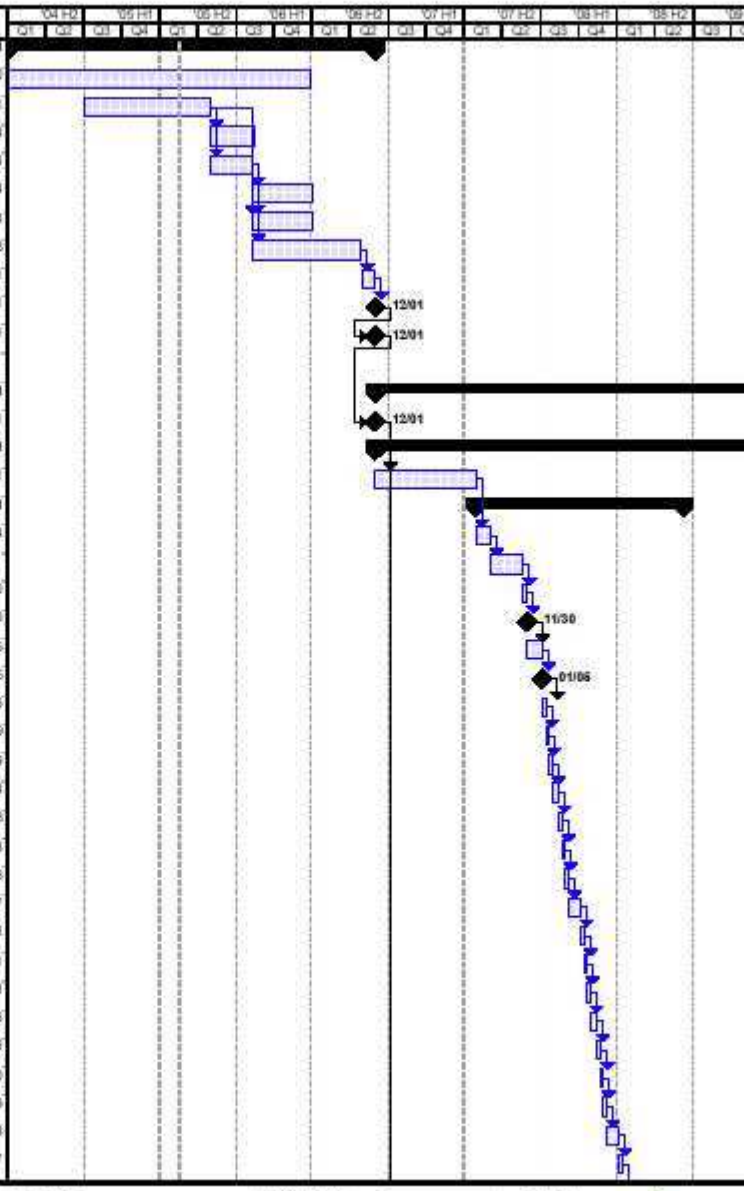




Graph 2 : QAQC WBJV Elandsfontein (CDN PGMS-6)

ELANDSFONTEIN
PROJECT - SHAFT OPTION

ID	Task Name	Duration	Start	Finish
1	**OVERALL ELANDSFONTEIN PROJECT**	**768 days**	**Thu 04/07/01**	**Fri 06/12/01**
2	Drilling & Seismics	628 days	Thu 04/07/01	Fri 06/06/30
3	Scoping studies	260 days	Mon 05/01/03	Tue 05/11/01
4	Pre-Feasibility Study	90 days	Wed 05/11/02	Tue 06/02/14
5	GeoStats & Modelling	85 days	Wed 05/11/02	Wed 06/02/08
6	Mine Planning etc. Mine24D	125 days	Thu 06/02/09	Tue 06/07/04
7	Metallurgical Testwork	125 days	Thu 06/02/09	Tue 06/07/04
8	Definitive Feasibility Study	225 days	Thu 06/02/09	Sat 06/10/28
9	Project Approval	29 days	Mon 06/10/30	Fri 06/12/01
10	Project Financing	0 days	Fri 06/12/01	Fri 06/12/01
11	Project Go-Ahead	0 days	Fri 06/12/01	Fri 06/12/01
12				
13	**Elandsfontein Project - Main and Vent Shafts**	**966 days**	**Fri 06/12/01**	**Mon 09/12/21**
14	Project Start Date	0 days	Fri 06/12/01	Fri 06/12/01
15	**Main Shaft**	**966 days**	**Sat 06/12/02**	**Mon 09/12/21**
16	Prepare to Sink - Main Shaft	207 days	Sat 06/12/02	Tue 07/07/31
17	**Sink and Line Main Shaft**	**428 days**	**Wed 07/08/01**	**Thu 08/12/11**
18	Mobilise Shaft Sinkers	30 days	Wed 07/08/01	Tue 07/09/04
19	Presink to 100m below collar (bc) (@1.5m/day)	67 days	Wed 07/09/05	Wed 07/11/21
20	Sink & line to 130 Level (30 m @ 4m/day)	8 days	Thu 07/11/22	Fri 07/11/30
21	Develop station on 130 Level (20m @ 6m/day)	0 days	Fri 07/11/30	Fri 07/11/30
22	Sink & line to 270 Level (140 m @ 4.5m/day)	31 days	Sat 07/12/01	Sat 08/01/05
23	Develop station on 270 Level (80m @ 6m/day)	0 days	Sat 08/01/05	Sat 08/01/05
24	Sink & line to 300 Level (30 m @ 4m/day)	8 days	Mon 08/01/07	Tue 08/01/15
25	Develop station on 300 Level (20m @ 6m/day)	4 days	Wed 08/01/16	Sat 08/01/19
26	Sink & line to 330 Level (30 m @ 4m/day)	8 days	Mon 08/01/21	Tue 08/01/29
27	Develop station on 330 Level (80m @ 6m/day)	14 days	Wed 08/01/30	Thu 08/02/14
28	Sink & line to 360 Level (30 m @ 4m/day)	8 days	Fri 08/02/15	Sat 08/02/23
29	Develop station on 360 Level (20m @ 6m/day)	4 days	Mon 08/02/25	Thu 08/02/28
30	Sink & line to 390 Level (30 m @ 4m/day)	8 days	Fri 08/02/29	Sat 08/03/08
31	Develop station on 390 Level (150m @ 6m/day)	25 days	Mon 08/03/10	Mon 08/04/07
32	Sink & line to 420 Level (30 m @ 4m/day)	8 days	Tue 08/04/08	Wed 08/04/16
33	Develop station on 420 Level (20m @ 6m/day)	4 days	Thu 08/04/17	Mon 08/04/21
34	Sink & line to 450 Level (30 m @ 4m/day)	8 days	Tue 08/04/22	Wed 08/04/30
35	Develop station on 450 Level (80m @ 6m/day)	14 days	Thu 08/05/01	Fri 08/05/16
36	Sink & line to 480 Level (30 m @ 4m/day)	8 days	Sat 08/05/17	Mon 08/05/26
37	Develop station on 480 Level (20m @ 6m/day)	4 days	Tue 08/05/27	Fri 08/05/30
38	Sink & line to 510 Level (30 m @ 4m/day)	8 days	Sat 08/05/31	Mon 08/06/09
39	Develop station on 510 Level (150m @ 6m/day)	25 days	Tue 08/06/10	Tue 08/07/08
40	Sink & line to 540 Level (30 m @ 4m/day)	8 days	Wed 08/07/09	Thu 08/07/17



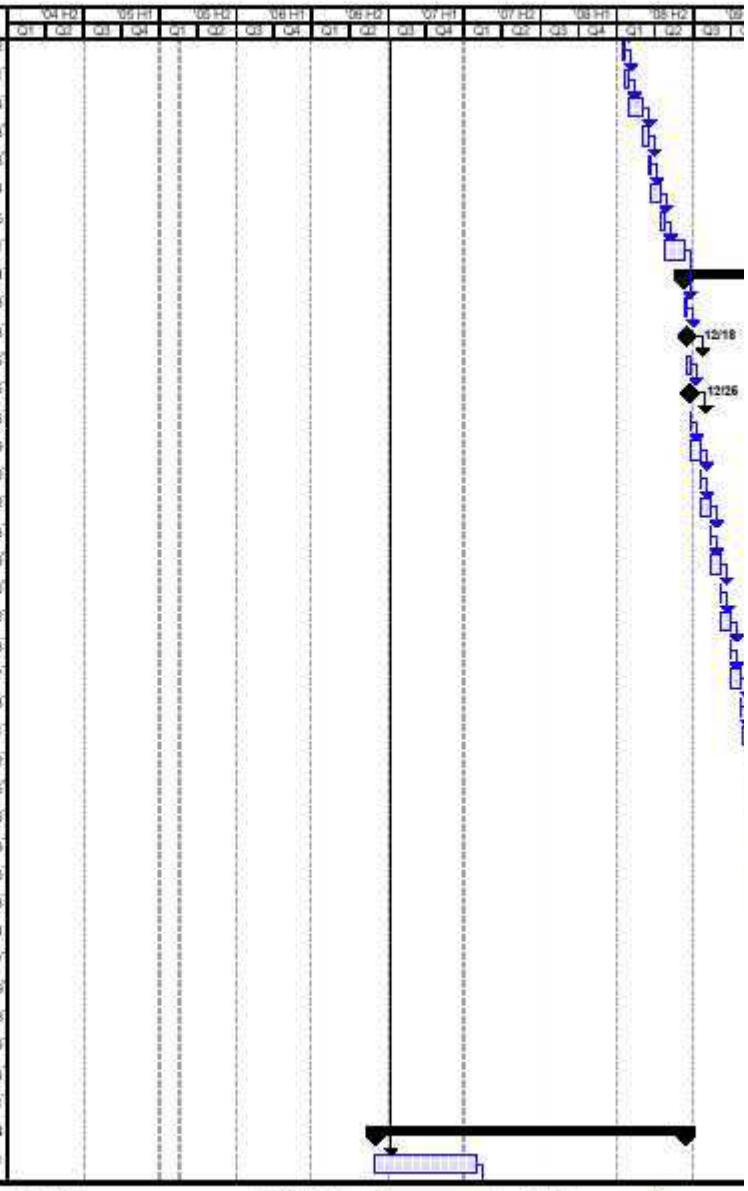
Project: PTM - Elandsfontein Project - Date: Tue 05/06/18

Task | Split | Progress | Milestone | Summary | Rolled Up Task | Rolled Up Split | Rolled Up Milestone | Rolled Up Progress | External Tasks | Project Summary | External Milestone | Deadline

GRAPH 3 - Page 1

ELANDSFONTEIN
PROJECT - SHAFT OPTION

ID	Task Name	Duration	Start	Finish
41	Develop station on 540 Level (20m @ 6m/day)	4 days	Fri 08/07/18	Tue 08/07/22
42	Sink & line to 570 Level (30 m @ 4m/day)	8 days	Wed 08/07/23	Thu 08/07/31
43	Develop station on 570 Level (180m @ 6m/day)	30 days	Fri 08/08/01	Thu 08/09/04
44	Sink & line to 620 Level (50 m @ 4m/day)	12 days	Fri 08/09/05	Thu 08/09/18
45	Develop station on 620 Level (20m @ 6m/day)	4 days	Fri 08/09/19	Tue 08/09/23
46	Sink & line to 665 Level (45 m @ 2.5m/day)	18 days	Wed 08/09/24	Tue 08/10/14
47	Develop station on 665 Level (60m @ 6m/day)	10 days	Wed 08/10/15	Sat 08/10/25
48	Changeover to equipping	40 days	Mon 08/10/27	Thu 08/12/11
49	**Equipping Main Shaft**	**321 days**	**Fri 08/12/12**	**Mon 09/12/21**
50	Equip from collar to 130 Level (130m @ 25m/day)	6 days	Fri 08/12/12	Thu 08/12/18
51	Equip station on 130 Level	0 days	Thu 08/12/18	Thu 08/12/18
52	Equip to 270 Level (140m @ 25m/day)	6 days	Fri 08/12/19	Thu 08/12/25
53	Equip station on 270 Level	0 days	Thu 08/12/25	Thu 08/12/25
54	Equip to 300 Level (30m @ 25m/day)	1 day	Fri 08/12/26	Fri 08/12/26
55	Equip station on 300 Level	20 days	Sat 08/12/27	Mon 09/01/19
56	Equip to 330 Level (30m @ 25m/day)	1 day	Tue 09/01/20	Tue 09/01/20
57	Equip station on 330 Level	20 days	Wed 09/01/21	Thu 09/02/12
58	Equip to 360 Level (30m @ 25m/day)	1 day	Fri 09/02/13	Fri 09/02/13
59	Equip station on 360 Level	20 days	Sat 09/02/14	Mon 09/03/09
60	Equip to 390 Level (30m @ 25m/day)	1 day	Tue 09/03/10	Tue 09/03/10
61	Equip station on 390 Level	20 days	Wed 09/03/11	Thu 09/04/02
62	Equip to 420 Level (30m @ 25m/day)	1 day	Fri 09/04/03	Fri 09/04/03
63	Equip station on 420 Level	20 days	Sat 09/04/04	Mon 09/04/27
64	Equip to 450 Level (30m @ 25m/day)	1 day	Tue 09/04/28	Tue 09/04/28
65	Equip station on 450 Level	20 days	Wed 09/04/29	Thu 09/05/21
66	Equip to 480 Level (30m @ 25m/day)	1 day	Fri 09/05/22	Fri 09/05/22
67	Equip station on 480 Level	20 days	Sat 09/05/23	Mon 09/06/15
68	Equip to 510 Level (30m @ 25m/day)	1 day	Tue 09/06/16	Tue 09/06/16
69	Equip station on 510 Level	20 days	Wed 09/06/17	Thu 09/07/09
70	Equip to 540 Level (30m @ 25m/day)	1 day	Fri 09/07/10	Fri 09/07/10
71	Equip station on 540 Level	20 days	Sat 09/07/11	Mon 09/08/03
72	Equip to570 Level (30m @ 25m/day)	1 day	Tue 09/08/04	Tue 09/08/04
73	Equip station on 570 Level	20 days	Wed 09/08/05	Thu 09/08/27
74	Equip to 620 Level (50m @ 25m/day)	2 days	Fri 09/08/28	Sat 09/08/29
75	Equip station on 620 Level - Conveyor Level	30 days	Mon 09/08/31	Sat 09/10/03
76	Equip to 665 Level (45m @ 2m/day)	22 days	Mon 09/10/05	Thu 09/10/29
77	Equip station on 665 Level	5 days	Fri 09/10/30	Wed 09/11/04
78	Changeover Shaft to Permanent Winding	40 days	Thu 09/11/05	Mon 09/12/21
79	**Vent Shaft**	**639 days**	**Sat 06/12/02**	**Tue 08/12/16**
80	Prepare to Sink - Vent Shaft	207 days	Sat 06/12/02	Tue 07/07/31



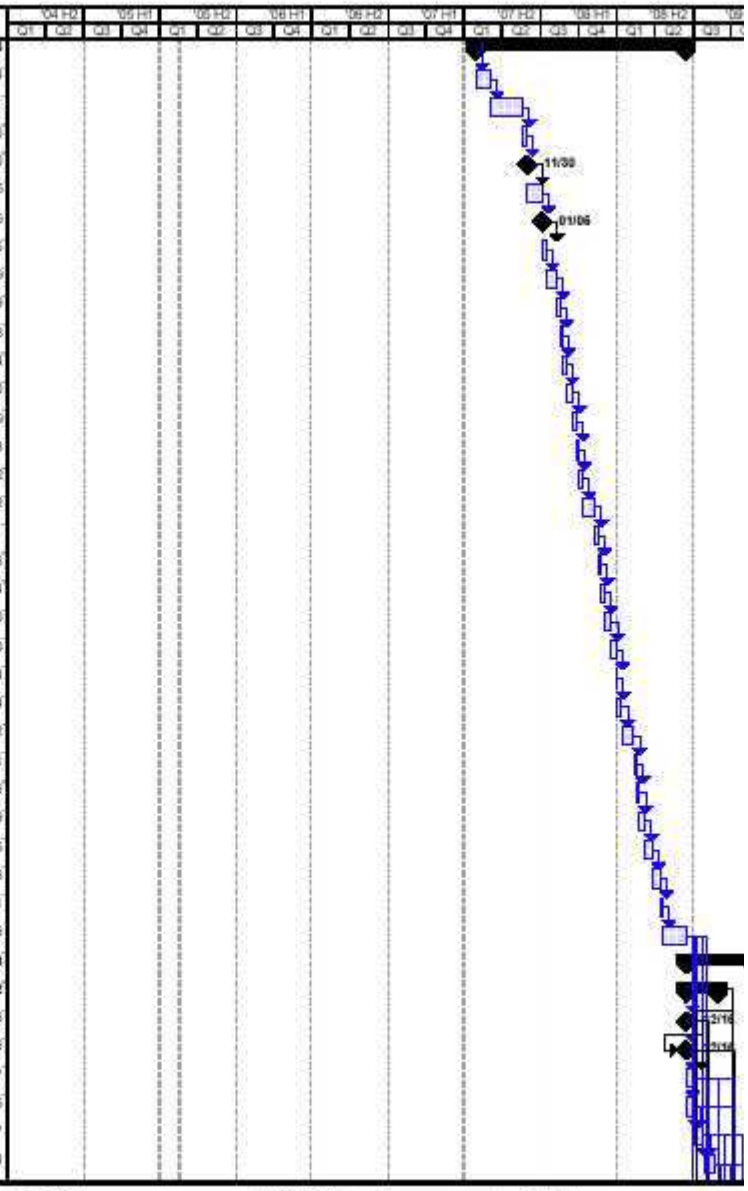
Project: PTM - Elandsfontein Project - Date: Tue 05/06/18

Task		Milestone		Rolled Up Split		External Tasks		Deadline	
Split		Summary		Rolled Up Milestone		Project Summary			
Progress		Rolled Up Task		Rolled Up Progress		External Milestone			

GRAPH 3 - Page 2

ELANDSFONTEIN
PROJECT - SHAFT OPTION

ID	Task Name	Duration	Start	Finish
81	**Sink and Line Vent Shaft**	**432 days**	**Wed 07/08/01**	**Tue 08/12/16**
82	Mobilise Shaft Sinkers	30 days	Wed 07/08/01	Tue 07/09/04
83	Presink to 100m below collar (bc) (@1.5m/day)	67 days	Wed 07/09/05	Wed 07/11/21
84	Sink & line to 130 Level (30 m @ 4m/day)	8 days	Thu 07/11/22	Fri 07/11/30
85	Develop station on 130 Level (80m @ 6m/day)	0 days	Fri 07/11/30	Fri 07/11/30
86	Sink & line to 270 Level (140 m @ 4.5m/day)	31 days	Sat 07/12/01	Sat 08/01/05
87	Develop station on 270 Level (20m @ 6m/day)	0 days	Sat 08/01/05	Sat 08/01/05
88	Sink & line to 300 Level (30 m @ 4m/day)	8 days	Mon 08/01/07	Tue 08/01/15
89	Develop station on 300 Level (130m @ 6m/day)	22 days	Wed 08/01/16	Sat 08/02/09
90	Sink & line to 330 Level (30 m @ 4m/day)	8 days	Mon 08/02/11	Tue 08/02/19
91	Develop station on 330 Level (20m @ 6m/day)	4 days	Wed 08/02/20	Sat 08/02/23
92	Sink & line to 360 Level (30 m @ 4m/day)	8 days	Mon 08/02/25	Tue 08/03/04
93	Develop station on 360 Level (80m @ 6m/day)	14 days	Wed 08/03/05	Thu 08/03/20
94	Sink & line to 390 Level (30 m @ 4m/day)	8 days	Fri 08/03/21	Sat 08/03/29
95	Develop station on 390 Level (20m @ 6m/day)	4 days	Mon 08/03/31	Thu 08/04/03
96	Sink & line to 420 Level (30 m @ 4m/day)	8 days	Fri 08/04/04	Sat 08/04/12
97	Develop station on 420 Level (150m @ 6m/day)	25 days	Mon 08/04/14	Mon 08/05/12
98	Sink & line to 450 Level (30 m @ 4m/day)	8 days	Tue 08/05/13	Wed 08/05/21
99	Develop station on 450 Level (20m @ 6m/day)	4 days	Thu 08/05/22	Mon 08/05/26
100	Sink & line to 480 Level (30 m @ 4m/day)	8 days	Tue 08/05/27	Wed 08/06/04
101	Develop station on 480 Level (80m @ 6m/day)	14 days	Thu 08/06/05	Fri 08/06/20
102	Sink & line to 510 Level (30 m @ 4m/day)	8 days	Sat 08/06/21	Mon 08/06/30
103	Develop station on 510 Level (20m @ 6m/day)	4 days	Tue 08/07/01	Fri 08/07/04
104	Sink & line to 540 Level (30 m @ 4m/day)	8 days	Sat 08/07/05	Mon 08/07/14
105	Develop station on 540 Level (150m @ 6m/day)	25 days	Tue 08/07/15	Tue 08/08/12
106	Sink & line to 570 Level (30 m @ 4m/day)	8 days	Wed 08/08/13	Thu 08/08/21
107	Develop station on 570 Level (20m @ 6m/day)	4 days	Fri 08/08/22	Tue 08/08/26
108	Sink & line to 620 Level (50 m @ 4m/day)	12 days	Wed 08/08/27	Tue 08/09/09
109	Develop station on 620 Level (80m @ 6m/day)	14 days	Wed 08/09/10	Thu 08/09/25
110	Sink & line to 665 Level (45 m @ 2.5m/day)	18 days	Fri 08/09/26	Thu 08/10/16
111	Develop station on 665 Level (20m @ 6m/day)	4 days	Fri 08/10/17	Tue 08/10/21
112	Changeover to development conditions	48 days	Wed 08/10/22	Tue 08/12/16
113	**In circle development**	**136 days**	**Tue 08/12/16**	**Sat 09/06/23**
114	**Critical in-circle development**	**66 days**	**Tue 08/12/16**	**Mon 09/03/02**
115	130 Level (2 ends @ 3m/day,75m)	0 days	Tue 08/12/16	Tue 08/12/16
116	270 Level (5 ends @ 3m/day,255m)	0 days	Tue 08/12/16	Tue 08/12/16
117	300 Level (5 ends @ 3m/day,280m)	19 days	Wed 08/12/17	Wed 09/01/07
118	330 Level (5 ends @ 3m/day,255m)	17 days	Wed 08/12/17	Mon 09/01/05
119	360 Level (5 ends @ 3m/day,280m)	19 days	Tue 09/01/06	Tue 09/01/27
120	390 Level (5 ends @ 3m/day,330m)	22 days	Wed 09/01/28	Sat 09/02/21



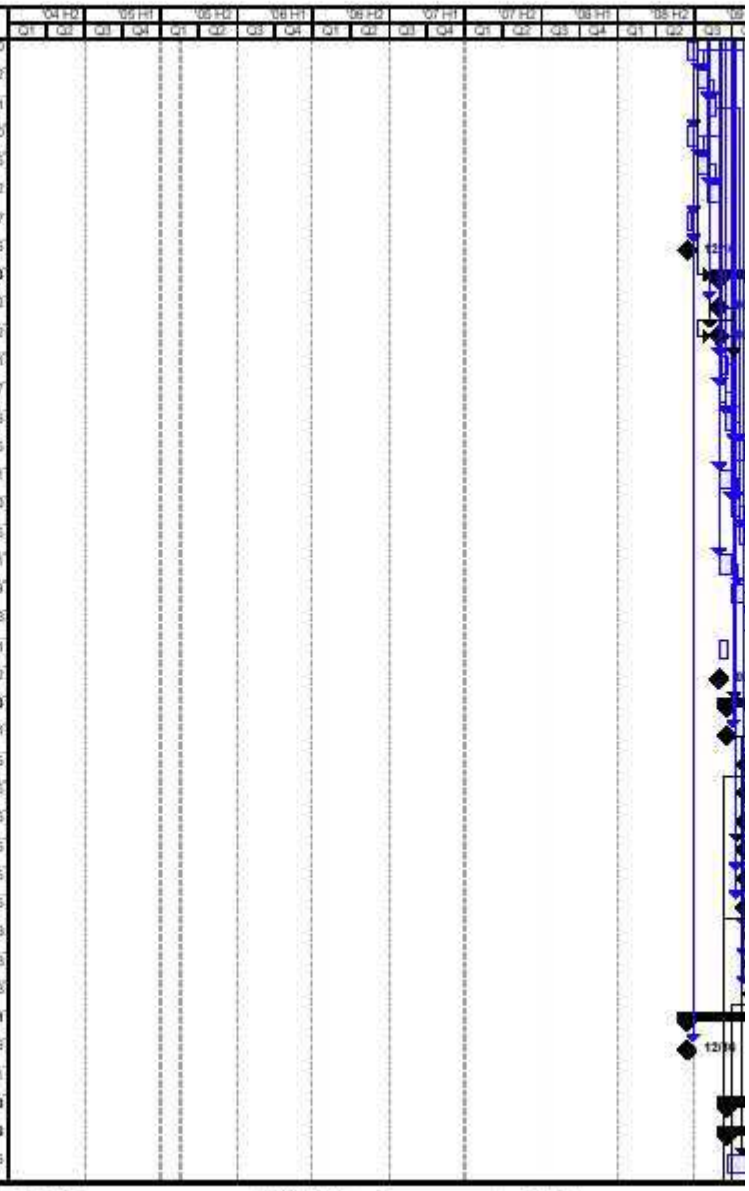
Project: PTM - Elandsfontein Project - Date: Tue 05/06/18

Task	Milestone	Rolled Up Split	External Tasks	Deadline
Split	Summary	Rolled Up Milestone	Project Summary	
Progress	Rolled Up Task	Rolled Up Progress	External Milestone	

GRAPH 3 - Page 3

ELANDSFONTEIN
PROJECT - SHAFT OPTION

ID	Task Name	Duration	Start	Finish
121	420 Level (35ends @ 3m/day,330m)	22 days	Wed 08/12/17	Sat 09/01/10
122	450 Level (5 ends @ 3m/day,280m)	19 days	Mon 09/01/12	Mon 09/02/02
123	480 Level (5 ends @ 3m/day, 255m)	17 days	Tue 09/02/03	Sat 09/02/21
124	510 Level (5 ends @ 3m/day, 330m)	22 days	Wed 08/12/17	Sat 09/01/10
125	540 Level (5 ends @ 3m/day, 330m)	22 days	Mon 09/01/12	Thu 09/02/05
126	570 Level (5 ends @ 3m/day, 305m)	21 days	Fri 09/02/06	Mon 09/03/02
127	620 Level (2 ends @ 3m/day, 55m)	10 days	Wed 08/12/17	Sat 08/12/27
128	865 Level (2 ends @ 3m/day, 0m)	0 days	Tue 08/12/16	Tue 08/12/16
129	**Completion of in-circle development**	**71 days**	**Mon 09/03/02**	**Sat 09/06/23**
130	130 Level (2 ends @ 3m/day,75m)	0 days	Mon 09/03/02	Mon 09/03/02
131	270 Level (2 ends @ 3m/day,75m)	0 days	Mon 09/03/02	Mon 09/03/02
132	300 Level (2 ends @ 3m/day,100m)	17 days	Tue 09/03/03	Sat 09/03/21
133	330 Level (2 ends @ 3m/day,75m)	13 days	Tue 09/03/03	Tue 09/03/17
134	360 Level (3 ends @ 3m/day,100m)	17 days	Wed 09/03/18	Mon 09/04/06
135	390 Level (3 ends @ 3m/day,150m)	25 days	Tue 09/04/07	Tue 09/05/05
136	420 Level (3 ends @ 3m/day,150m)	25 days	Tue 09/03/03	Tue 09/03/31
137	450 Level (2 ends @ 3m/day,100m)	17 days	Wed 09/04/01	Mon 09/04/20
138	480 Level (2 ends @ 3m/day, 75m)	13 days	Tue 09/04/21	Tue 09/05/05
139	510 Level (2 ends @ 3m/day, 150m)	25 days	Tue 09/03/03	Tue 09/03/31
140	540 Level (2 ends @ 3m/day, 150m)	25 days	Wed 09/04/01	Wed 09/04/29
141	570 Level (2 ends @ 3m/day, 125m)	21 days	Thu 09/04/30	Sat 09/05/23
142	620 Level (2 ends @ 3m/day, 100m)	17 days	Tue 09/03/03	Sat 09/03/21
143	865 Level (2 ends @ 3m/day, 0m)	0 days	Mon 09/03/02	Mon 09/03/02
144	**Start Primary Development to first Crosscut**	**64 days**	**Sat 09/03/21**	**Sat 09/06/23**
145	300 Level	0 days	Sat 09/03/21	Sat 09/03/21
146	330 Level	0 days	Tue 09/05/05	Tue 09/05/05
147	360 Level	0 days	Tue 09/05/05	Tue 09/05/05
148	390 Level	0 days	Tue 09/05/05	Tue 09/05/05
149	420 Level	0 days	Tue 09/05/05	Tue 09/05/05
150	450 Level	0 days	Tue 09/05/05	Tue 09/05/05
151	480 Level	0 days	Tue 09/05/05	Tue 09/05/05
152	510 Level	0 days	Sat 09/05/23	Sat 09/05/23
153	540 Level	0 days	Sat 09/05/23	Sat 09/05/23
154	570 Level	0 days	Sat 09/05/23	Sat 09/05/23
155	**Completion of Shafts**	**317 days**	**Tue 08/12/16**	**Mon 09/12/21**
156	**Complete Vent Shaft**	0 days	Tue 08/12/16	Tue 08/12/16
157	**Complete Main Shaft**	0 days	Mon 09/12/21	Mon 09/12/21
158	**Primary Development on Levels**	**660 days**	**Mon 09/03/23**	**Sat 11/04/30**
159	**Horizontal**	**610 days**	**Mon 09/03/23**	**Sat 10/11/06**
160	300 Level - Zone 3	100 days	Mon 09/03/23	Thu 09/07/16



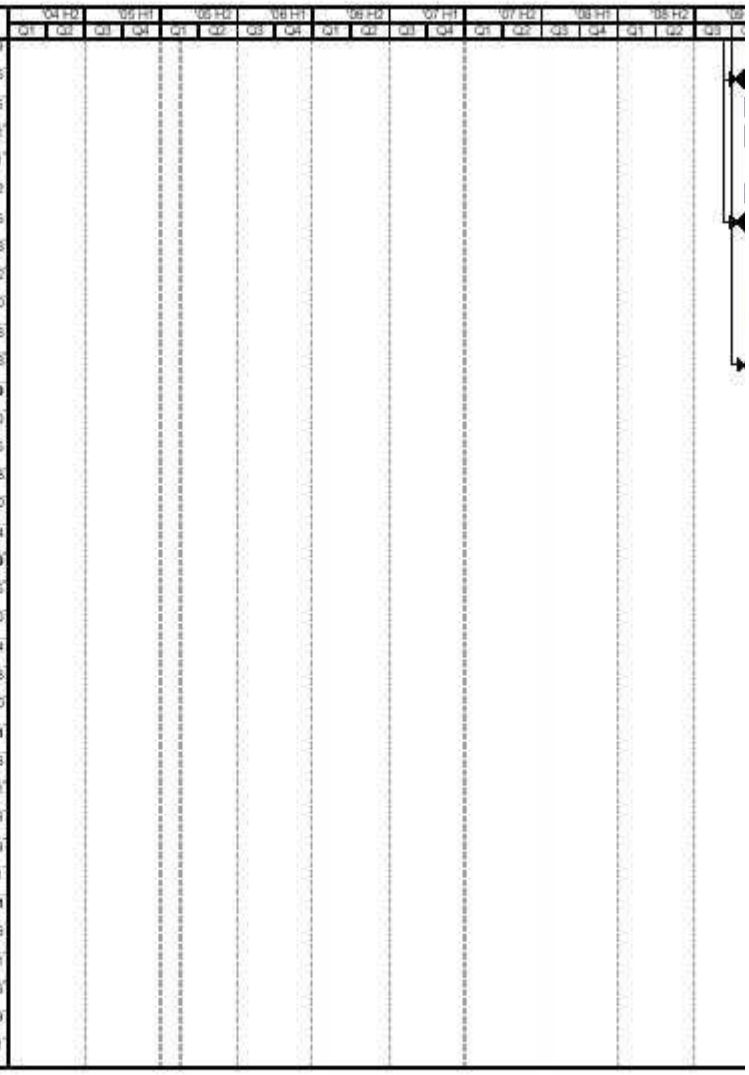
Project: PTM - Elandsfontein Project - Date: Tue 05/06/18

Task | Split | Progress | Milestone | Summary | Rolled Up Task | Rolled Up Split | Rolled Up Milestone | Rolled Up Progress | External Tasks | Project Summary | External Milestone | Deadline

GRAPH 3 - Page 4

ELANDSFONTEIN
PROJECT - SHAFT OPTION

ID	Task Name	Duration	Start	Finish
161	300 Level - Zone 5	125 days	Fri 09/07/17	Wed 09/12/06
162	330 Level	0 days	Tue 09/05/05	Tue 09/05/05
163	360 Level - Zone 6	175 days	Wed 09/05/06	Wed 09/11/25
164	390 Level - Zone 4	75 days	Wed 09/05/06	Fri 09/07/31
165	420 Level - Zone 11	200 days	Tue 09/12/22	Wed 10/08/11
166	450 Level - Zone 3	120 days	Wed 09/05/06	Tue 09/09/22
167	480 Level	0 days	Tue 09/05/05	Tue 09/05/05
168	510 Level - Zone 5	150 days	Mon 09/10/12	Sat 10/04/03
169	510 Level - Zone 6	210 days	Mon 09/10/12	Sat 10/06/12
170	540 Level - Zone 4	120 days	Mon 09/05/25	Sat 09/10/10
171	540 Level - Zone 11	275 days	Tue 09/12/22	Sat 10/11/06
172	570 Level	0 days	Sat 09/05/23	Sat 09/05/23
173	**Decline**	**660 days**	**Fri 09/07/17**	**Sat 11/04/30**
174	Zone 4	225 days	Sat 09/08/01	Tue 10/04/20
175	Zone 3	225 days	Fri 09/07/17	Mon 10/04/05
176	Zone 5	225 days	Thu 09/12/10	Sat 10/08/28
177	Zone 10	225 days	Thu 10/08/12	Sat 11/04/30
178	Zone 6	225 days	Thu 09/11/26	Sat 10/08/14
179	**Initial Stoping**	**336 days**	**Mon 10/04/05**	**Sat 11/04/30**
180	Zone 3	0 days	Mon 10/04/05	Mon 10/04/05
181	Zone 4	0 days	Tue 10/04/20	Tue 10/04/20
182	Zone 6	0 days	Sat 10/08/14	Sat 10/08/14
183	Zone 5	0 days	Sat 10/08/28	Sat 10/08/28
184	Zone 10	0 days	Sat 11/04/30	Sat 11/04/30
185	**Stoping Ramp up in Production**	**710 days**	**Tue 10/04/06**	**Wed 12/07/11**
186	Zone 3	375 days	Tue 10/04/06	Thu 11/06/16
187	Zone 4	375 days	Wed 10/04/21	Fri 11/07/01
188	Zone 6	375 days	Mon 10/08/16	Wed 11/10/26
189	Zone 5	375 days	Mon 10/08/30	Wed 11/11/09
190	Zone 10	375 days	Mon 11/05/02	Wed 12/07/11
191	**Block / Stope in full production**	**336 days**	**Thu 11/06/16**	**Wed 12/07/11**
192	Zone 3	0 days	Thu 11/06/16	Thu 11/06/16
193	Zone 4	0 days	Fri 11/07/01	Fri 11/07/01
194	Zone 6	0 days	Wed 11/10/26	Wed 11/10/26
195	Zone 5	0 days	Wed 11/11/09	Wed 11/11/09
196	Zone 10	0 days	Wed 12/07/11	Wed 12/07/11



Project: PTM - Elandsfontein Project - Date: Tue 05/06/18

Task | Split | Progress | Milestone | Summary | Rolled Up Task | Rolled Up Split | Rolled Up Milestone | Rolled Up Progress | External Tasks | Project Summary | External Milestone | Deadline

ITEM 26: ILLUSTRATIONS

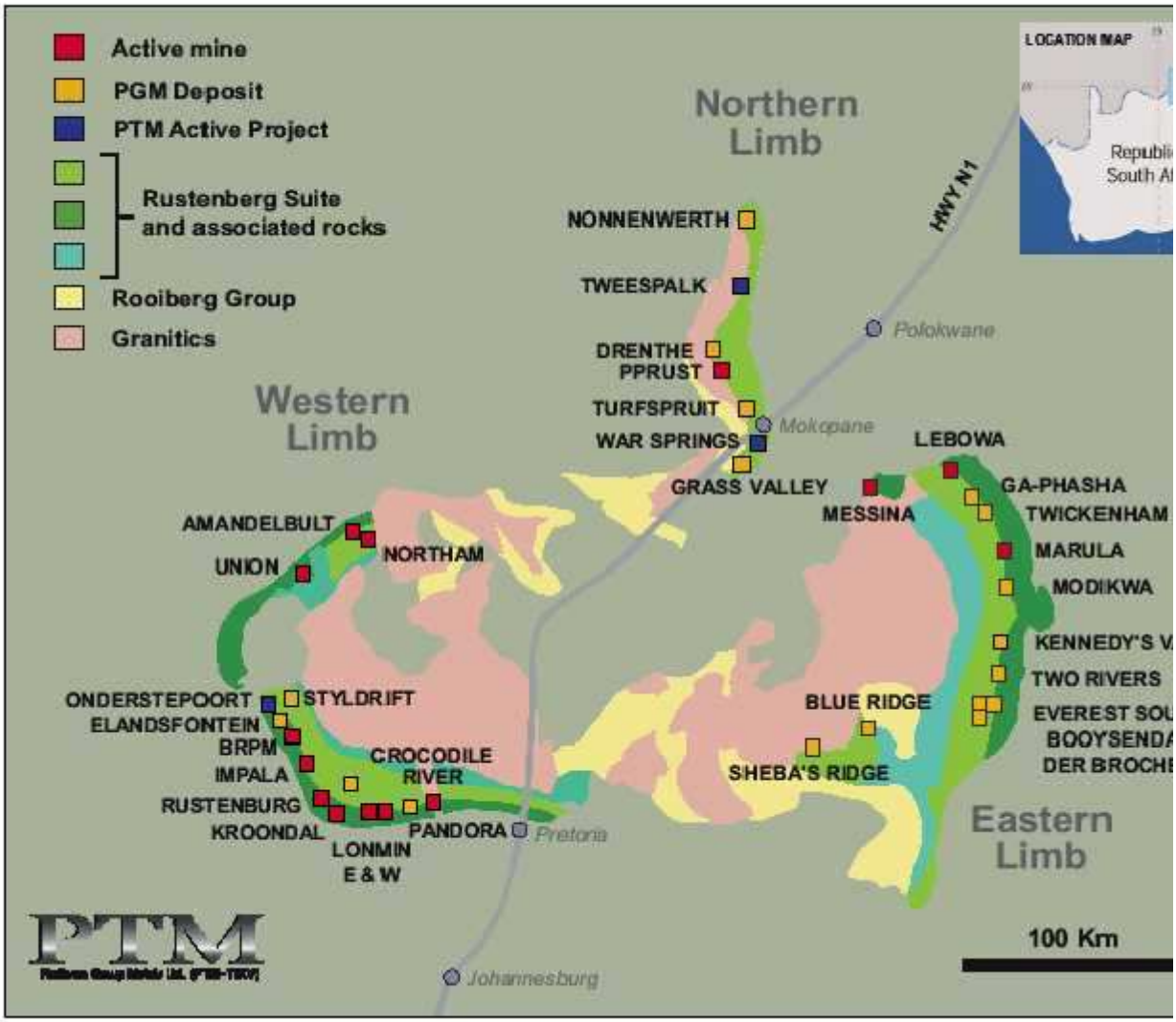


Diagram 1: Illustration of the Eastern, Western and Northern Limb of the Bushveld Igneous Com

WESTERN BUSHVELD JOINT VENTURE

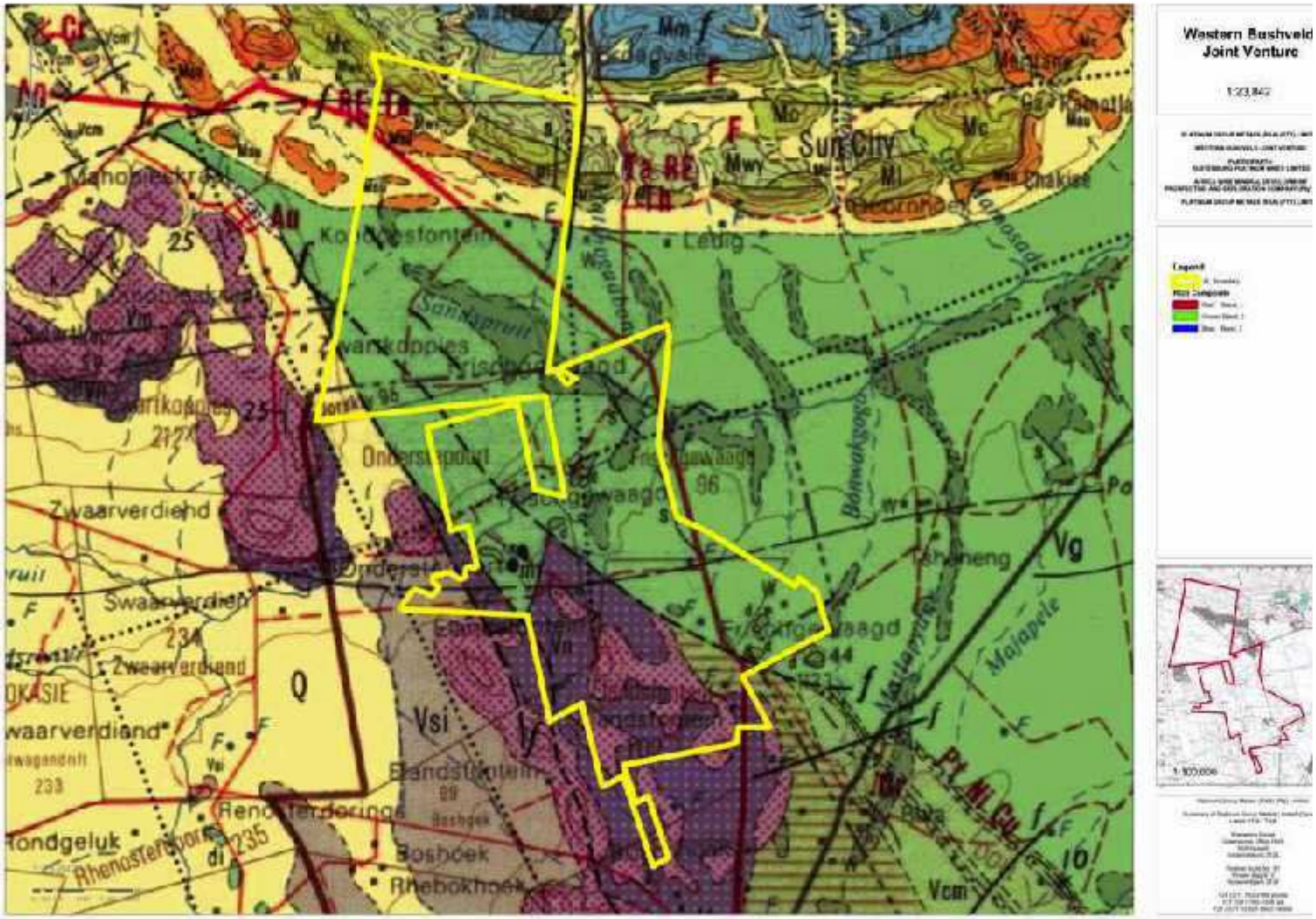


Diagram 2: Illustration of the Western Bushveld Joint Venture Location

WESTERN BUSHVELD JOINT VENTURE

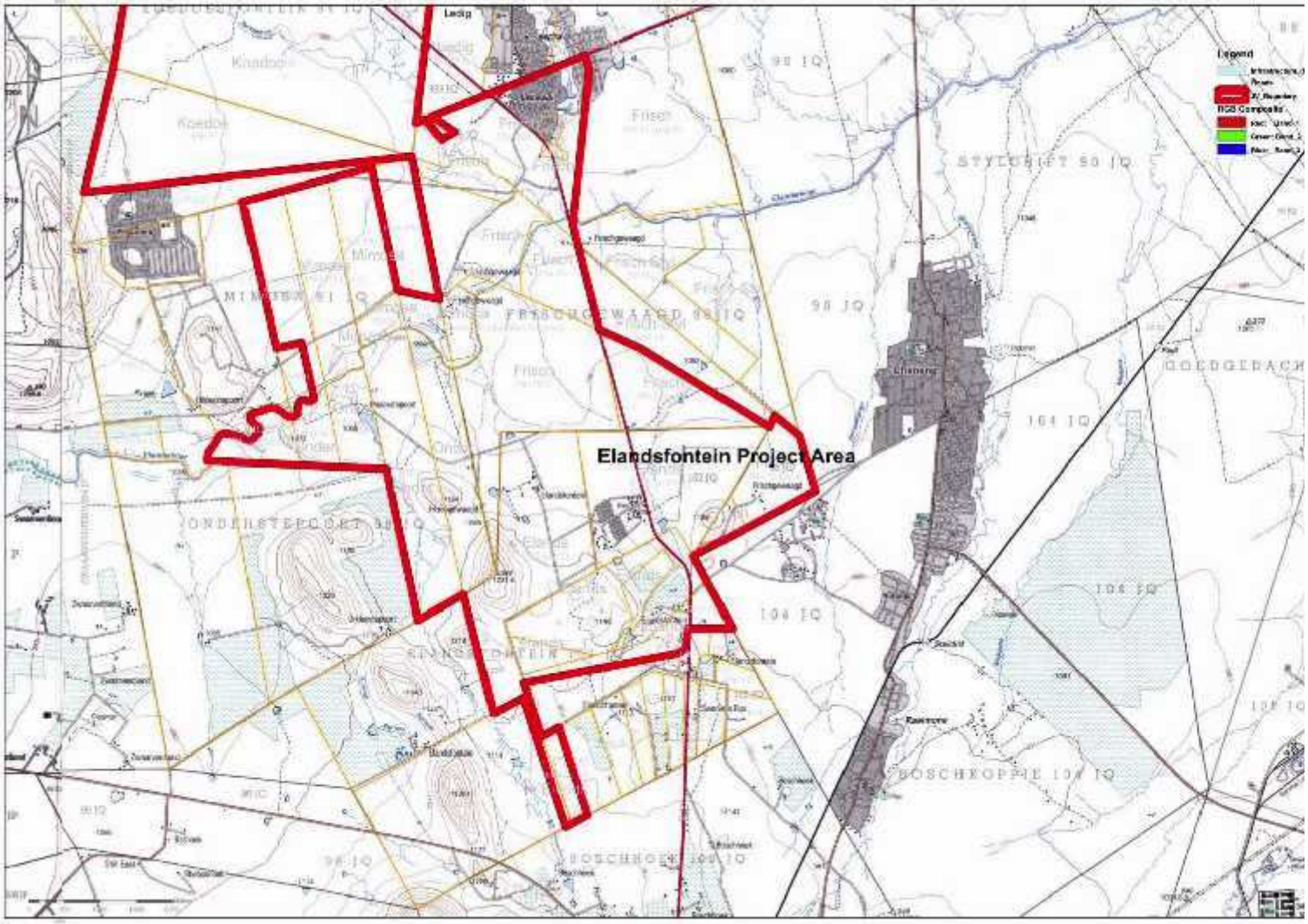


Diagram 3a: Topocadastral Map of the Surrounding Area.

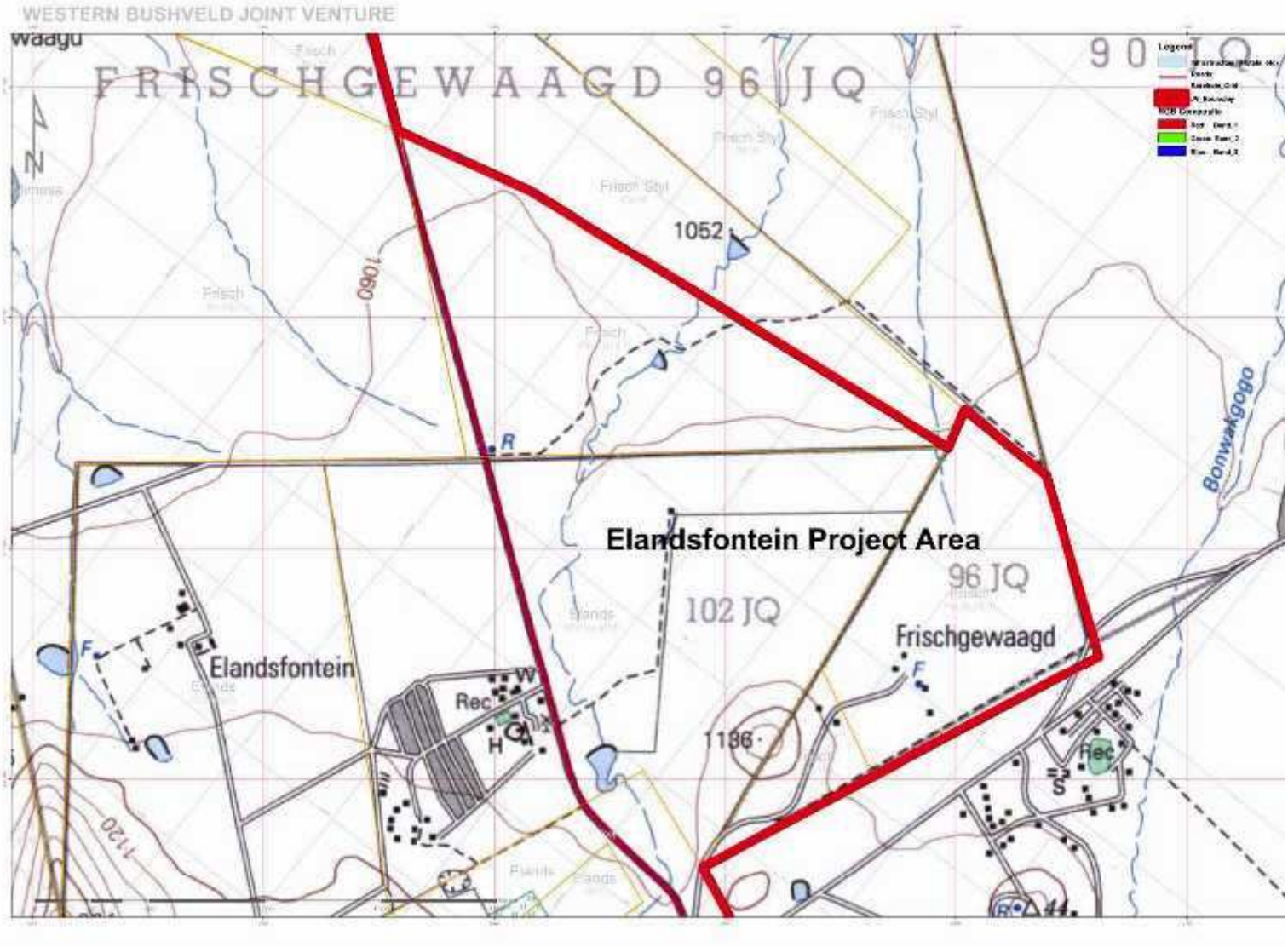


Diagram 3b: Illustration of the Elandsfontein Project Area

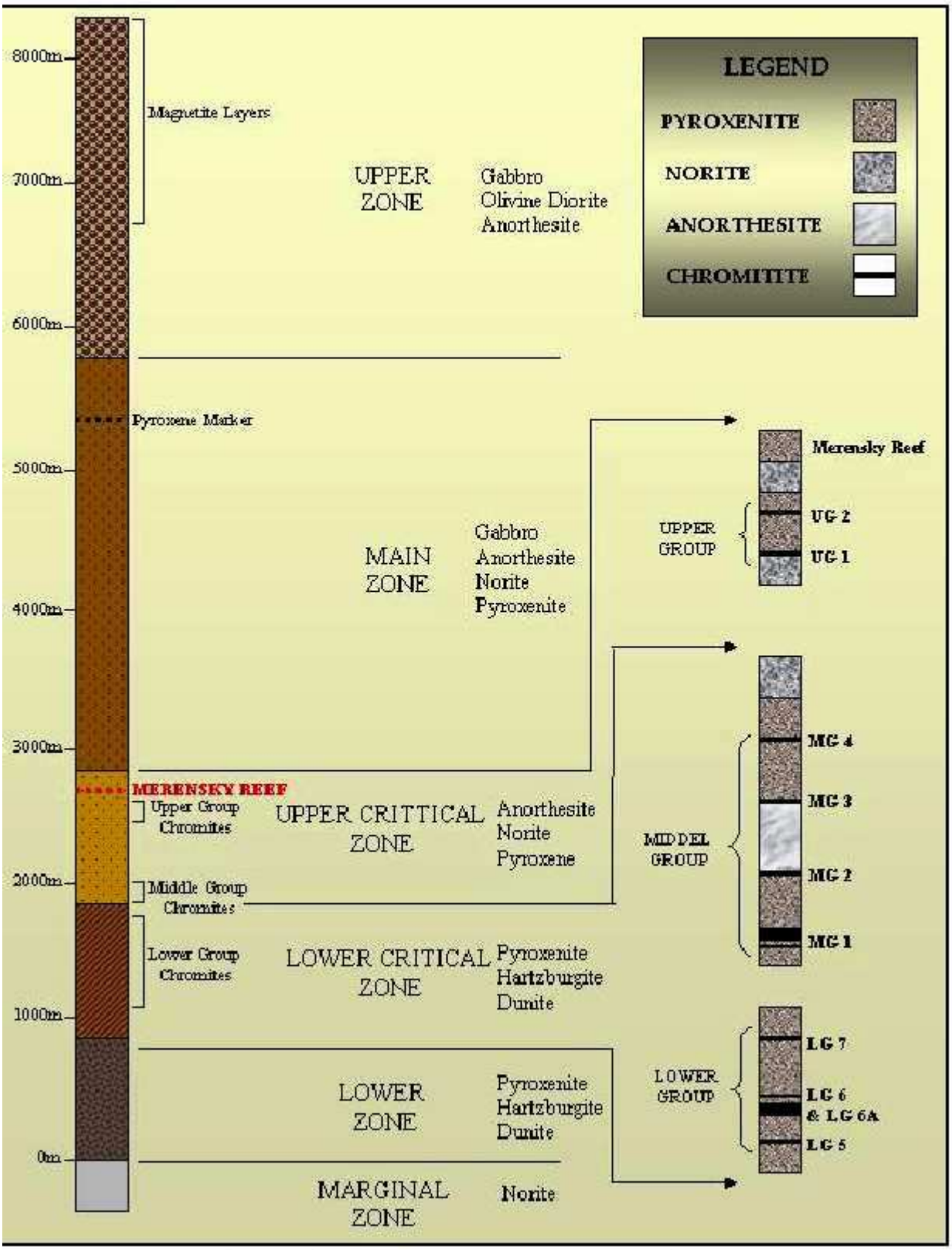


Diagram 4a: General Stratigraphy of the Western Bushveld Sequence (BRPM)

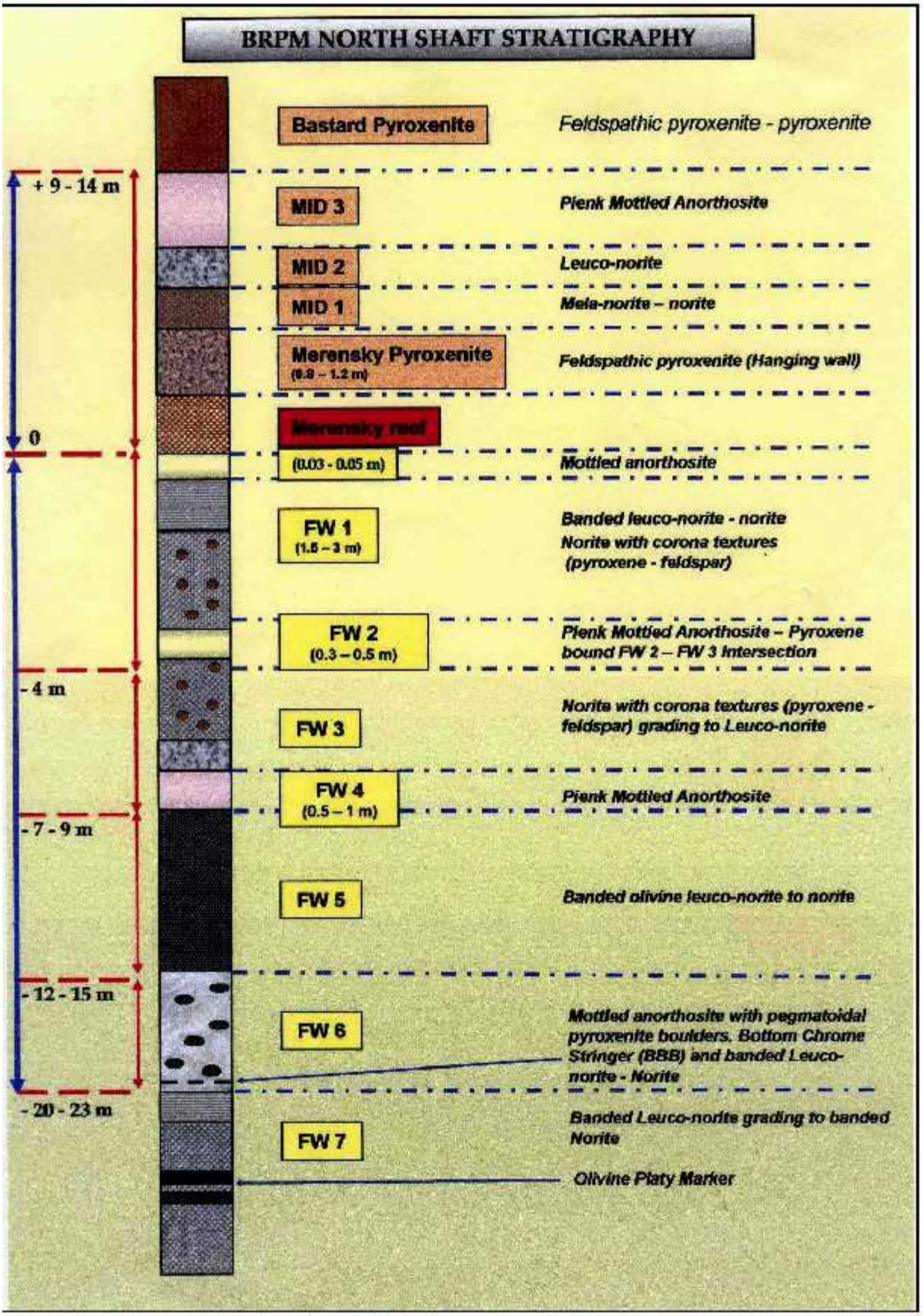


Diagram 4b: Detailed Bastard, Merensky and Footwall Stratigraphy on BRPM.

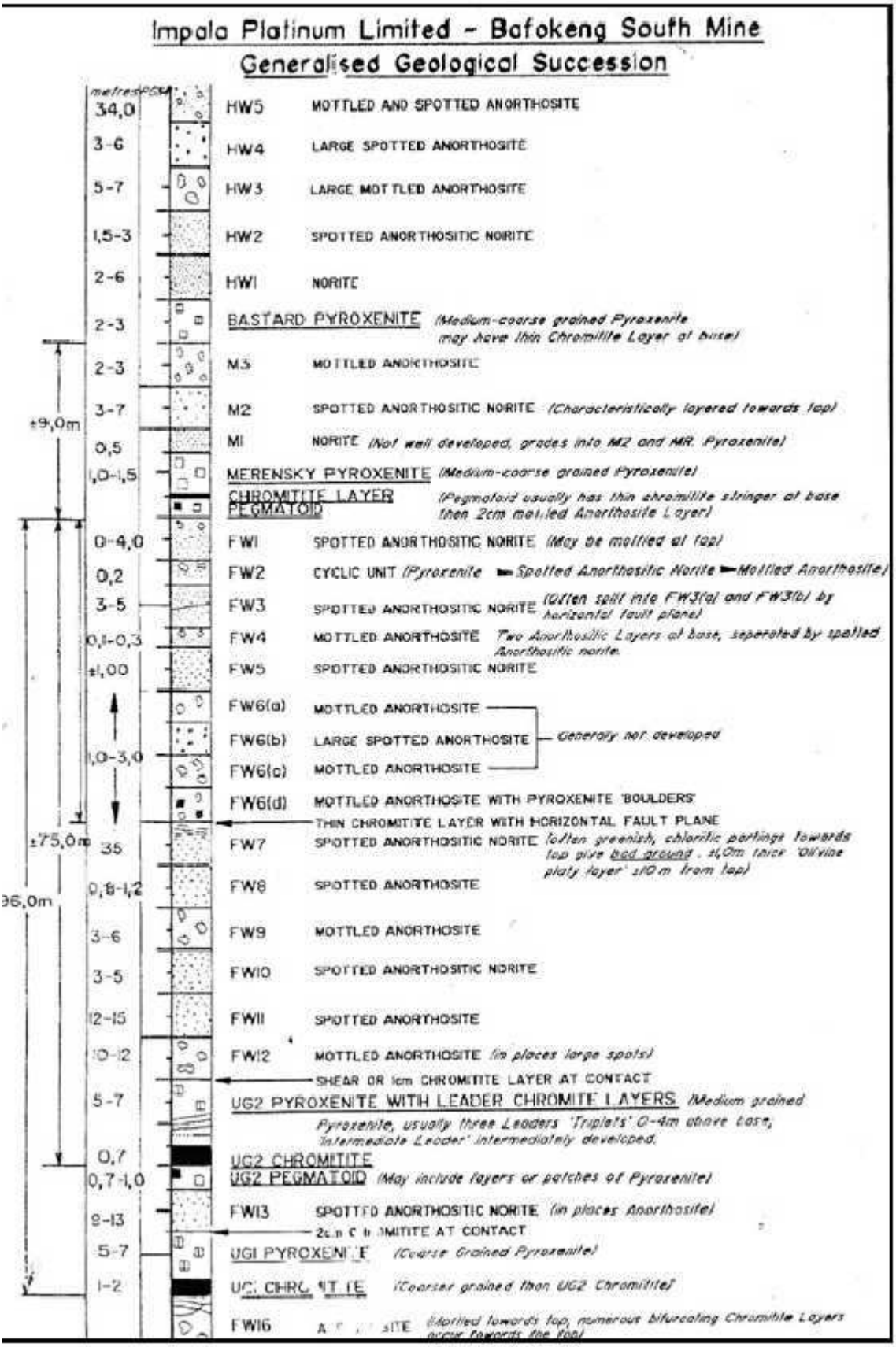


Diagram 4c: Generalised Geological Section - BRPM

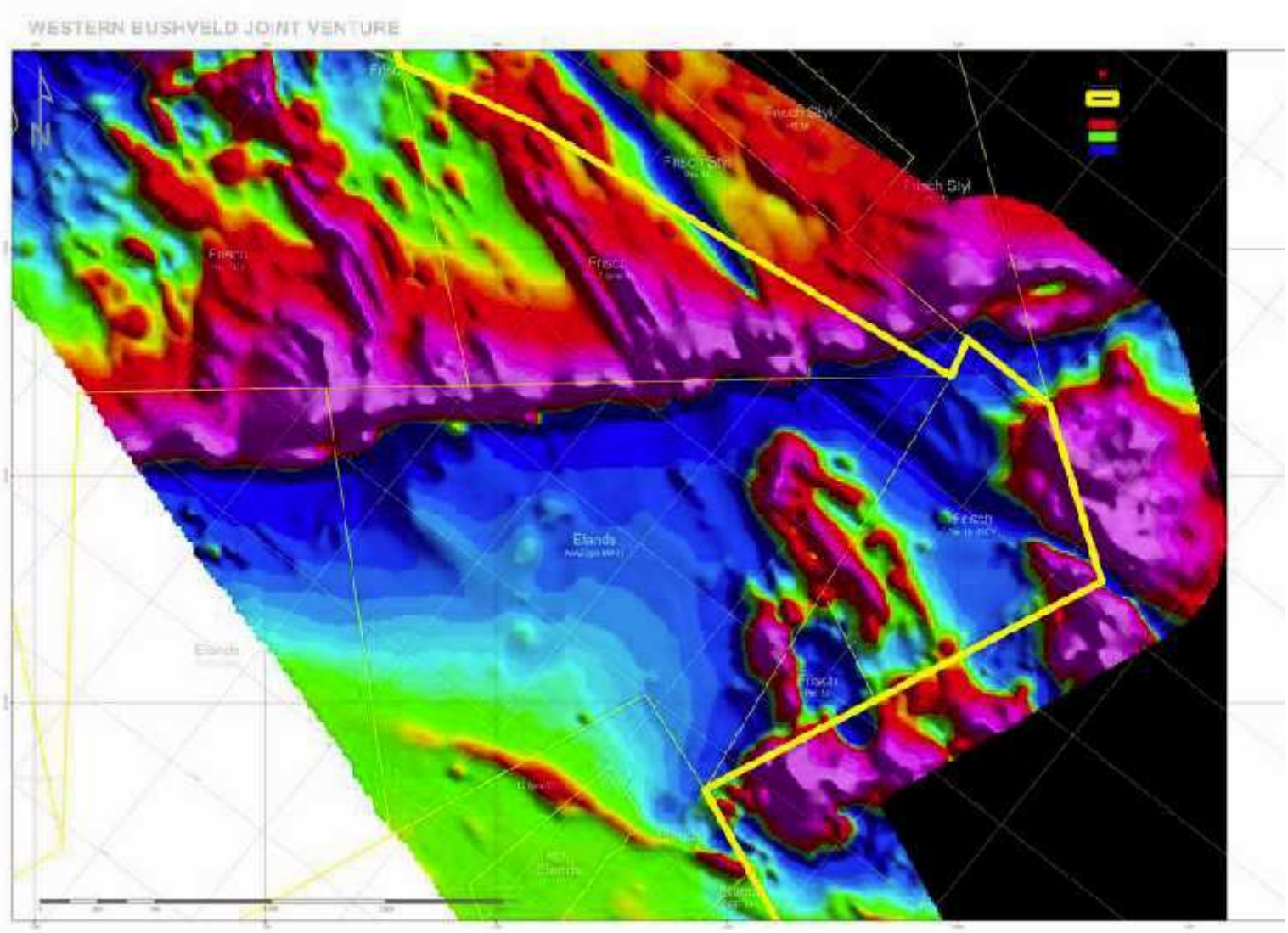


Diagram 5a: Geophysics – Total Magnetic field

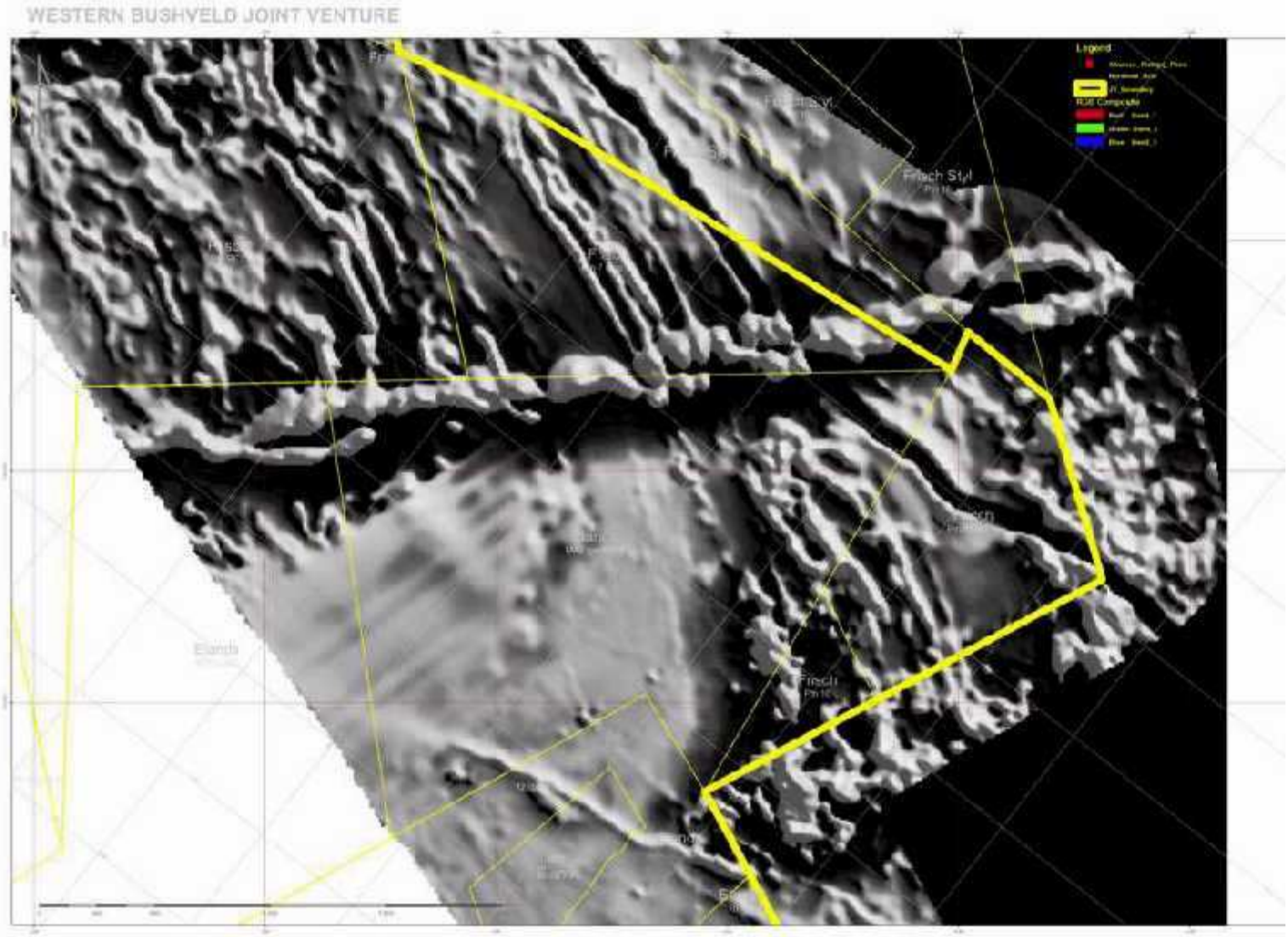


Diagram 5b: Geophysics – Greyed image of 1st derivative of Magnetic Field

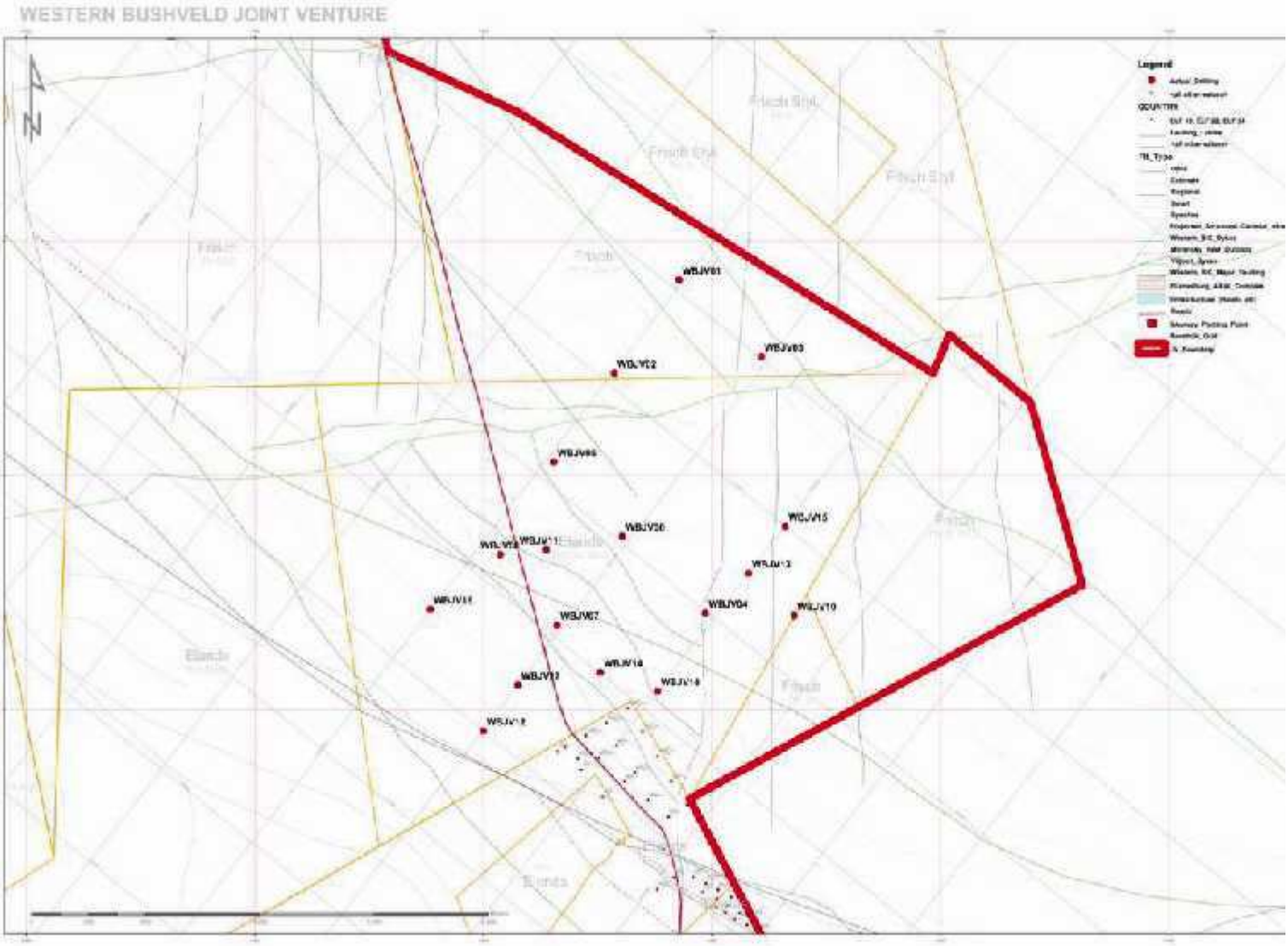


Diagram 6a: Project Showing Drilling Coverage by PTM

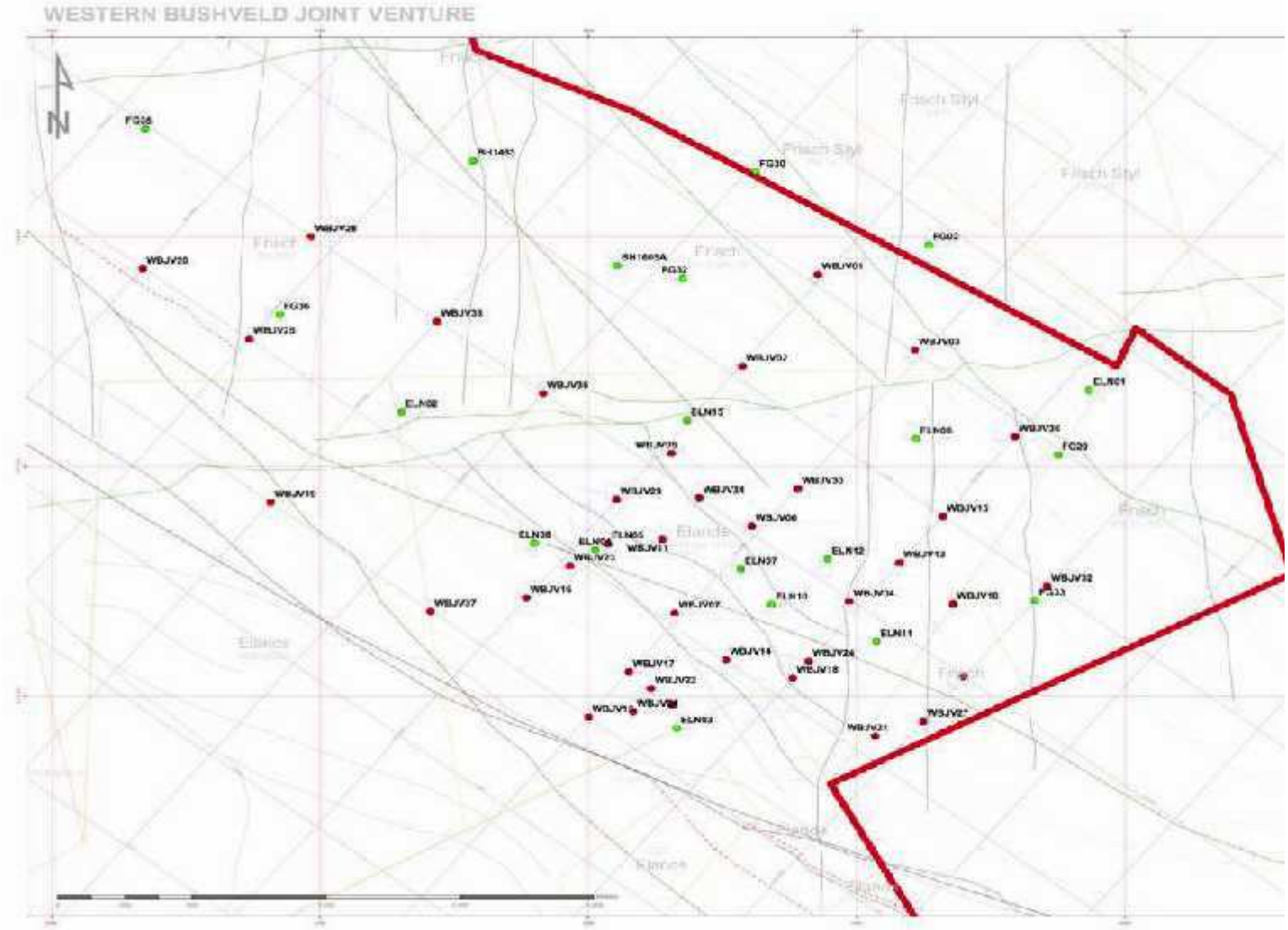


Diagram 6b: Project Showing Drilling Coverage by Anglo and PTM

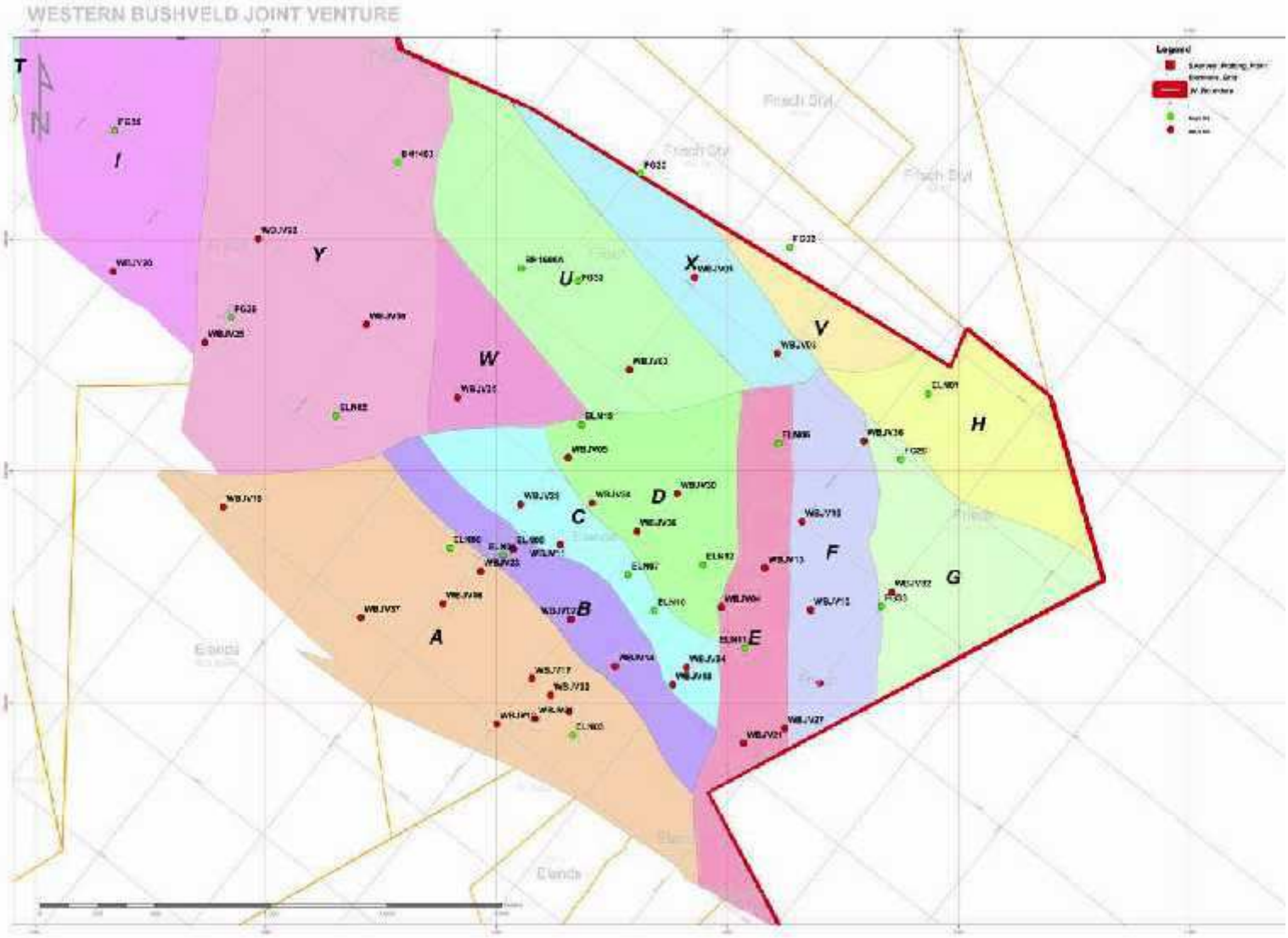


Diagram 7: Structural blocks interpreted from Magnetics, Drilling and Publishing Info

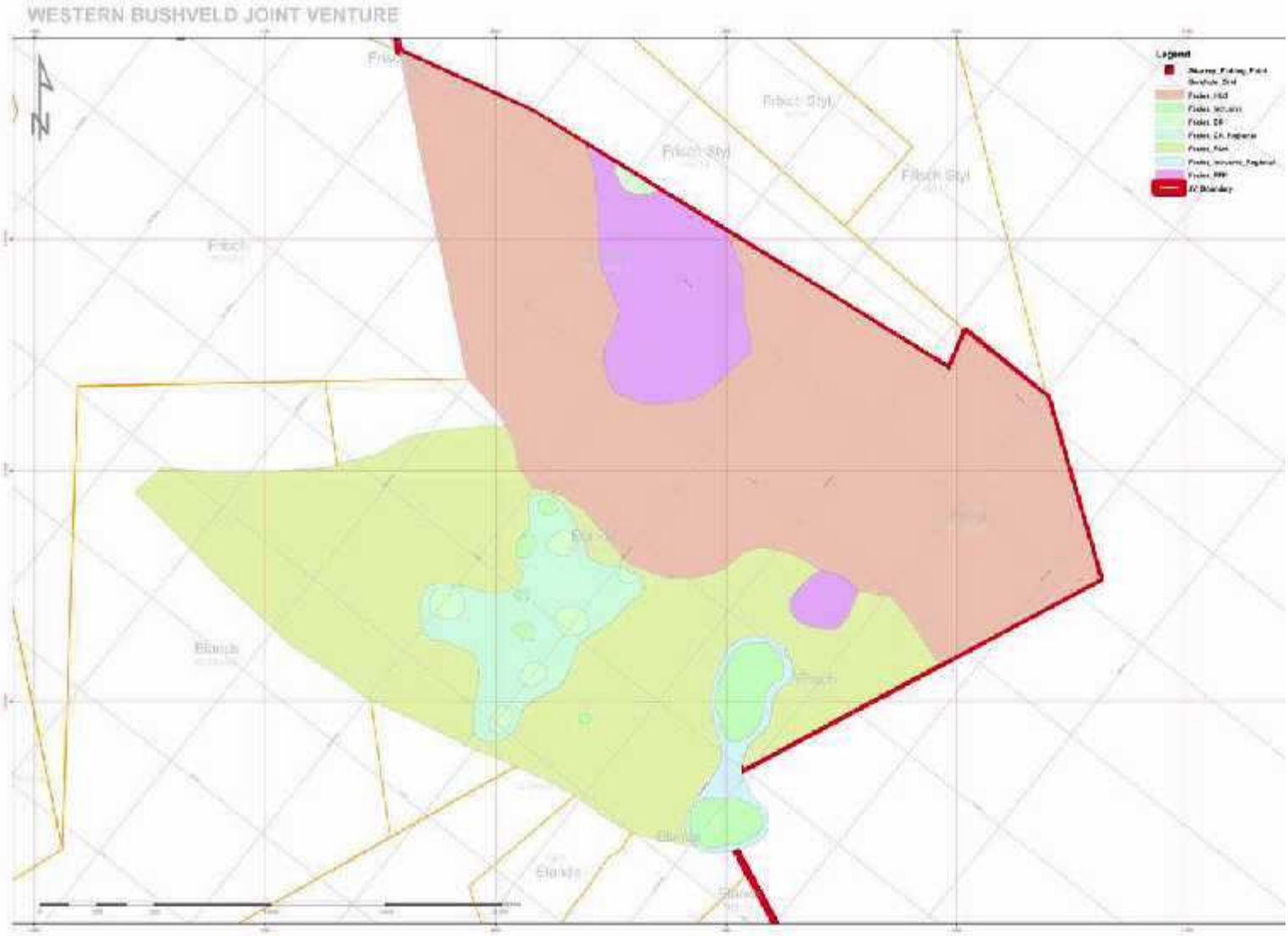


Diagram 8a: Merensky Reef Facies plan

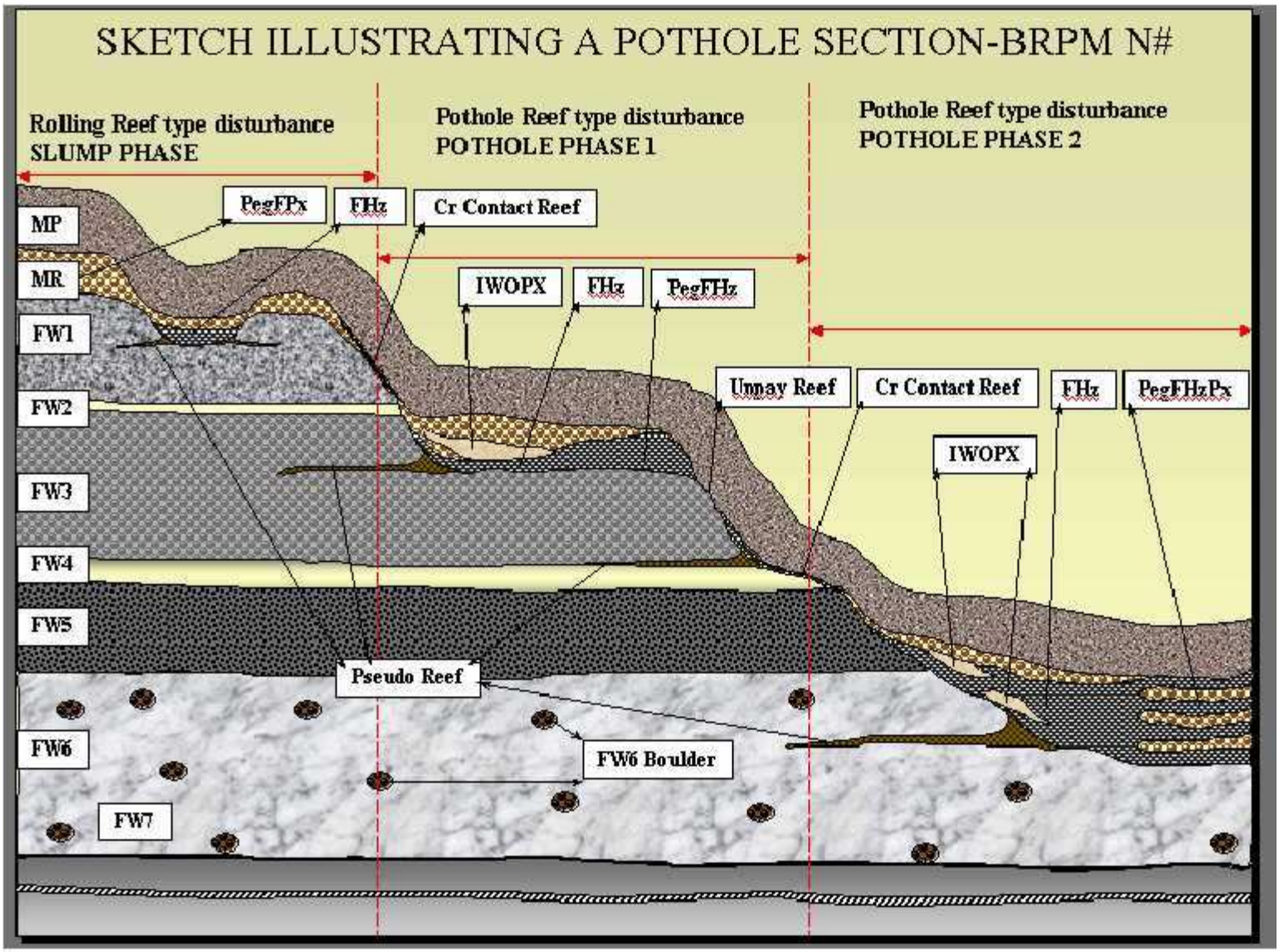


Diagram 8b: Merensky Reef Geological Facies Model (BRPM)

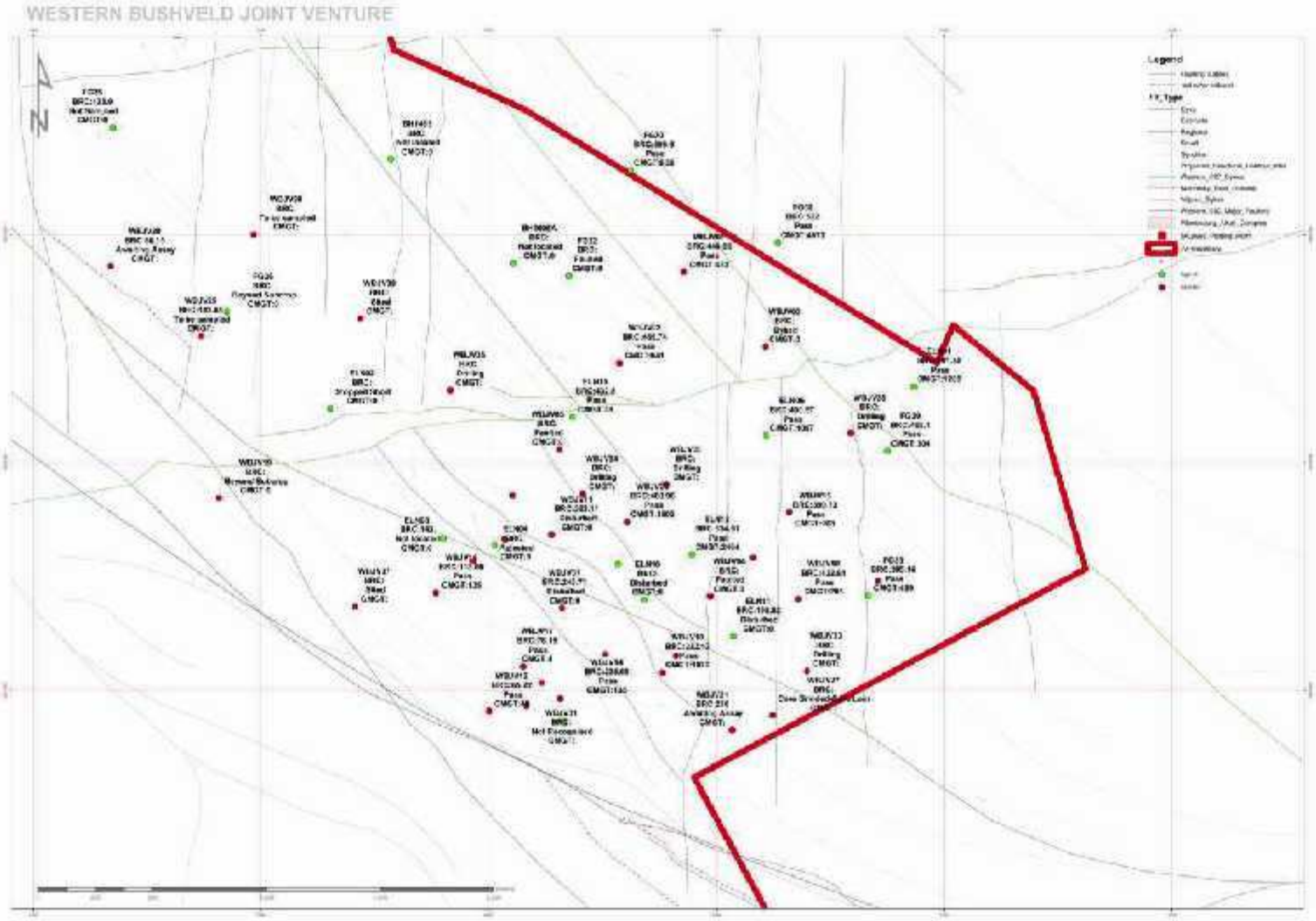


Diagram 9: Merensky Reef intersections, showing Bottom Reef Contact and Content (cm gt)

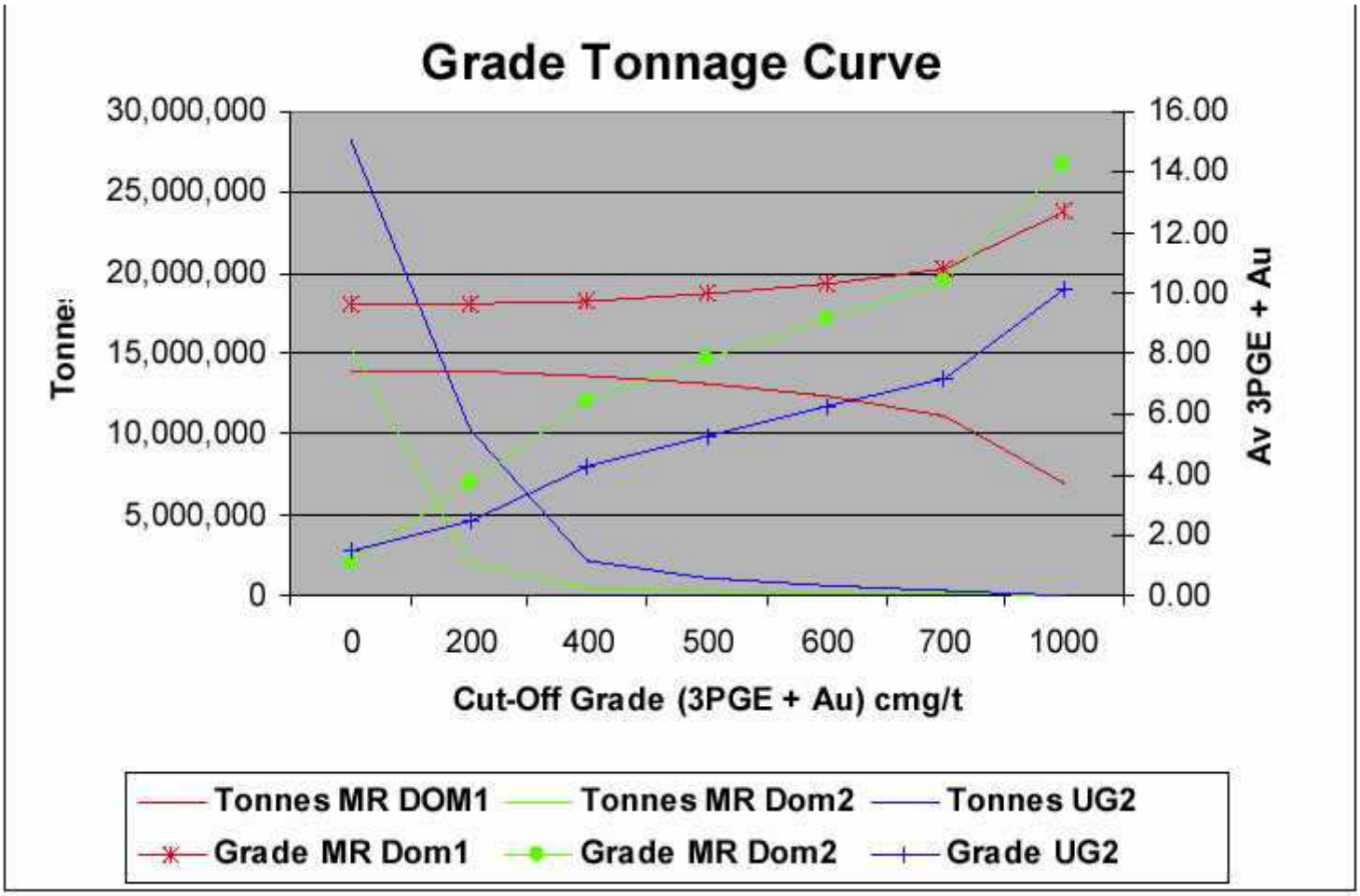


Diagram 10: Grade Tonnage Curve

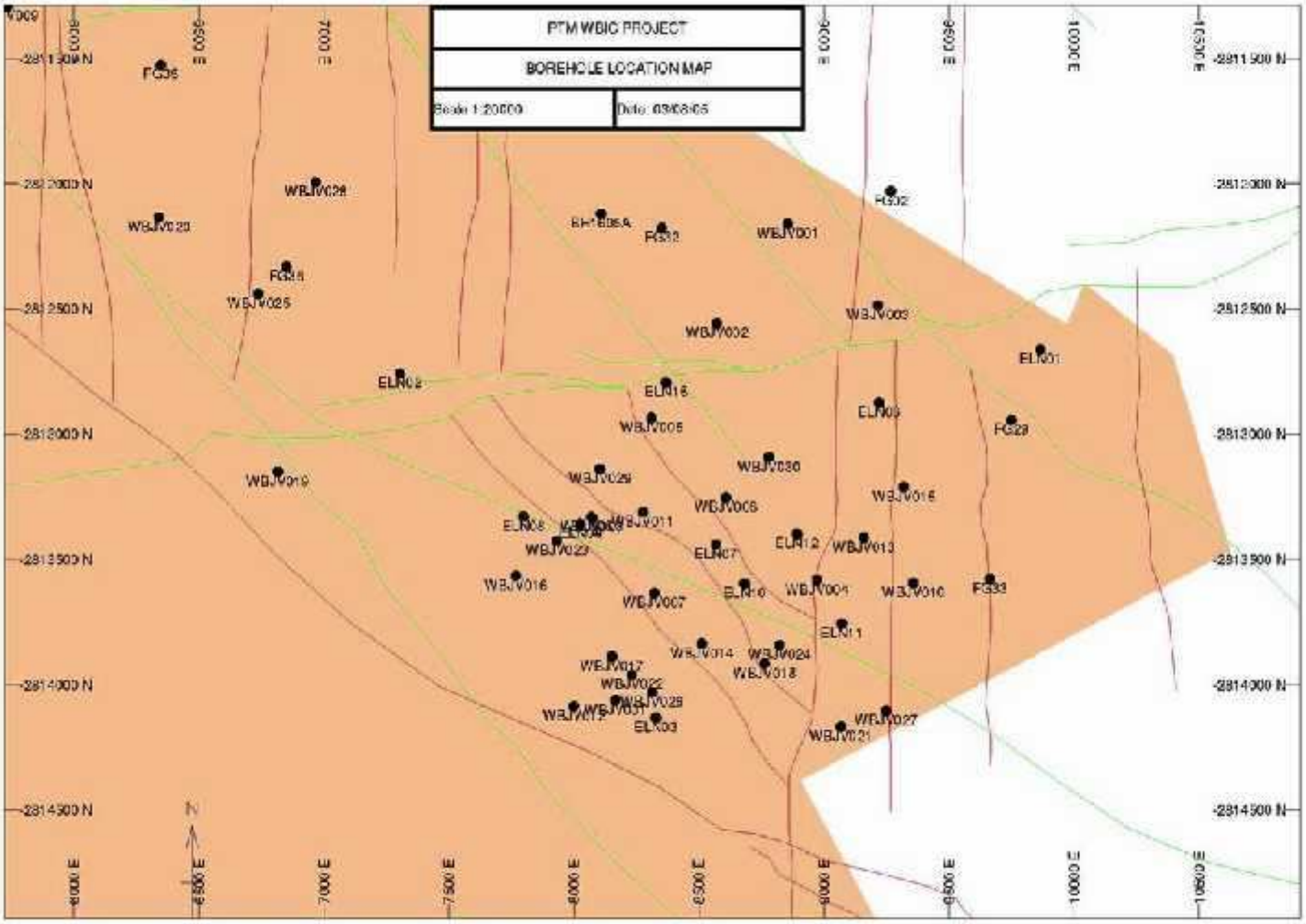


Diagram 11: Borehole Location used for Evaluation

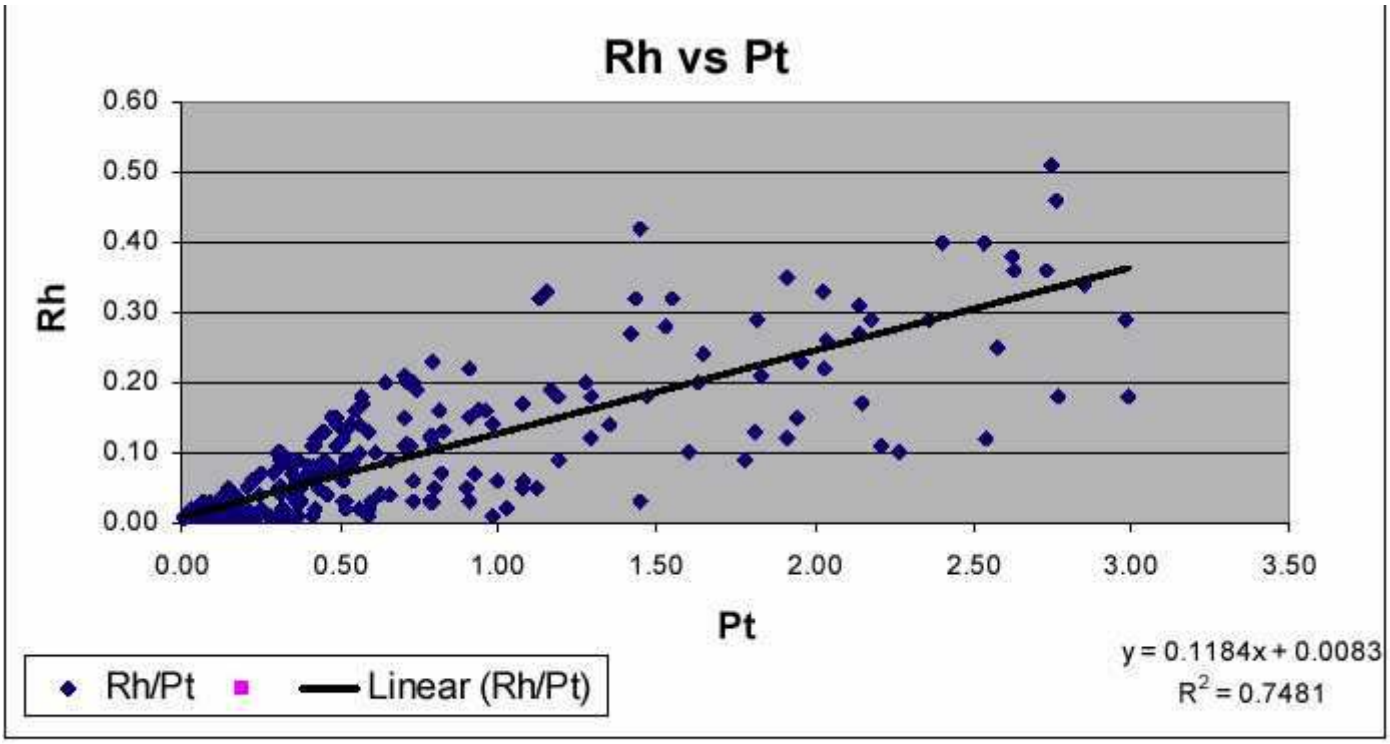


Diagram 12: Scatter plot of Rh vs Pt

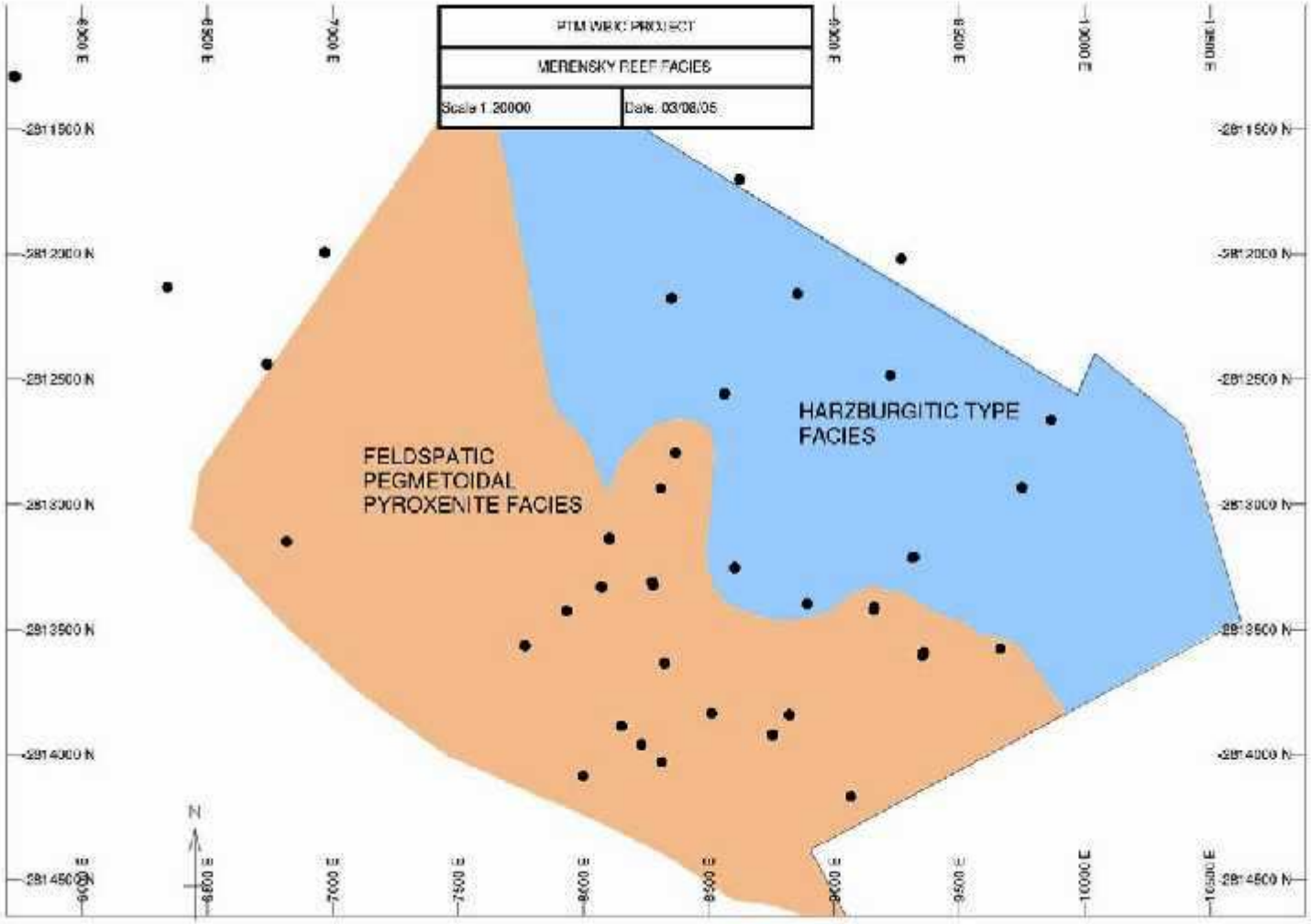


Diagram 13: Modelled Merensky Reef Facies

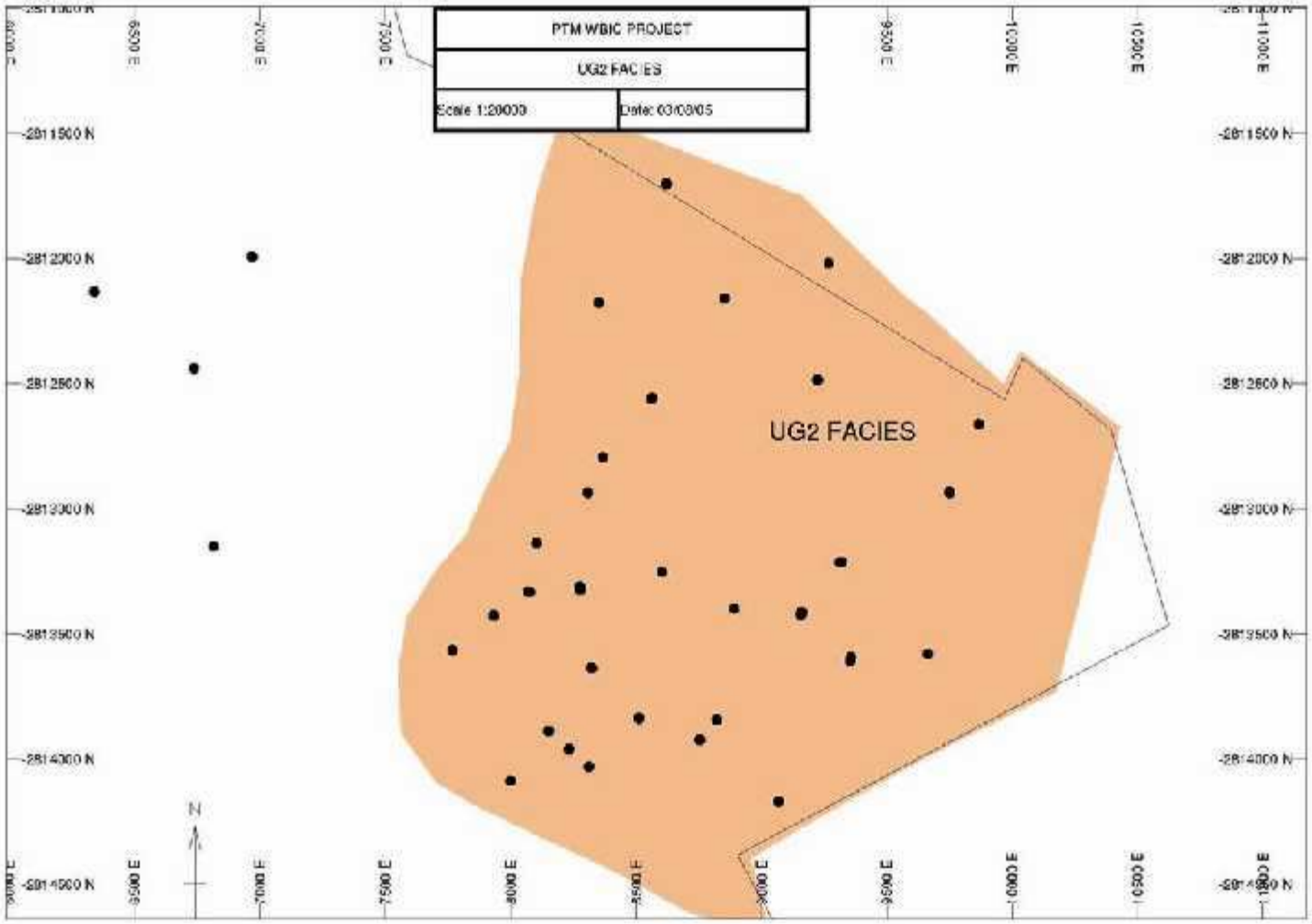


Diagram 14: UG2 Facies Plan

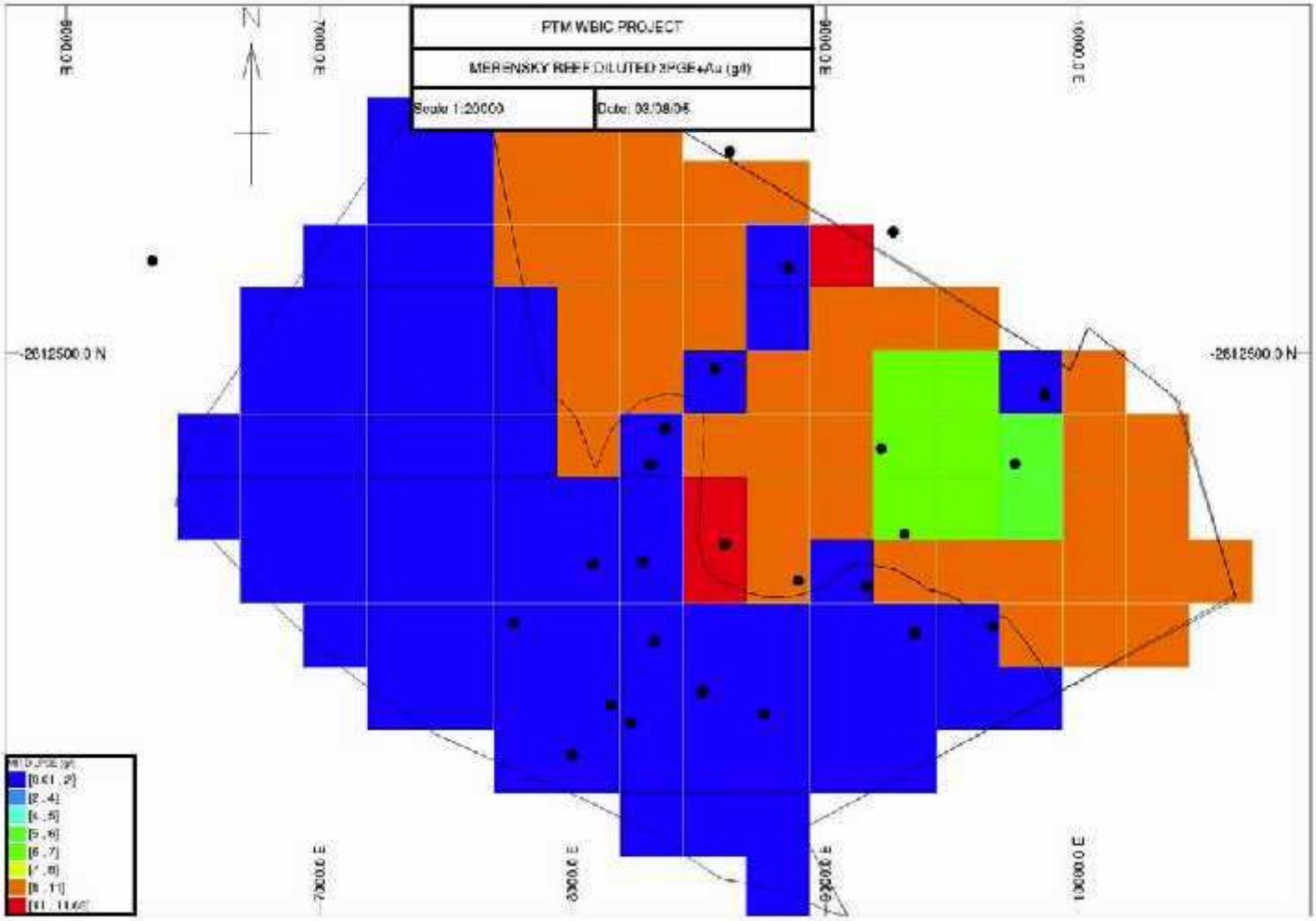


Diagram 15: Merensky Reef Diluted 3PGE+Au (g/t)

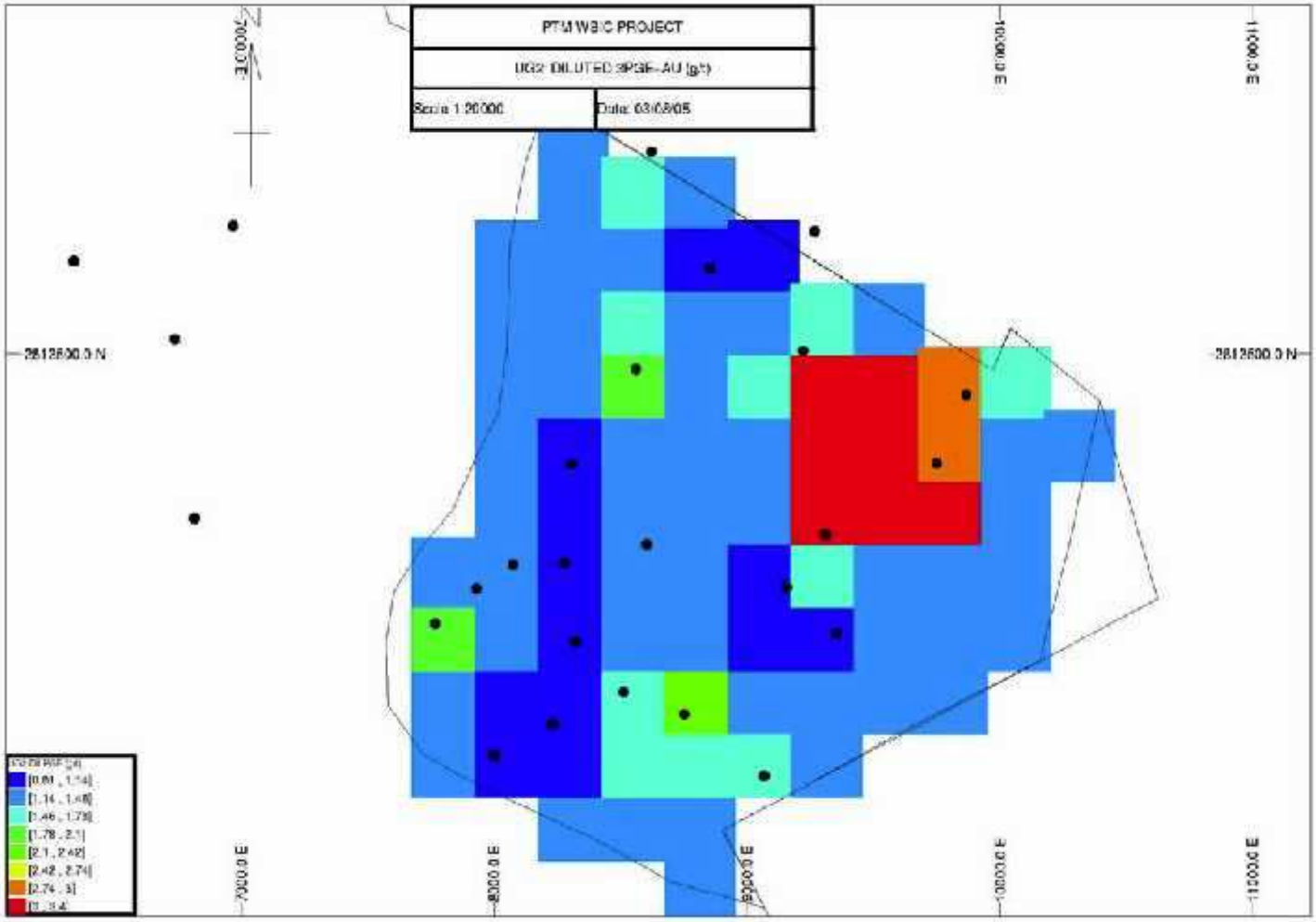


Diagram 16: UG2 Reef Diluted 3PGE+Au (g/t)

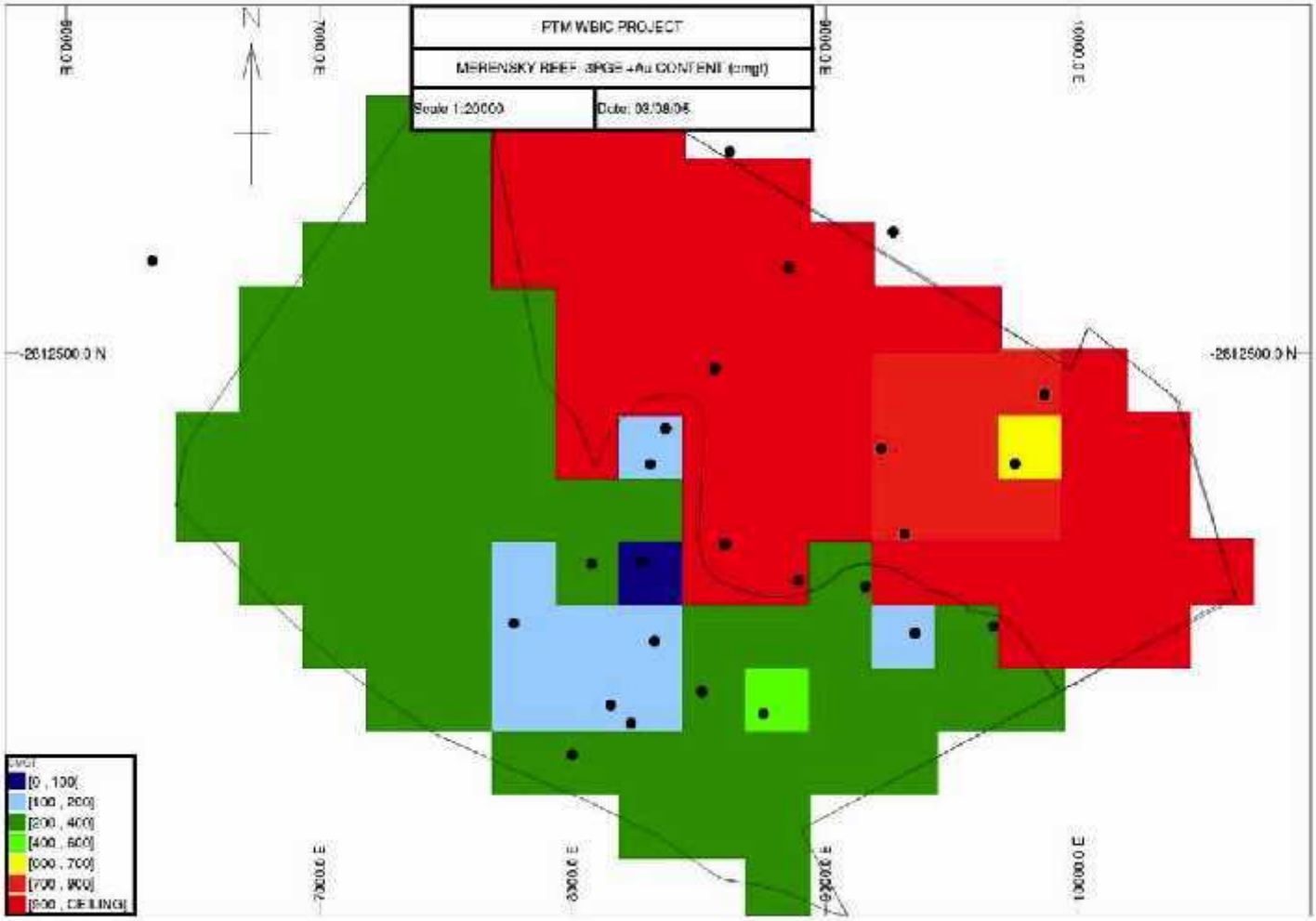


Diagram 17:Merensky Reef 3PGE+AU Content (cm gt)

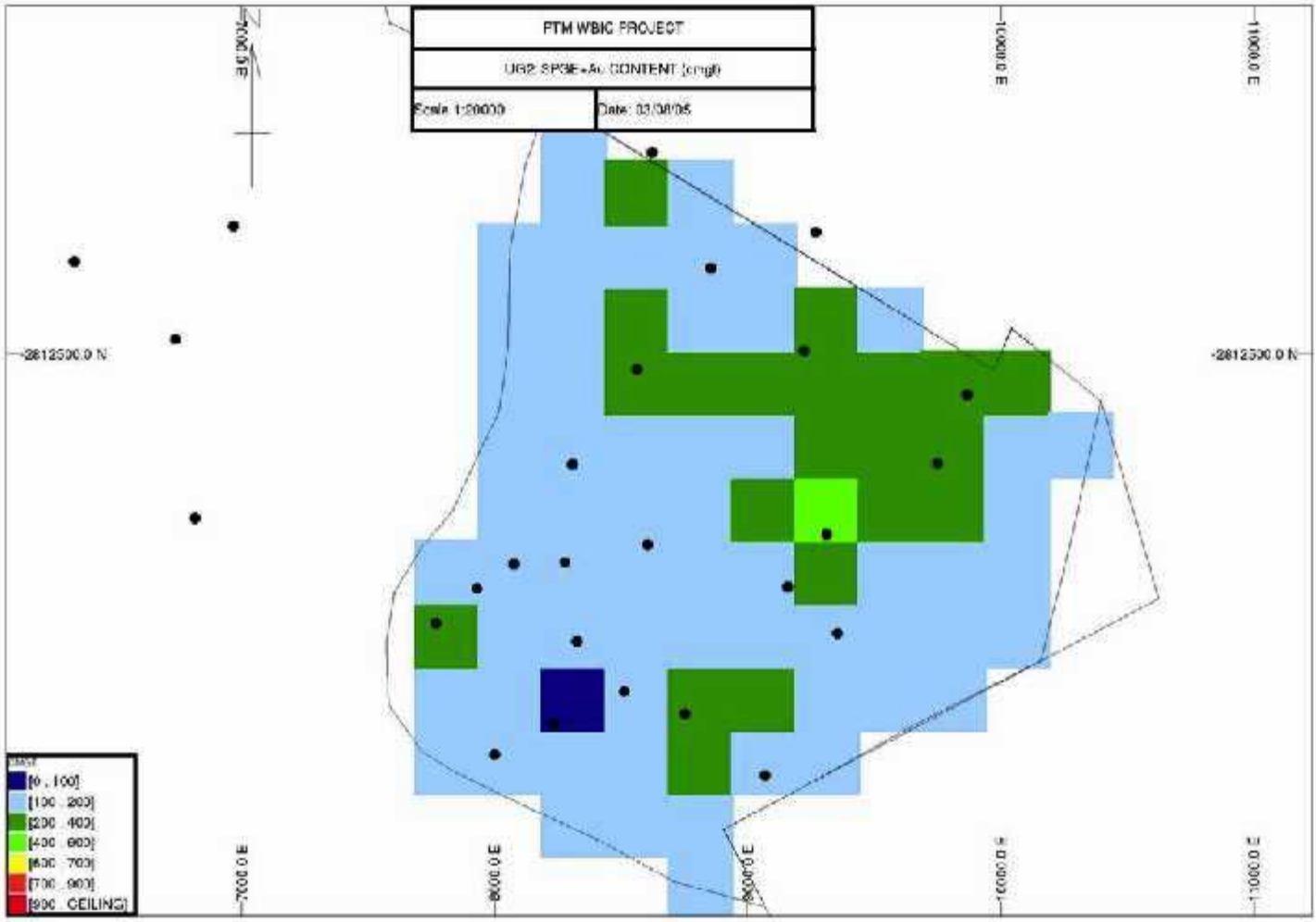


Diagram 18: UG2 Reef 3PGE+AU Content (cm gt)

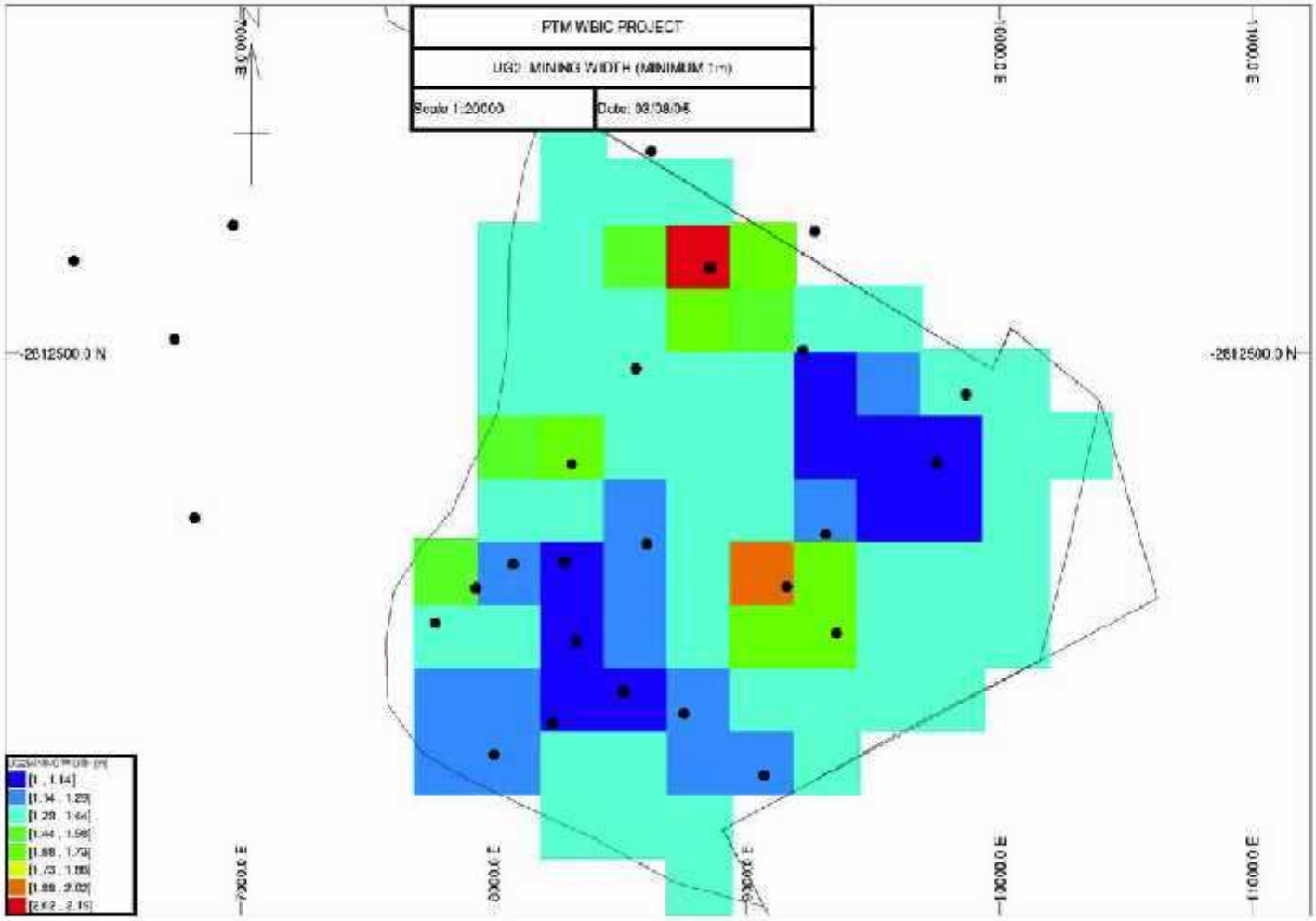


Diagram 19:UG2 Mining Width (minimum 1m)

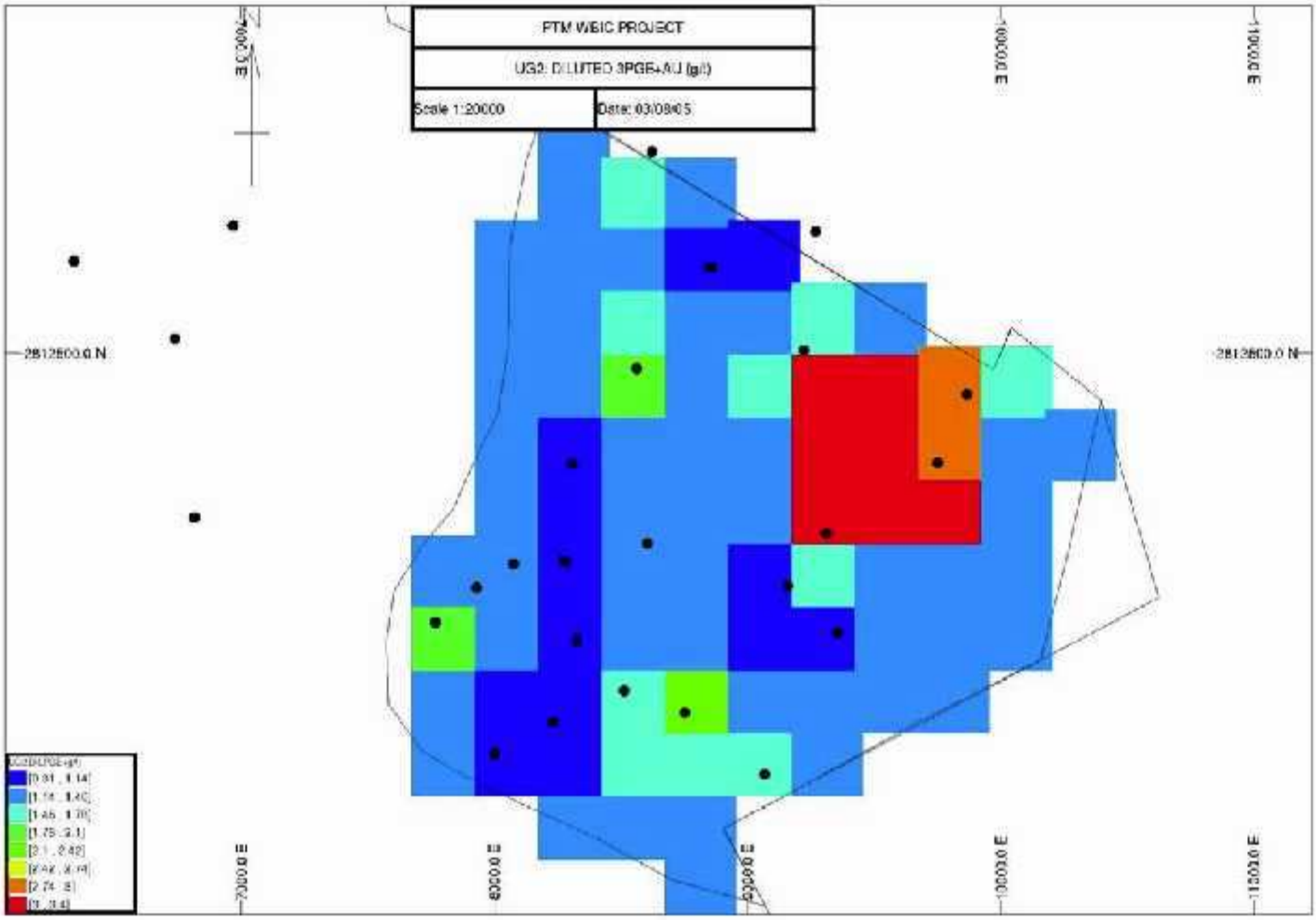


Diagram 20: UG2 Diluted 3PGE+Au (g/t)

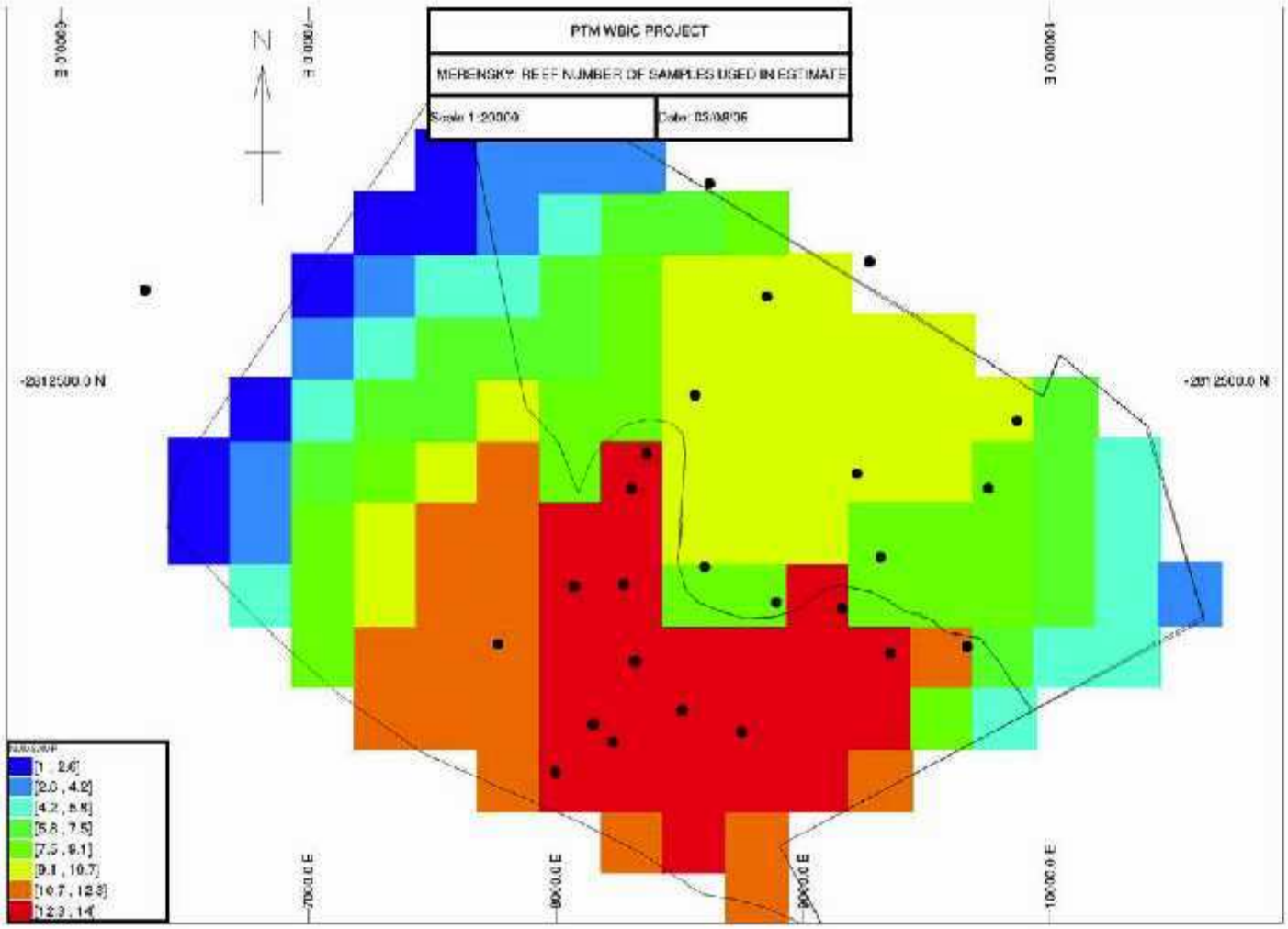


Diagram 21: Merensky Reef Number of Samples used in Estimate

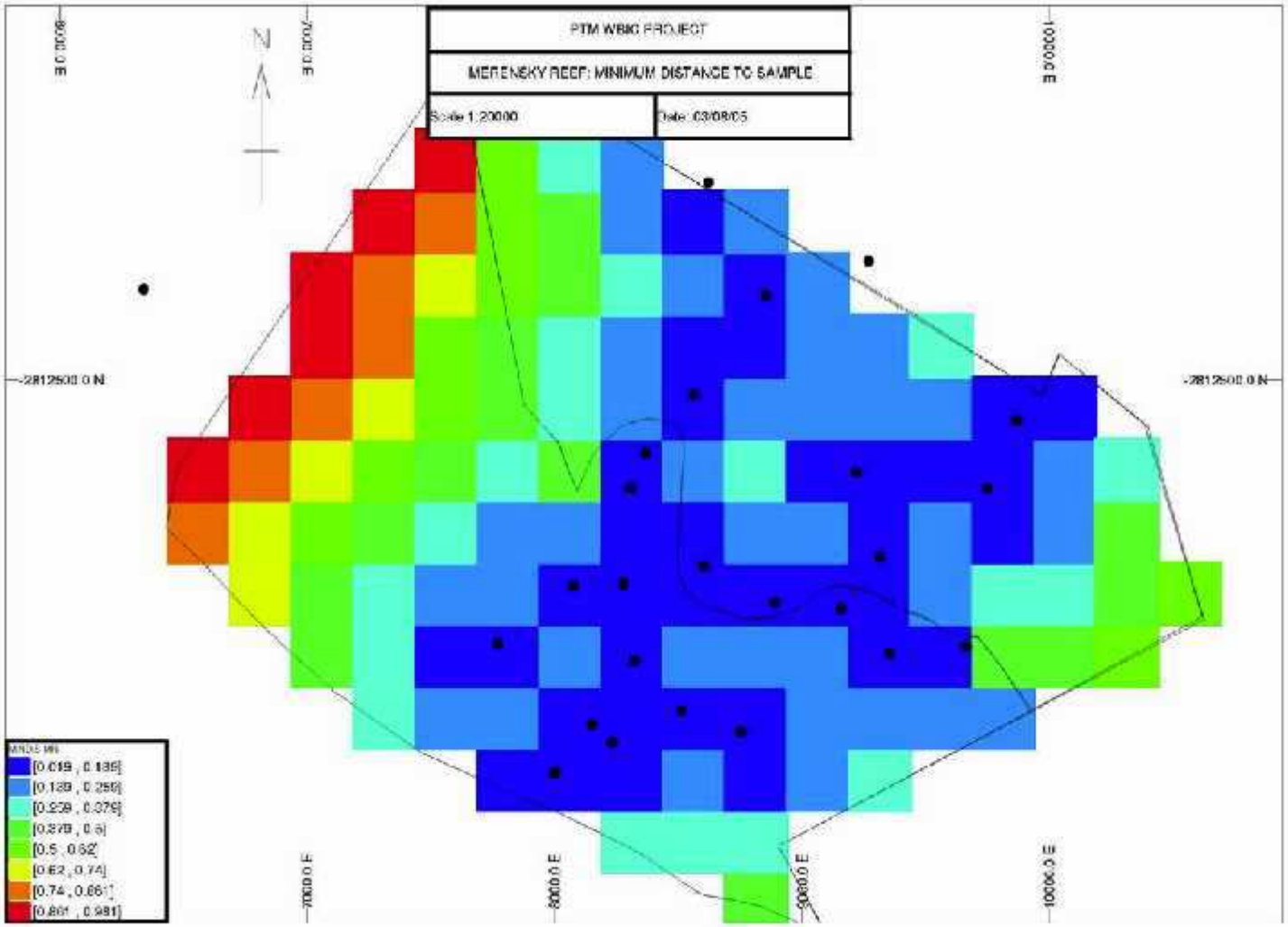


Diagram 22: Merensky Reef Minimum distance to Sample

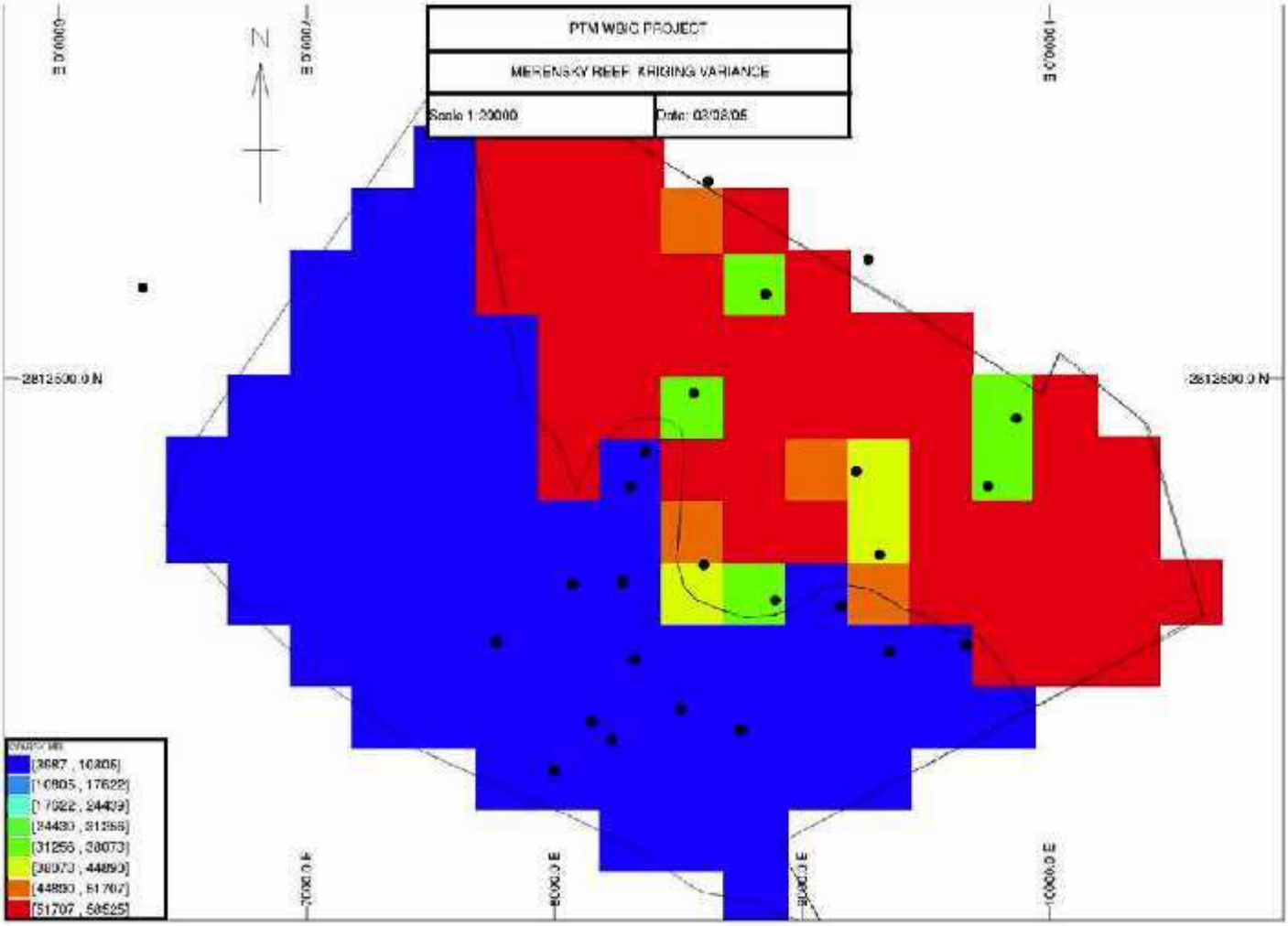


Diagram 23: Merensky Reef Kriging Variance

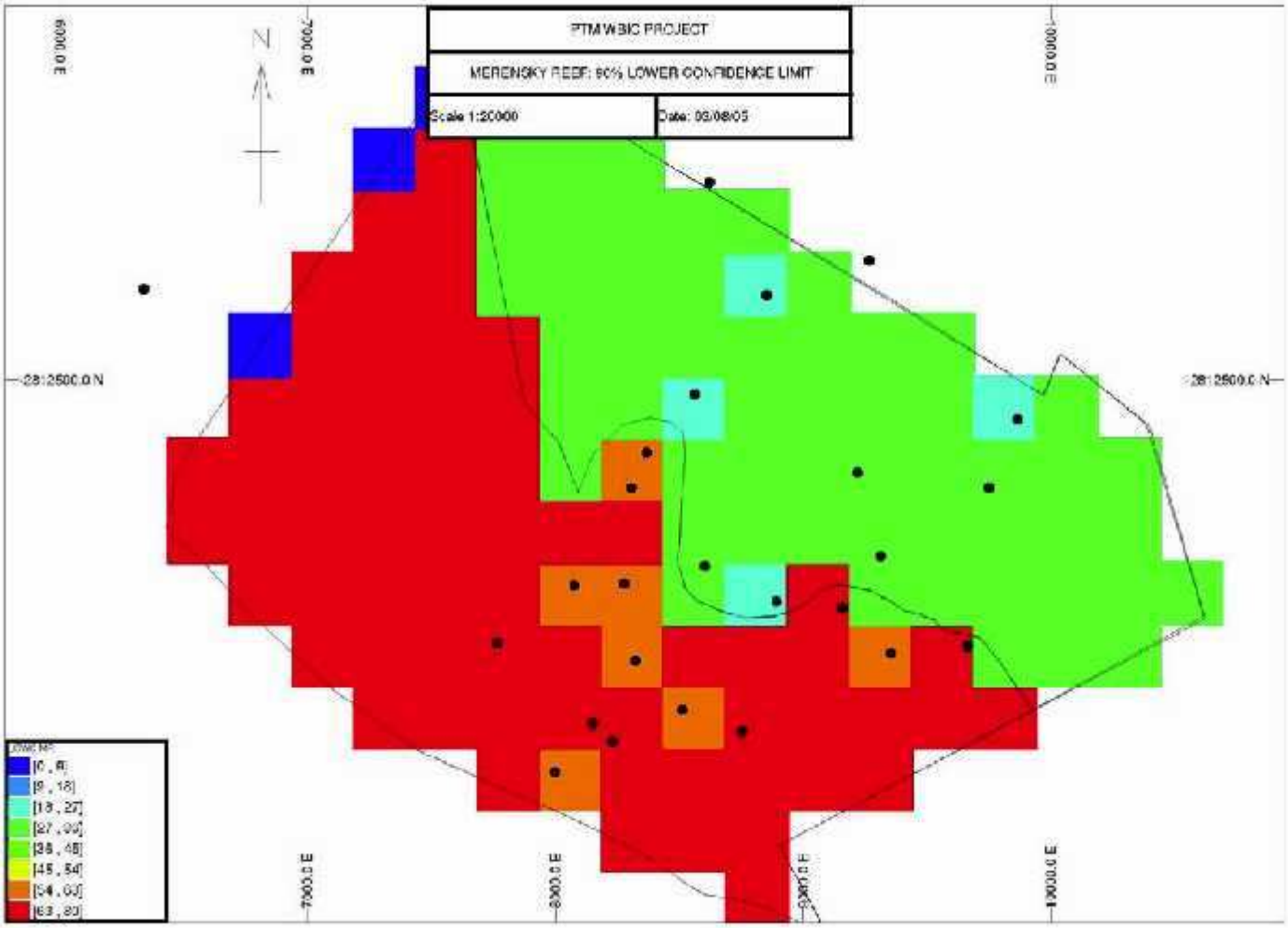


Diagram 24: Merensky Reef 90% Lower Confidence Limit

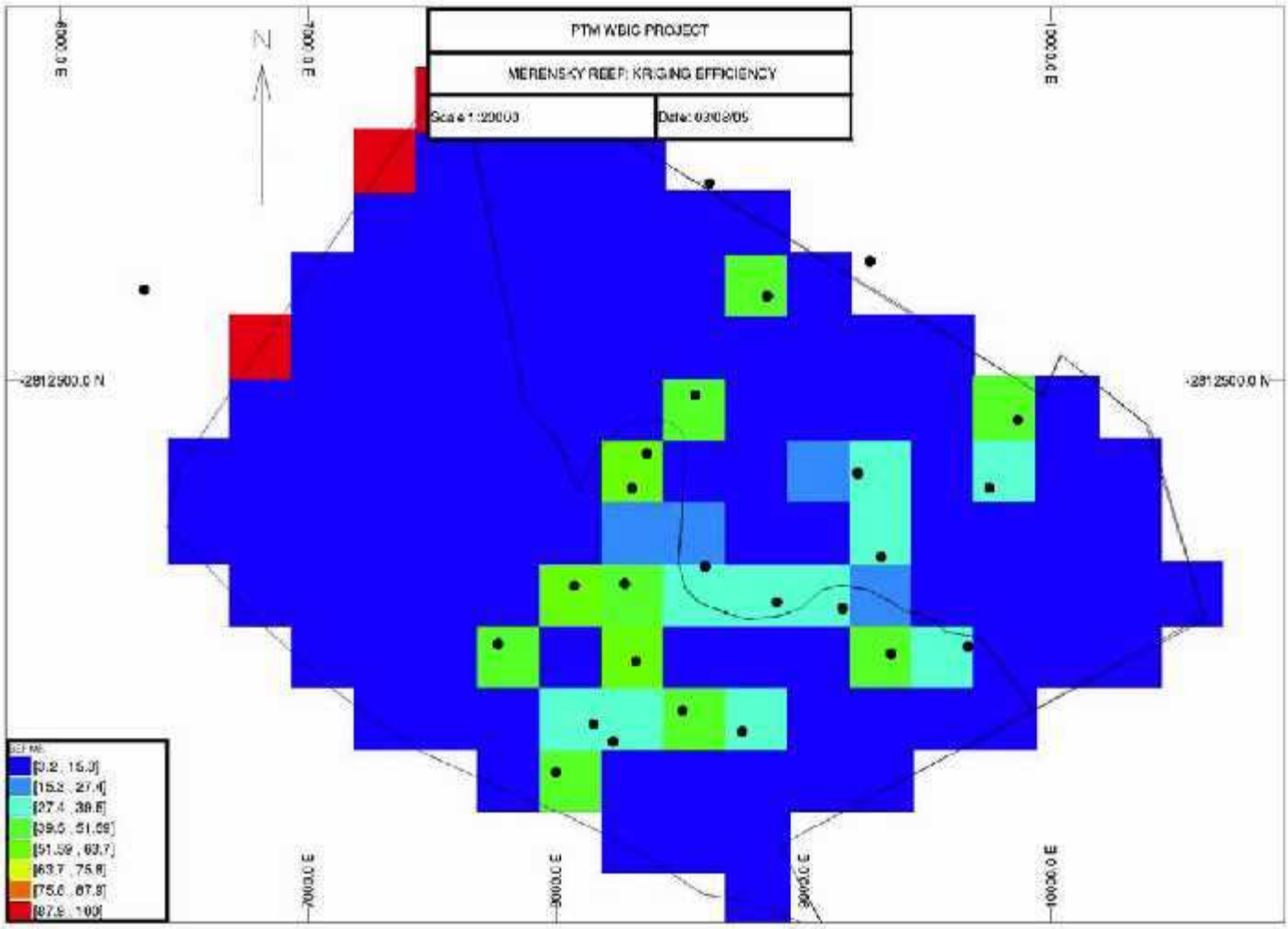


Diagram 25: Merensky Reef Kriging Efficiency

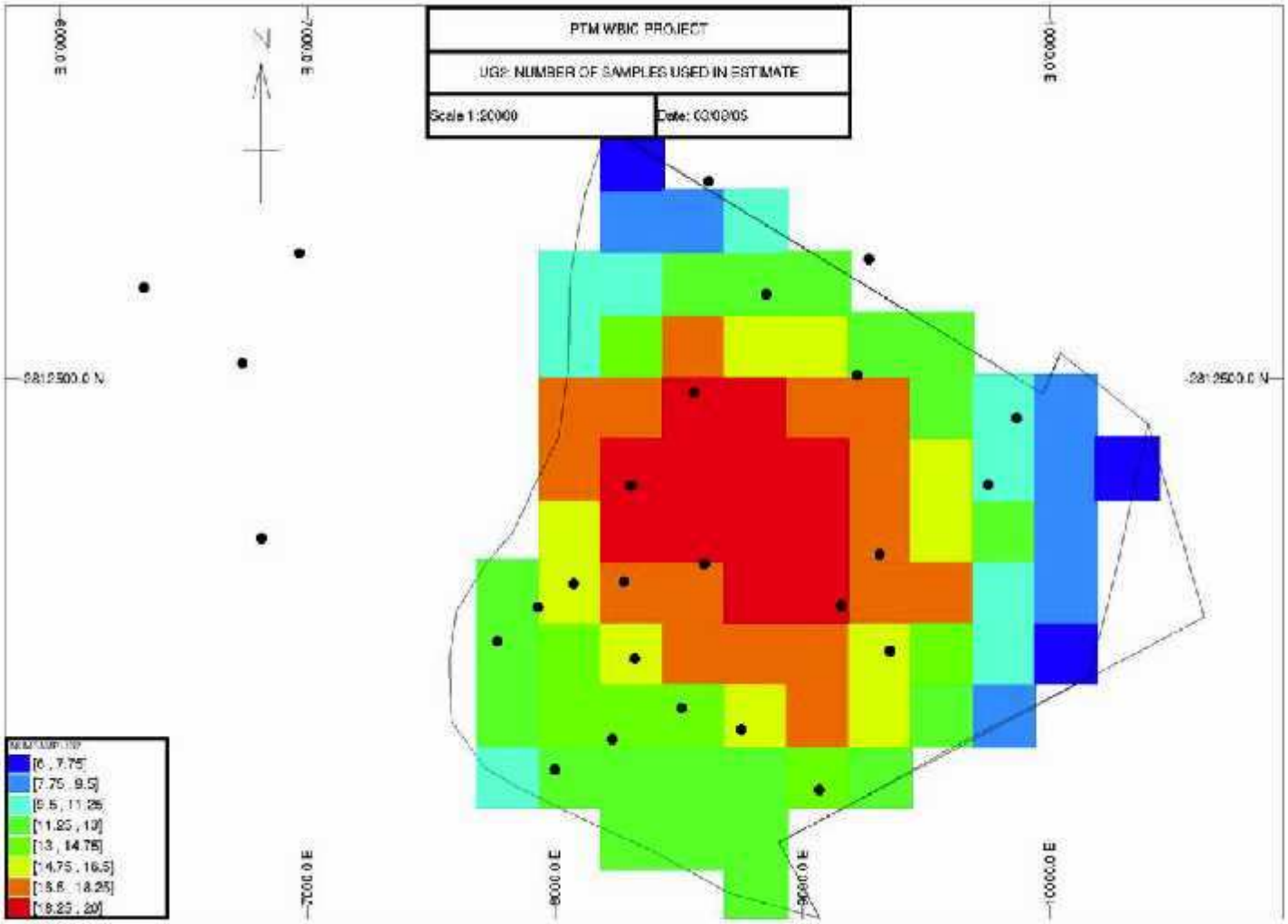


Diagram 26: UG2 Number of Samples used in Estimate

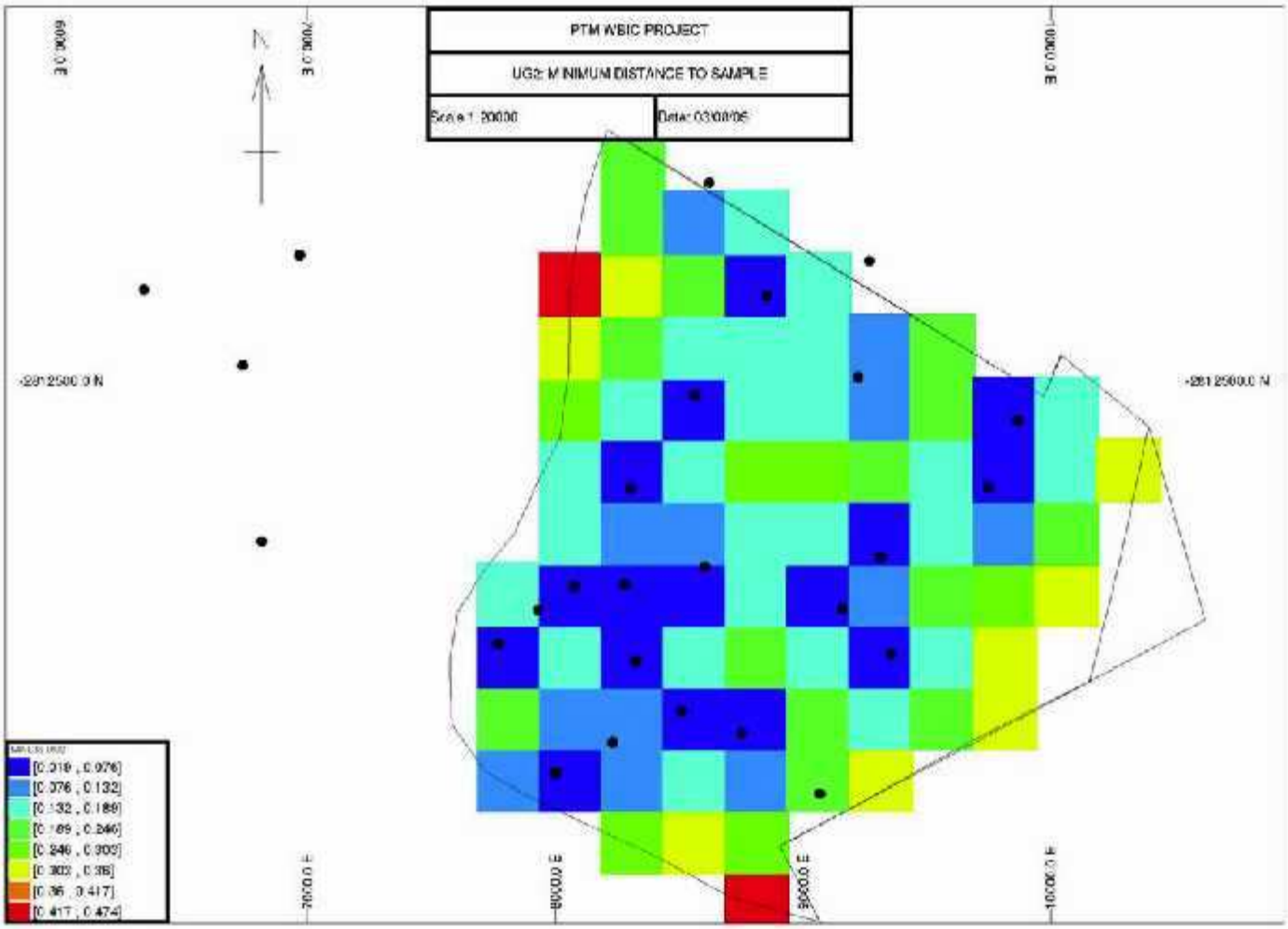


Diagram 27: UG2 Minimum Distance to Sample

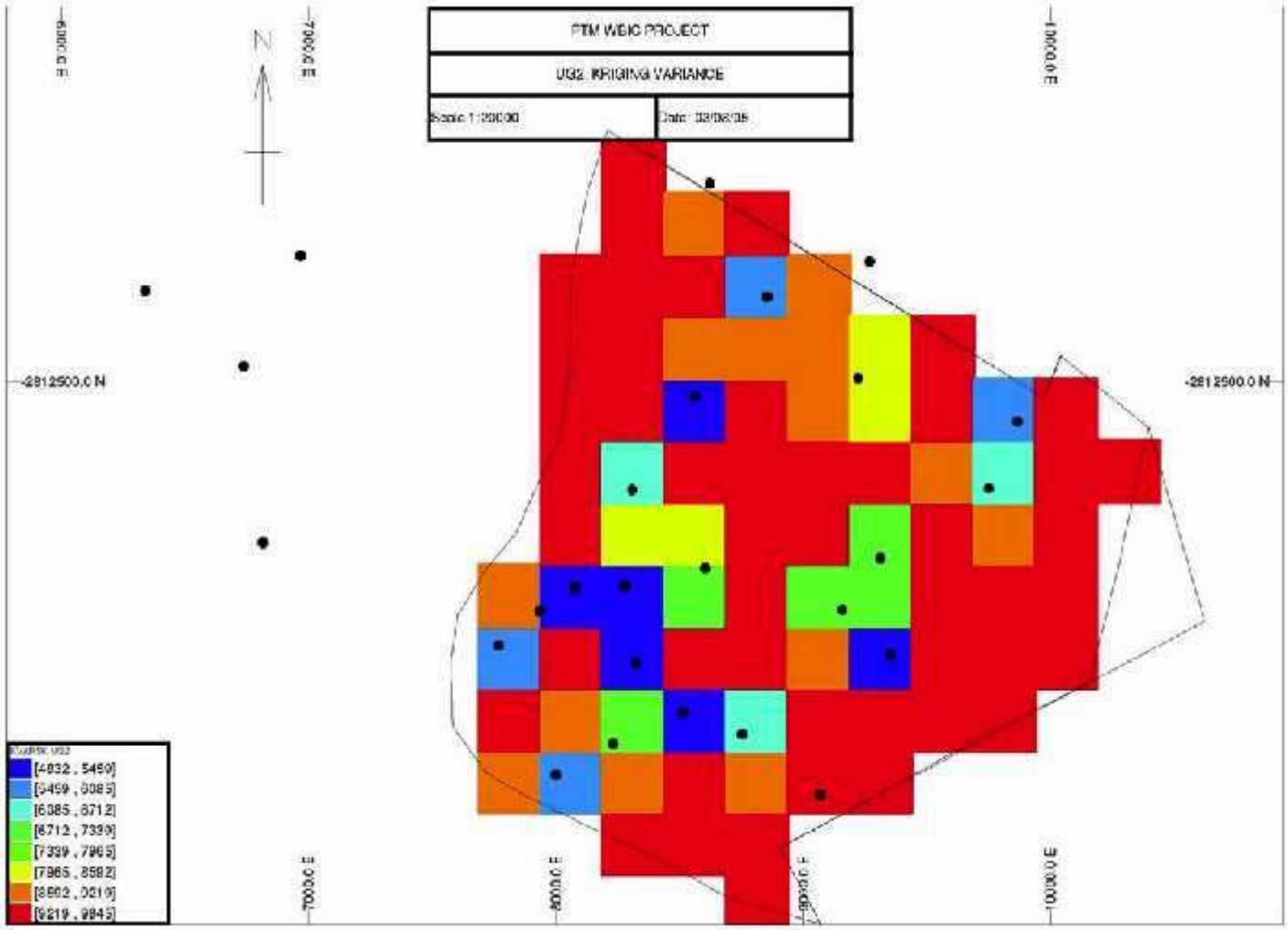


Diagram 28: UG2 Kriging Variance

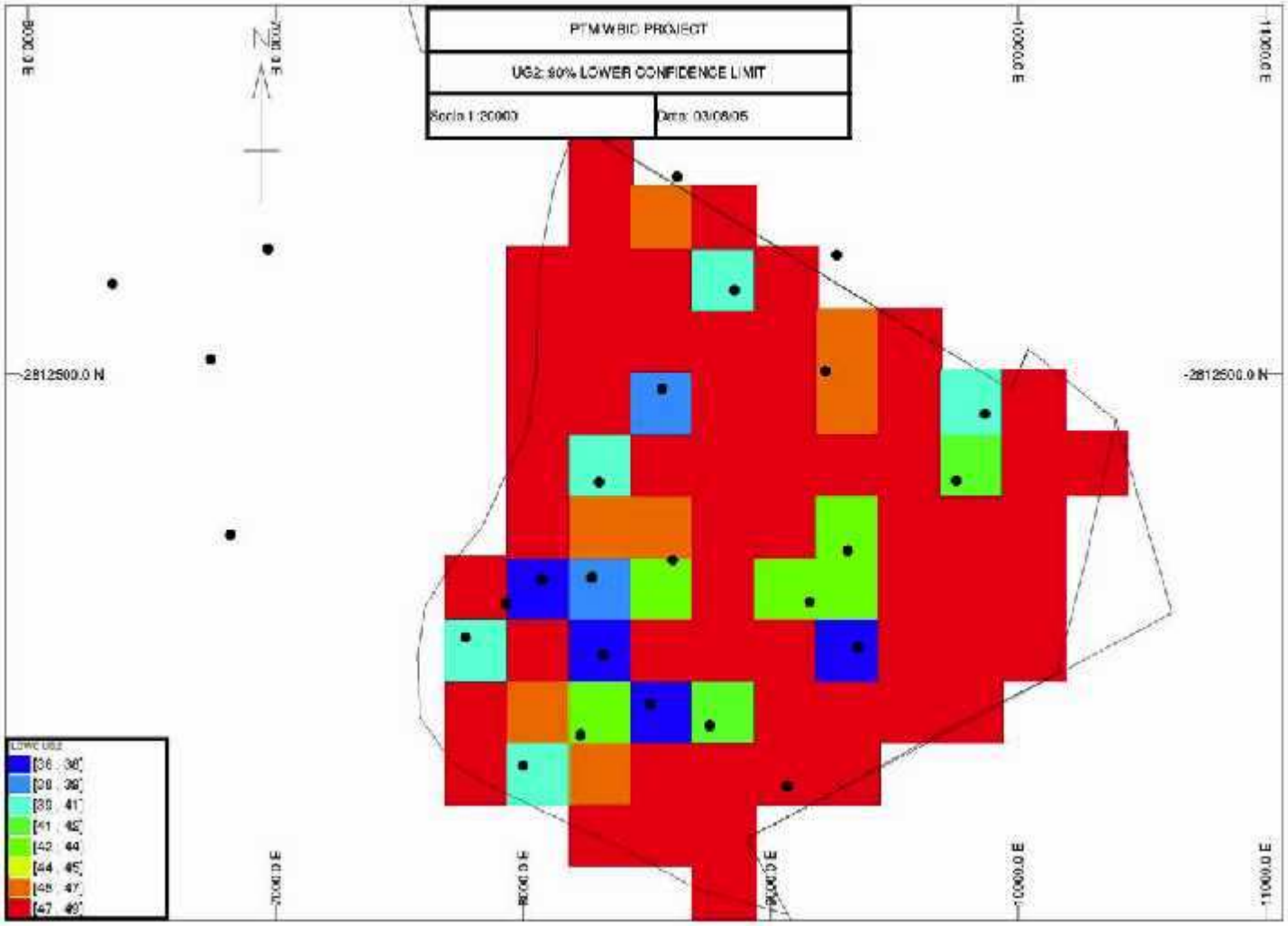


Diagram 29: UG 2 90% Lower Confidence Limit

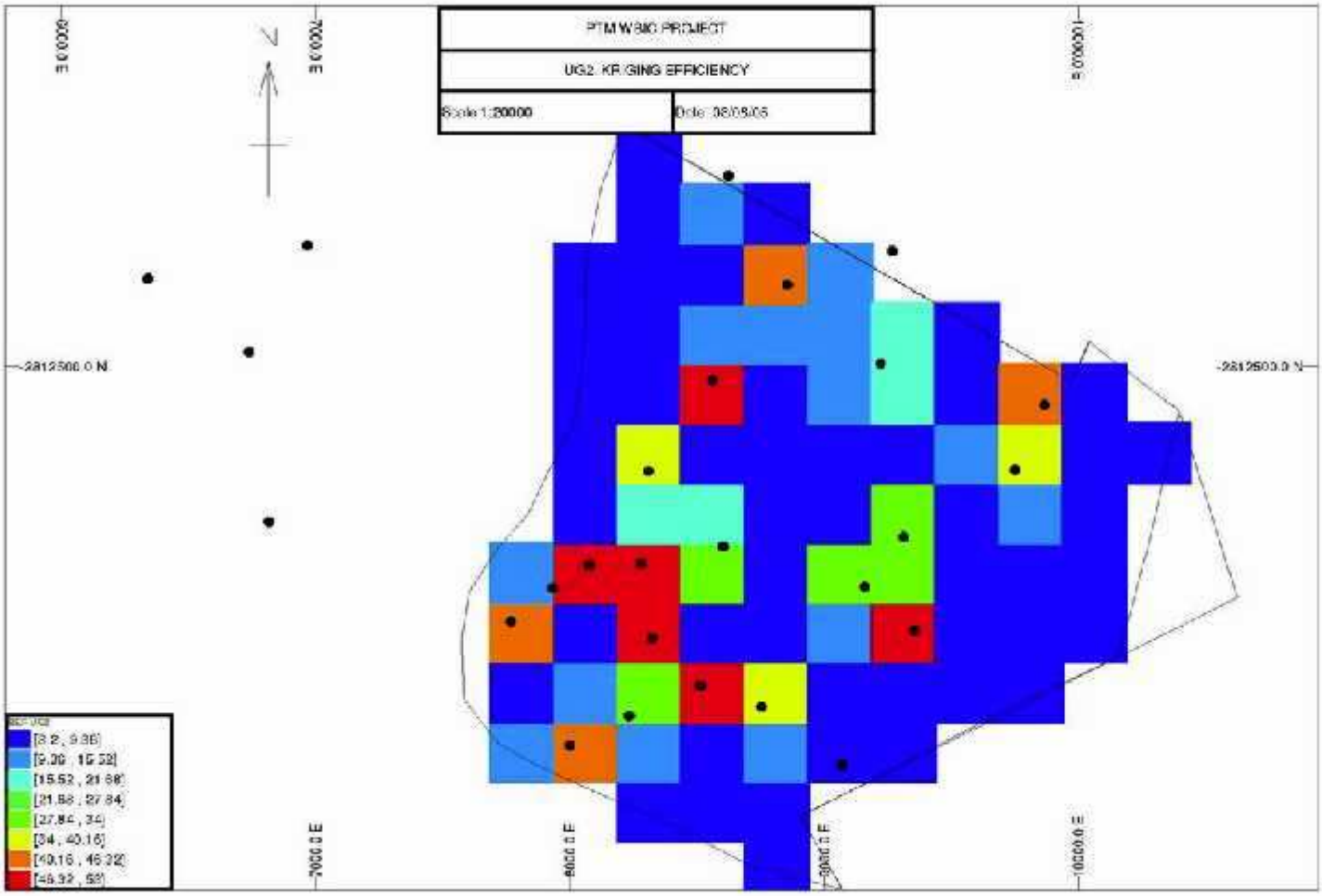


Diagram 30: UG2 Kriging Efficiency

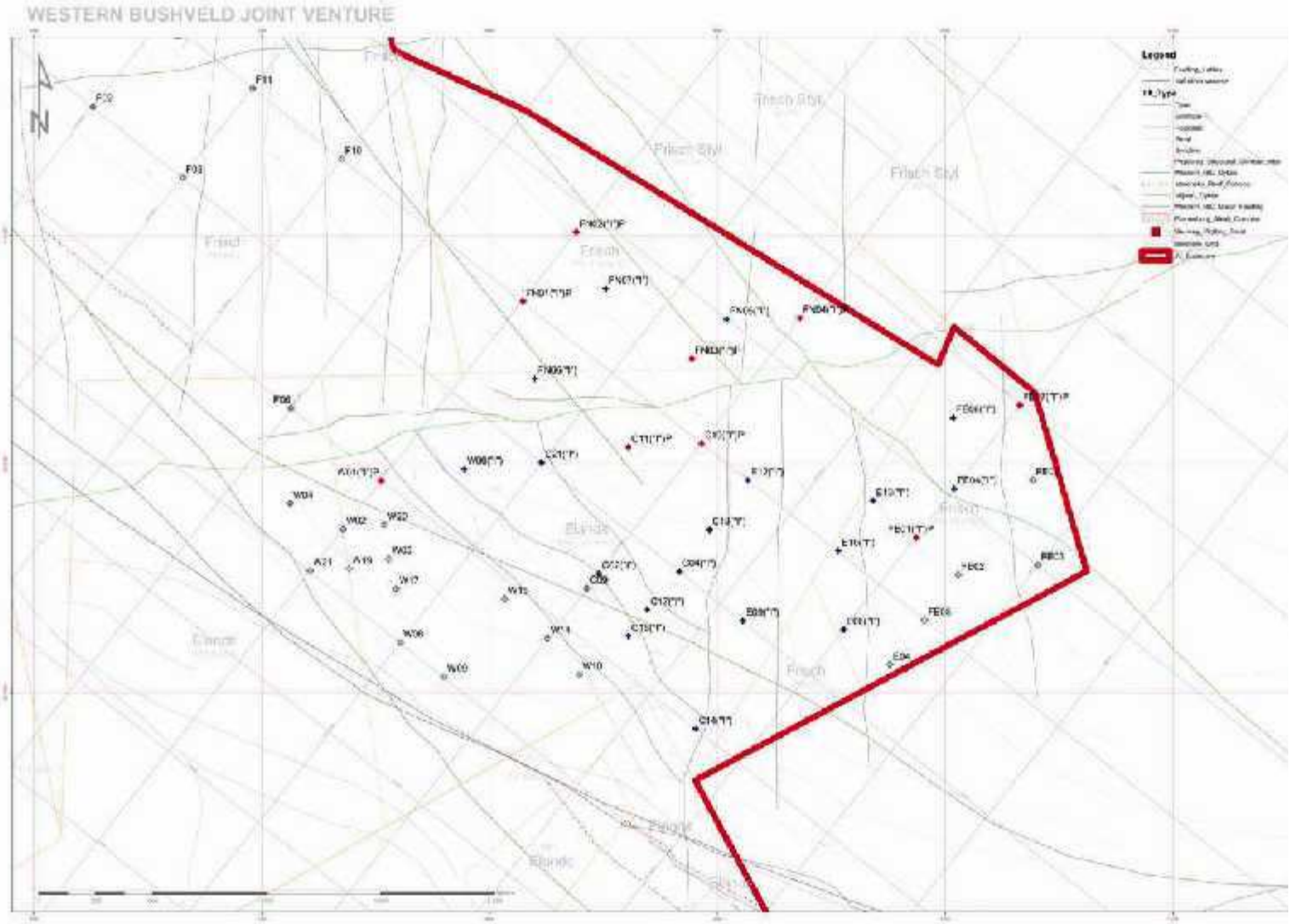


Diagram 31: Planned Drilling

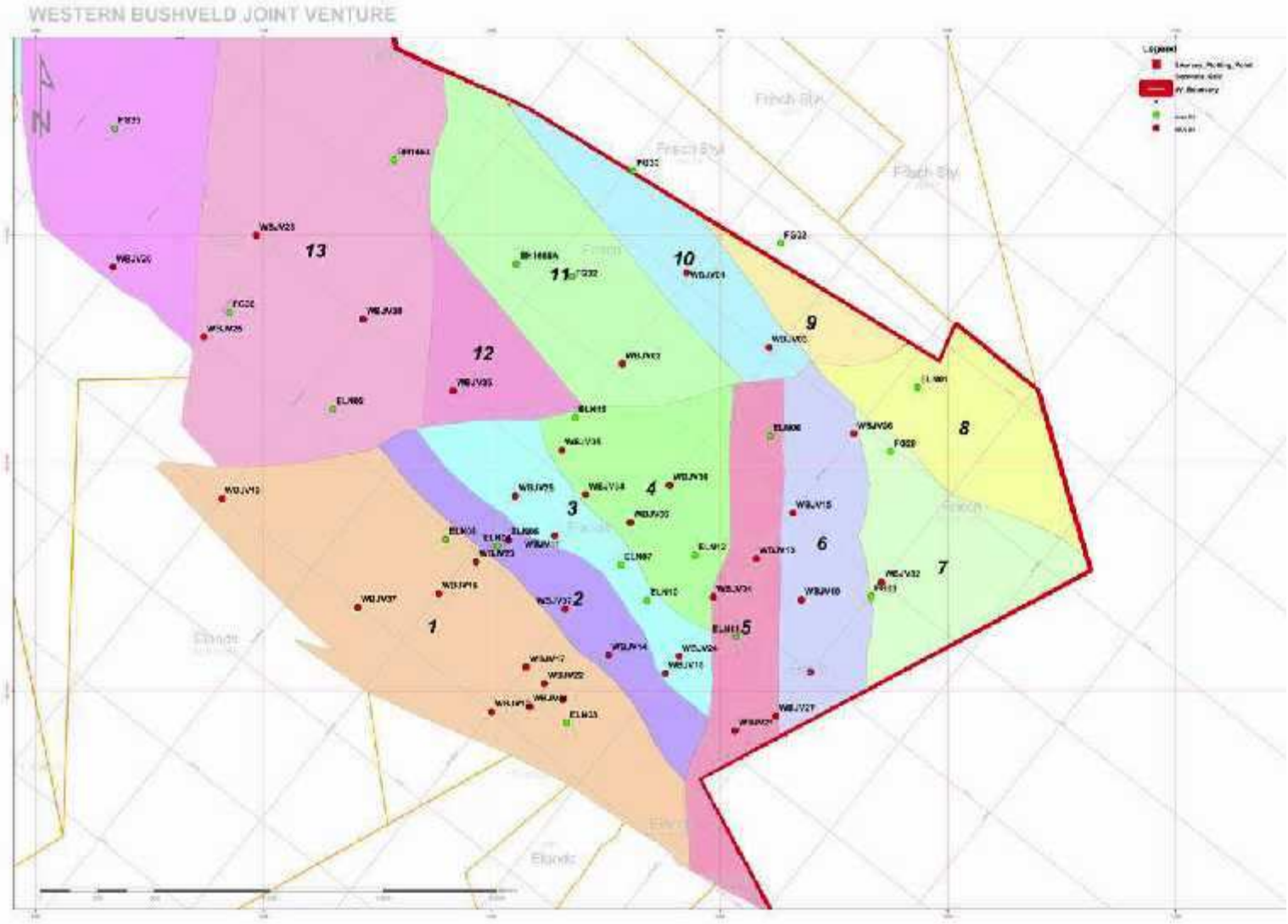


Diagram 32: Showing the numbered blocks of ground as used in Mine planning

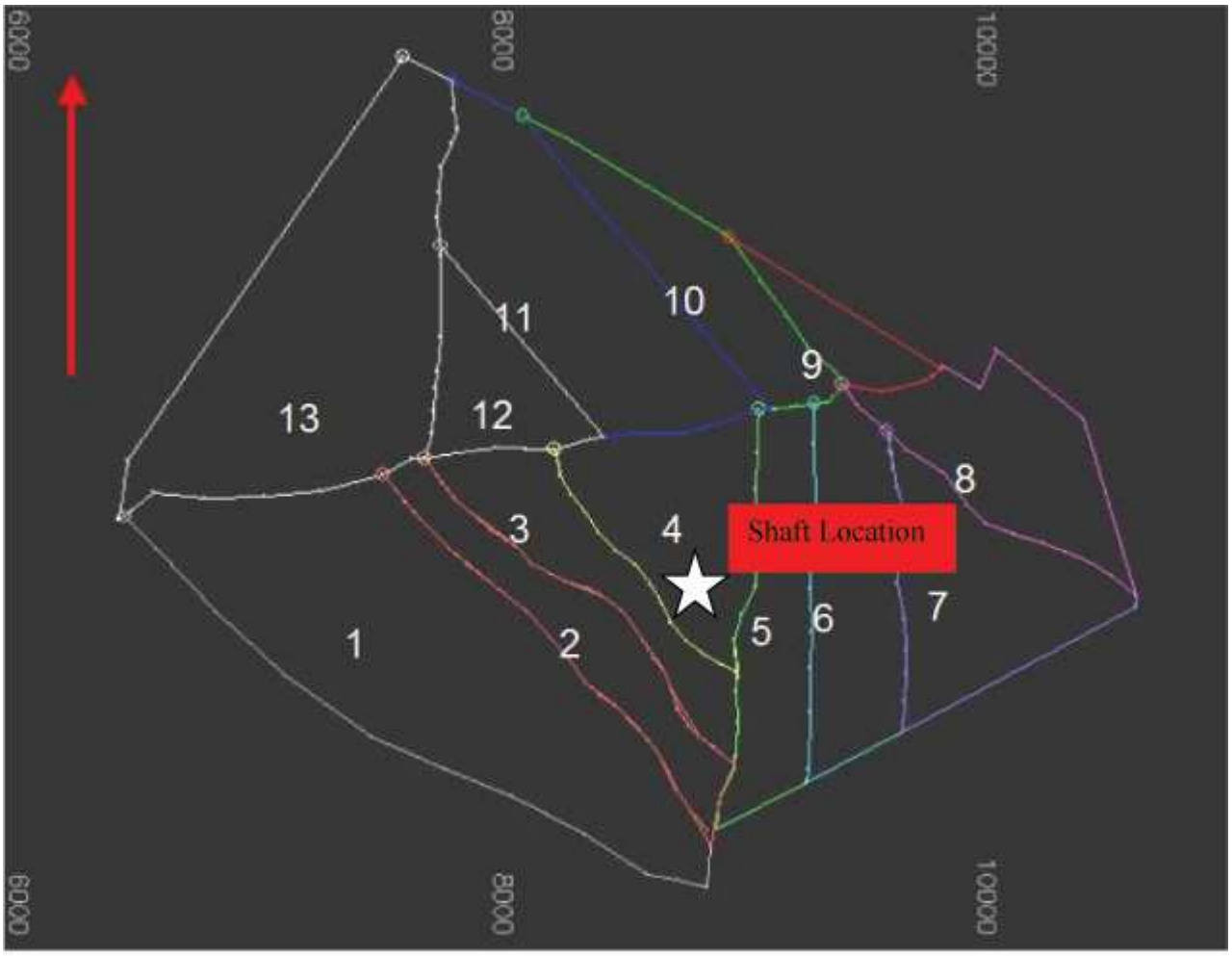


Diagram 33 – Geological Blocks at Elandsfontein for mine design

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End of Filing