SAFEGUARD SCIENTIFICS INC
Form SC 13G/A
February 11, 2016

UNITED STATES

SECURITIES AND EXCHANGE COMMISSION

Washington, D.C. 20549

SCHEDULE 13G

Under the Securities Exchange Act of 1934

(Amendment No. 8)

SAFEGUARD SCIENTIFICS INC

(Name of Issuer)

COMMON STOCK

(Title of Class of Securities)

786449207

(CUSIP Number)

December 31, 2015

(Date of Event which Requires Filing of Statement)

Check the appropriate box to designate the Rule pursuant to which this Schedule is filed:

[x] Rule 13d - 1(b)

Rule 13d - 1(c)

Rule 13d - 1(d)

1 Name of Reporting Person

T. ROWE PRICE ASSOCIATES, INC.

52-0556948

2 Check the Appropriate Box if a Member of a Group

NOT APPLICABLE

3 SEC Use Only

4 Citizenship or Place of Organization

MARYLAND

Number of Shares Beneficially Owned by Each Reporting Person With

5 Sole Voting Power*M35,096

6 Shared Voting Power* -0-

7 Sole Dispositive Power* J,052,929

8 Shared Dispositive Power -0-

9 Aggregate Amount Beneficially Owned by Each Reporting Person

2,052,929

10 Check Box if the Aggregate Amount in Row (9) Excludes Certain Shares

NOT APPLICABLE

11 Percent of Class Represented by Amount in Row 9

9.6%

12 Type of Reporting Person

IA

*Any shares reported in Items 5 and 6 are also reported in Item 7.

1 Name of Reporting Person

T. ROWE PRICE SMALL-CAP VALUE FUND, INC.

52-1575325

2 Check the Appropriate Box if a Member of a Group

NOT APPLICABLE

3 SEC Use Only

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4 Citizenship or Place of Organization

Maryland

Number of Shares Beneficially Owned by Each Reporting Person With

5 Sole Voting Power* I,510,132

6 Shared Voting Power* -0-

7 Sole Dispositive Power* -0-

8 Shared Dispositive Power -0-

9 Aggregate Amount Beneficially Owned by Each Reporting Person

1,510,132

10 Check Box if the Aggregate Amount in Row (9) Excludes Certain Shares

NOT APPLICABLE

11 Percent of Class Represented by Amount in Row 9

7.1%

12 Type of Reporting Person

IV

*The aggregate amount reported on this page is also included in the aggregate amount reported by T. Rowe Price Associates, Inc. on this Schedule 13G.

Item 1(a) Name of Issuer:

Reference is made to page 1 of this Schedule 13G

Item 1(b) Address of Issuer's Principal Executive Offices:

435 DEVON PARK DR, Building 800, WAYNE, PA 19087

Item 2(a) Name of Person(s) Filing:

(1) T. Rowe Price Associates, Inc. ("Price Associates")

(2) T. ROWE PRICE SMALL-CAP VALUE FUND, INC.

X Attached as Exhibit A is a copy of an agreement between the Persons Filing (as specified hereinabove) that this Schedule 13G is being filed on behalf of each of them.

Item 2(b) Address of Principal Business Office:

100 E. Pratt Street, Baltimore, Maryland 21202

Item 2(c) Citizenship or Place of Organization:

(1) Maryland

(2) Maryland

Item 2(d) Title of Class of Securities:

Reference is made to page 1 of this Schedule 13G

Item 2(e) CUSIP Number: 786449207

Item 3 The person filing this Schedule 13G is an:

X Investment Adviser registered under Section 203 of the Investment Advisers Act of 1940

X Investment Company registered under Section 8 of the Investment Company Act of 1940

Item 4 Reference is made to Items 5-11 on the preceding pages of this Schedule 13G.

Item 5 Ownership of Five Percent or Less of a Class.

X Not Applicable.

This statement is being filed to report the fact that, as of the date of this report, the reporting person(s) has (have) ceased to be the beneficial owner of more than five percent of the class of securities.

Item 6 Ownership of More than Five Percent on Behalf of Another Person

(1) Price Associates does not serve as custodian of the assets of any of its clients; accordingly, in each instance only the client or the client's custodian or trustee bank has the right to receive dividends paid with respect to, and proceeds from the sale of, such securities.

The ultimate power to direct the receipt of dividends paid with respect to, and the proceeds from the sale of, such securities, is vested in the individual and institutional clients which Price Associates serves as investment adviser. Any and all discretionary authority which has been delegated to Price Associates may be revoked in whole or in part at any time

Except as may be indicated if this is a joint filing with one of the registered investment companies sponsored by Price Associates which it also serves as investment adviser ("T. Rowe Price Funds"), not more than 5% of the class of such securities is owned by any one client subject to the investment advice of Price Associates.

(2) With respect to securities owned by any one of the T. Rowe Price Funds, only the custodian for each of such Funds, has the right to receive dividends paid with respect to, and proceeds from the sale of, such securities. No other person is known to have such right, except that the shareholders of each such Fund participate proportionately in any dividends and distributions so paid.

Item 7 Identification and Classification of the Subsidiary Which Acquired the Security Being Reported on By the Parent Holding Company.

Not Applicable.

Item 8 Identification and Classification of Members of the Group.

Not Applicable.

Item 9 Notice of Dissolution of Group.

Not Applicable.

Item 10 Certification.

By signing below I certify that, to the best of my knowledge and belief, the securities referred to above were acquired in the ordinary course of business and were not acquired and are not held for the purpose of or with the effect of changing or influencing the control of the issuer of the securities and were not acquired and are not held in connection with or as a participant in any transaction having that purpose or effect. T. Rowe Price Associates, Inc. hereby declares and affirms that the filing of Schedule 13G shall not be construed as an admission that Price Associates is the beneficial owner of the securities referred to, which beneficial ownership is expressly denied.

Signature.

After reasonable inquiry and to the best of my knowledge and belief, I certify that the information set forth in this statement is true, complete and correct.

T. ROWE PRICE ASSOCIATES, INC.

Date: February 16, 2016

Signature: /s/ David Oestreicher

Name & Title: David Oestreicher, Vice President

T. ROWE PRICE SMALL-CAP VALUE

FUND, INC.

Date: February 16, 2016

Signature: /s/ David Oestreicher

Name & Title: David Oestreicher, Vice President

12/31/2015

EXHIBIT A

AGREEMENT

JOINT FILING OF SCHEDULE 13G

Price Associates, Inc. (an investment adviser registered under the Investment Advisers Act of 1940), and T. ROWE PRICE SMALL-CAP VALUE FUND, INC., all of which are Maryland corporations, hereby agree to file jointly the statement on Schedule 13G to which this Agreement is attached, and any amendments thereto which may be deemed necessary, pursuant to Regulation 13D-G under the Securities Exchange Act of 1934.

It is understood and agreed that each of the parties hereto is responsible for the timely filing of such statement and any amendments thereto, and for the completeness and accuracy of the information concerning such party contained therein, but such party is not responsible for the completeness or accuracy of information concerning the other party unless such party knows or has reason to believe that such information is inaccurate.

It is understood and agreed that a copy of this Agreement shall be attached as an exhibit to the statement on Schedule 13G, and any amendments hereto, filed on behalf of each of the parties hereto.

T. ROWE PRICE ASSOCIATES, INC.

Date: February 16, 2016

Signature: /s/ David Oestreicher

Name & Title: David Oestreicher, Vice President

T. ROWE PRICE SMALL-CAP VALUE FUND, INC.

Date: February 16, 2016

Signature: /s/ David Oestreicher

Name & Title: David Oestreicher, Vice President

width="5%">JFMAMJJASONDTEMP (0^{o}C)24.123.222.018.415.011.712.014.818.821.322.623.7SUNSHINE (hrs)259237246218268261290306298276250274RAINFALL (mm)1178374571453513376467

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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.

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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

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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.

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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 6ºnorth-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.

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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.

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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 6ºto the north-east.

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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.

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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.

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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.

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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

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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.

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.	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 5: 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

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.

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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

									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

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.

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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.

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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

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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%

Inferred: <20%

c.

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.

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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 (cm g/t)	**Tonnes (Mt)**	**Grade 3PGM+Au (g/t)**	**Channel Width (metres)**	**Diluted Channel Width (metres)**	**Tonnes 3PGM+Au (t)**	**Ounce (Millions)**
MR Domain1	200	13.87	9.67	1.11	1.12	134.11	4.31
MR Domain 2	400	0.53	6.47	0.42	1.00	3.46	0.11

UG2 Domain 1	400	2.21	4.32	1.35	1.35	9.57	0.31
TOTAL						**147.13**	**4.73**

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.

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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.

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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

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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.

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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	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**

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.

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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.

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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.

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.

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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.

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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 2.3m3 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.

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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.

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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.

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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 m3/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.

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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.

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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	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 - 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

- 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.

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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 0.5m3/tonne milled for the concentrator and 0.4 m3/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.

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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 m3. A footprint of about 42 hectares would be required for a tailings dam of 30 m in height.

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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.

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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

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- 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.

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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.

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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.

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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	**R 31,000,000**
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 71,700,000**
	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			
	Cu	$3,500.00	6.55	$150.00	R 21,943	Prices 01-Jul-05 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.

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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

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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

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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

- 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

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- 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.

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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

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ADDENDUM (a) - CERTIFICATES

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ADDENDUM (b) - SUPPLEMENTS

(P t y) L t d

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.
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.

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Director: G.I.Cunningham

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 Coordinat UTM**
1	ELN01	D0		MR	Pass	Htz	491.59	1.00						8.08	808	9866.310
2	ELN01	D3		MR	Pass	Htz	491.58	1.00						12.82	1282	9866.310
3	ELN06	D2	New	MR	Pass	Htz	400.57	1.45	1314	3873	1783	272	7.24	7.24	1047	9219.720
4	ELN12	D0	New	MR	Pass	Htz	334.81	1.00	2469	1979	188	54	4.69	4.69	469	8892.174
5	ELN12	D1		MR	Pass	Htz	334.71	2.03						7.93	1611	8892.174
6	ELN12	D2		MR	Pass	Htz	334.61	2.14						11.51	2464	8892.174
7	ELN15	D0	New	MR	Pass	Htz	432.80	1.00	363	172	34	8	0.58	0.58	58	8367.607
8	FG02	D0		MR	Pass	Htz	521.79	2.46						13.57	3338	9268.620
9	FG02	D2		MR	Pass	Htz	522.00	2.50						18.69	4673	9268.620
10	FG29	D0		MR	Pass	CR	468.10	1.13	1585	841	82	182	2.69	2.69	304	9749.469
11	FG29	D1		MR	Pass	CR	469.40	1.77	1219	666	43	172	2.10	2.10	372	9749.469
12	FG30	D0		MR	Pass	CR	506.00	1.02	2781	1586	167	272	4.81	4.81	491	8622.793
13	FG30	D3		MR	Pass	CR	505.60	1.00	4871	2506	257	448	8.08	8.08	808	8622.793
14	FG31	D0		MR	Pass	FPP	336.50	1.25	4691	2148	224	552	7.62	7.61	951	7781.225
15	FG33	D0		MR	Pass	Pxnt	395.16	1.32	2234	912	88	312	3.55	3.55	469	9664.449
16	FG33	D6		MR	Pass	Pxnt	395.60	1.00	2972	1328	174	289	4.76	4.76	476	9664.449
17	FG34	D7		MR	Pass	FPP	890.20	1.23	4111	1789	299	325	6.52	6.52	848	6890.892
18	FG34	D9		MR	Pass	FPP	889.70	1.07	9555	2881	971	97	13.50	13.50	1464	6890.892
19	WBJV01	D0	New	MR	Pass	FPP	448.65	1.00	2854	1307	-	171	4.33	4.33	433	8855.941
20	WBJV01	D2	New	MR	Pass	FPP	442.14	1.00	3058	1392	-	276	4.73	4.73	473	8855.941
21	WBJV02	D0	New	MR	Pass	FPP	465.74	1.00	4054	1948	-	406	6.41	6.41	641	8573.092
22	WBJV02	D1	New	MR	Pass	FPP	459.84	1.00	2029	1252	-	180	3.46	3.46	346	8573.092
23	WBJV06	D0	New	MR	Pass	Htz	460.98	1.00	10051	4525	-	448	15.02	15.02	1502	8608.128
24	WBJV06	D1	New	MR	Pass	Htz	457.37	1.00	8483	4252	-	555	13.29	13.29	1329	8608.128
25	WBJV08	D0	New	MR	Pass	Pxnt	243.98	1.00	1472	691	-	127	2.29	2.29	229	8072.823
26	WBJV08	D1	New	MR	Pass	Pxnt	240.67	1.00	2310	1299	32	547	4.19	4.19	419	8072.823
27	WBJV09	D1	New	MR	Pass	Htz	265.05	1.00	963	502	9	27	1.50	1.50	150	5733.700
28	WBJV10	D1	New	MR	Pass	FPP	422.81	1.00	1312	523	74	143	2.05	2.05	205	9358.783
29	WBJV12	D0	New	MR	Pass	Pxnt	65.22	1.00	340	143	0	2	0.49	0.49	49	7999.966
30	WBJV14	D0	New	MR	Pass	Pxnt	235.65	1.00	926	328	22	73	1.35	1.35	135	8511.950
31	WBJV14	D1	New	MR	Pass	Pxnt	238.59	1.00	239	100	30	3	0.37	0.37	37	8511.950
32	WBJV15	D0	New	MR	Pass	FPP	390.73	1.00	6612	2403	300	368	9.68	9.68	968	9320.907
33	WBJV15	D1	New	MR	Pass	FPP	392.05	1.00	2943	1238	142	195	4.52	4.52	452	9320.907
34	WBJV16	D0	New	MR	Pass	Pxnt	118.06	1.17	605	330	22	108	1.07	1.07	125	7768.690
35	WBJV16	D1	New	MR	Pass	Pxnt	117.71	1.00	309	153	11	92	0.57	0.57	57	7768.690
36	WBJV17	D0	New	MR	Pass	CR	78.15	1.00	26	11	1	3	0.04	0.04	4	8151.766
37	WBJV17	D1	New	MR	Pass	CR	77.65	1.00	29	10	1	1	0.04	0.04	4	8151.766
38	WBJV18	D0	New	MR	Pass	Pxnt	231.49	1.11	378	237	18	132	0.77	0.77	85	8761.791
39	WBJV18	D1	New	MR	Pass	Pxnt	232.12	1.00	6799	2996	242	665	10.70	10.70	1070	8761.791

Coordinates in WGS84, Hartebeeshoek datum and UTM 35S

Entry:	Explanation	Facies:	Explanation
-	**Designates No Value**	Htz	Harzburgitic Facies
Pass	**QAQC**	Pxnt	Pyroxenite Facies
SNV	**Sampled but no value Return-Lost**	FPP	Pegmatoidal Feldspathic Pyroxenite Facies
Insufficient Sample	**Not enough material to accuratley assay**	CR	Contact Facies
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 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 Coo U**
1	ELN01	D0		UG2	Pass	543.48	1.83						2.81	514	9866.310	-28126
2	ELN01	D1		UG2	Pass	543.47	1.80	–	–	–	–		2.43	437	9866.310	-2812
3	ELN01	D2		UG2	Pass	542.95	1.68	–	–	–	–		2.71	455	9866.310	-2812
4	ELN05	D1		UG2	Pass	532.61	1.80	2106	691	298	12		3.11	560	8070.813	-2813
5	FG02	D0		UG2	Pass	593.06	1.70						0.20	34	9268.620	-2812
6	FG02	D1		UG2	Pass	592.94	1.80	–	–	–	–		0.79	142	9268.620	-2812
7	FG29	D0		UG2	Pass	543.84	0.64	1412	410	143	20	1.99	1.99	127	9749.469	-2812
8	FG30	D0		UG2	Pass	559.67	0.89	2284	891	354	23	3.55	3.55	316	8622.793	-2811
9	FG30	D1		UG2	Pass	559.65	0.96	2372	999	373	25	3.77	3.77	362	8622.793	-2811
10	FG30	D2		UG2	Pass	560.05	1.10	1963	496	197	12		2.67	294	8622.793	-2811
11	WBJV01	D0	New	UG2	Pass	474.70	2.20	541	187	–	0	0.73	0.73	161	8855.941	-2812
12	WBJV01	D1	New	UG2	Pass	475.86	3.66	260	109	–	6	0.38	0.38	139	8855.941	-2812
13	WBJV01	D2	New	UG2	Pass	468.68	1.93	505	180	–	1	0.69	0.69	133	8855.941	-2812
14	WBJV02	D0	New	UG2	Pass	557.62	1.70	2029	728	–	13	2.77	2.77	471	8573.092	-2812

15	WBJV02	D1	New	UG2	Pass	550.00	1.00	2123	747	-	10	2.88	2.88	288	8573.092	-2812
16	WBJV02	D2	New	UG2	Pass	554.10	1.19	2224	730	-	3	2.96	2.96	352	8573.092	-2812
17	WBJV03	D0	New	UG2	Pass	537.68	0.84	3167	1067	-	26	4.26	4.26	358	9215.668	-2812
18	WBJV03	D1	New	UG2	Pass	538.59	1.48	1973	1165	-	30	3.17	3.17	469	9215.668	-2812
19	WBJV03	D2	New	UG2	Pass	557.32	0.64	587	170	-	1	0.76	0.76	49	9215.668	-2812
20	WBJV05	D0	New	UG2	Pass	486.03	2.14	507	159	-	0	0.67	0.67	143	8309.380	-2812
21	WBJV06	D0	New	UG2	Pass	477.84	0.62	473	50	-	0	0.52	0.52	32	8608.128	-2813
22	WBJV07	D0	New	UG2	Pass	256.78	0.92	2514	728	253	28	3.52	3.52	324	8322.021	-2813
23	WBJV08	D0	New	UG2	Pass	352.31	0.29	800	469	-	10	1.28	1.28	37	8072.823	-2813
24	WBJV08	D1	New	UG2	Pass	323.86	1.15	1545	643	192	15	2.40	2.40	276	8072.823	-2813
25	WBJV09	D0	New	UG2	Pass	280.36	0.43	360	111	68	1	0.54	0.54	23	5733.700	-2811
26	WBJV09	D3	New	UG2	Pass	281.05	0.91	522	198	92	4	0.82	0.81	74	5733.700	-2811
27	WBJV10	D0	New	UG2	Pass	457.19	1.48	463	126	70	5	0.66	0.66	98	9358.783	-2813
28	WBJV10	D1	New	UG2	Pass	456.46	1.96	495	247	108	13	0.86	0.86	169	9358.783	-2813
29	WBJV12	D0	New	UG2	Pass	70.71	0.61	340	143	0	2	0.49	0.18	11	7999.966	-2814
30	WBJV13	D0	New	UG2	Pass	475.20	4.91	377	133	94	2	0.61	0.61	300	9160.170	-2813
31	WBJV13	D1	New	UG2	Pass	471.21	1.10	258	68	64	7	0.40	0.40	44	9160.170	-2813
32	WBJV14	D0	New	UG2	Pass	248.36	0.78	389	111	96	1	0.60	0.60	47	8511.950	-2813
33	WBJV14	D1	New	UG2	Pass	248.17	0.26	409	106	80	19	0.61	0.61	16	8511.950	-2813
34	WBJV15	D0	New	UG2	Pass	435.24	1.27	2636	1115	349	36	4.14	4.14	526	9320.907	-2813
35	WBJV15	D1	New	UG2	Pass	438.31	1.18	2976	979	360	17	4.33	4.33	511	9320.907	-2813
36	WBJV16	D0	New	UG2	Pass	134.41	0.96	3403	1256	430	41	5.13	5.13	492	7768.690	-2813
37	WBJV16	D1	New	UG2	Pass	117.71	1.36	2190	568	294	17	3.07	3.07	418	7768.690	-2813
38	WBJV18	D0	New	UG2	Pass	245.61	2.08	2100	1041	307	24	3.47	3.47	722	8761.791	-2813
39	WBJV18	D1	New	UG2	Pass	246.72	1.08	1900	803	303	20	3.03	3.03	327	8761.791	-2813

Entry:	**Explanation**
	Coordinates in WGS84, Hartebeeshoek datum and UTM 35S
-	**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**
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

Elandsfontein Project Summary of Ore Flow Calculations

TABLE 7a

TABLE 8a

TABLE 8b

TABLE 9

PTM - Platinum Group Metals Ltd.

TABLE 10

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

TABLE 14

WBJV, PRELIMINARY ASSESSMENT, 22 JULY 2005, ITEM 29, ILLUSTRATIONS, V1

WBJV, PRELIMINARY ASSESSMENT, 22 JULY 2005, ITEM 29, ILLUSTRATIONS, V1